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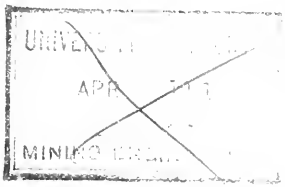

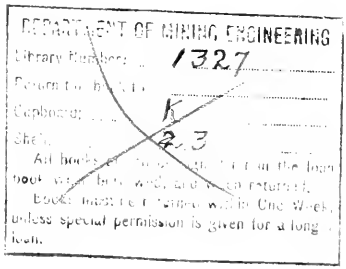

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THE JOURNAL

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NOTICES.

PAPERS AND DISCUSSIONS.—*Country and Foreign Members and Associates, unable to be present at the Meetings of the Society, are invited to send in Papers to be read, and also to contribute in writing to the various subjects under discussion.*

The Ordinary Monthly General Meeting will be held in the Lecture Theatre of the S. A. School of Mines and Technology, Eloff Street, Johannesburg, on Saturday, August 15th, 1914, at 7.45 p.m.

Owing to the great expense of reproducing diagrams, authors of papers are requested not to submit for publication in the Journal any other than those absolutely necessary to illustrate the text.

See also bye laws 5 and 7.

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THE CHEMICAL, METALLURGICAL & MINING

Society of South Africa.

BYE-LAWS.

(ADOPTED BY THE COUNCIL, AUGUST, 1909.)

1. Annual General Meetings will be held on the third Saturday of the month of June in each year.

2. Ordinary General Meetings will be held on the third Saturday in each month.

3. All meetings, unless otherwise provided for in the notice convening the same, shall be held in the Transvaal University College, Johannesburg, at 7.45 p.m.

4. Any Member or Associate may introduce a visitor to the Ordinary General Meetings; visitors may, with the consent of the Chairman take part in any discussion and read papers.

5. Every paper which it is desired to bring before the Society shall be clearly written on one side of the paper only, and shall be lodged with the Secretary at least one calendar month before the date of the meeting at which it is proposed to be read.

6. The Council shall decide what papers shall be read at meetings of the Society.

7. All papers brought before the Society shall be read, discussed and replied to within a period of four months, subject to the Council having power, should they deem it desirable, to extend that time.

8. Members and Associates whose subscriptions for the current year remain unpaid after the 1st day of October may be denied the privileges of the Society pending payment of the same; Members and Associates whose subscriptions for the current year remain unpaid after the 15th day of November may be removed from the roll of the Society.

9. Applicants for admission as Associates or Students shall forward the amount of the first year's subscription with the form of application. Should such application be not entertained, the amount remitted shall be returned to the applicant.

10. The Council may, on the report of a Sub-Committee appointed by them for the purpose, remit the subscription of any Member, Associate or Student for such period as they may deem desirable, if good and sufficient reason be advanced by any Member, Associate or Student.

11. The Minutes of all Special General Meetings shall be read and confirmed at the next convenient Council Meeting.

12. No statement shall be published in the Society's *Journal* which claims advantages for any novel method or device, unless such statement be accompanied by an adequate description of the method or device referred to.

PROCEEDINGS AND PERIODICALS.

Numerous technical periodicals and Proceedings of other Societies are received regularly and may be seen on arrival at the Society's Office every week between Thursday and Monday.

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Vol. XV. JULY, 1914. No. 1.

THE BLUE IRON-CYANOGEN COMPOUNDS.

By H. E. Williams (*Associate.*)

The addition of a solution of a ferric salt to a solution of a ferrocyanide results in the formation of a bulky deep blue precipitate, which varies in composition according to the conditions of the precipitation, and to the particular ferrocyanide used. The nature and constitution of these precipitates is still the subject of much discussion.

By the addition of a solution of hydroferrocyanic acid or calcium ferrocyanide to an excess of an acidified solution of ferric chloride, a blue precipitate is obtained which when washed and dried has a composition corresponding to the formula:—

$$Fe'''_4[Fe(CN)_6]_3 x H_2O$$

this compound is pure ferric ferrocyanide; the proportion of combined water in the compound varies with the temperature of the precipitation and the concentration of the solutions.

The precipitation of this compound should not take place in neutral solutions, otherwise some ferric hydroxide may be carried down with the blue.

If a solution of an alkali metal ferrocyanide is employed for the preparation of the blue, a precipitate is obtained which is always found to retain a certain proportion of the alkali metal which cannot be removed by the most careful washing with water; the amount of alkali metal retained by the precipitate varies with the particular alkali metal ferrocyanide used, and with the method of precipitation, but whatever method is employed it is impossible to obtain a blue by precipitating these ferrocyanides, quite free from the alkali metal.

Blue double salts of the alkali metal and ferric iron of the type $Fe'''M'Fe(CN)_6$ are known and the potassium or ammonium compounds may be prepared by the addition of a soluble ferric salt to a very large excess of the corresponding ferrocyanide, but the sodium salt of this series has not yet been prepared.

In blues prepared under other conditions where the proportion of the alkali metal retained falls between the limits of the formula $Fe'''M'Fe(CN)_6$ and $Fe_4[Fe(CN)_6]_3$, it is generally assumed either that the alkali metal is not in combination with the blue but is merely retained by the colloidal nature of the precipitate or that it is a mere mechanical mixture of the double salt $Fe'''M'Fe(CN)_6$ and pure ferric ferrocyanide.

To elucidate this point a large number of blue compounds have been prepared by the author from the various alkali metal ferrocyanides and very carefully washed and analysed. In the majority of cases the alkali metal was found to be present in a definite ratio to the ferrocyanogen, the analytical figures being in close agreement with those calculated from the formula. Also that, when prepared in duplicate, compounds were obtained of exactly the same composition and in many cases blues were obtained of the same composition by different methods.

The freshly prepared blue precipitates were washed a number of times by decantation with distilled water, then filtered through a large filter paper and allowed to drain, the blue paste thus obtained placed in an even layer on a Buchner vacuum filter, the pump started and the blue washed by displacement with distilled water until the filtrate was entirely free from soluble salts.

The washed blue was then carefully air dried, ground to a fine powder, washed several times with distilled water, drained on the vacuum filter and again air dried.

By these means a number of blues were obtained, the examination and analysis of

which leave no doubt that a number of blue compounds of constant composition may be prepared.

Of these compounds those prepared from ammonium and potassium ferrocyanide show great similarity in composition and appearance, whereas the blues prepared from sodium ferrocyanide usually contain a smaller proportion of alkali metal in combination, and sometimes a large proportion of combined water, and the air dried compound is usually less dense and of a lighter shade of colour approaching ultramarine, but almost devoid of bronze lustre.

These blue compounds may be divided into a number of types, and below is given a list of these blue compounds together with a short account of the method of preparation.

Compounds of the type $Fe'''_2M_6[Fe(CN)_6]_4$

If a ammoniacal solution of ferric tartrate (or citrate) containing a considerable amount of ammonium tartrate (or citrate) and free ammonia is added to a large excess of ammonium ferrocyanide solution, a dark brown liquid is obtained; if now a mineral acid, such as hydrochloric or dilute sulphuric acid, is gradually added in sufficient proportion to neutralise the free ammonia, and maintain an acidity due to the organic acid, a very finely divided green precipitate is obtained, which must first be washed by decantation a number of times and then filtered and the washing completed.

This compound when carefully washed and air dried forms a fine green powder which has a composition agreeing with the formula $Fe'''_2(NH_4)_6[Fe(CN)_6]_4 27H_2O$ and is the true double ferric ammonium ferrocyanide, in which exactly half the basic iron of ferric ferrocyanide is replaced by ammonium

$$2Fe'''_2(NH_4)_6[Fe(CN)_6]_3 + Fe'''_4[Fe(CN)_6]_3 = 3(NH_4)_3Fe(CN)_6$$

It belongs to the same series of compounds as the ferrous ammonium or ferrous potassium ferrocyanide $Fe''M'_2Fe(CN)_6$ or the well-known double salts such as

$Ca(NH_4)_2Fe(CN)_6$ and $Cu(NH_4)_2Fe(CN)_6$

If to the freshly precipitated compound prepared as described above, but without washing, a further quantity of mineral acid is added until the solution contains an excess of free mineral acid as well as the organic acid, the green compound changes to blue; this product, which may be washed more easily than the green compound, has when purified and dried a composition agreeing with the formula

$$Fe'''_8(NH_4)_{12}[Fe(CN)_6]_9 52H_2O$$

which may be considered as a molecular compound of two molecules of the green compound, with one molecule of ferric ferrocyanide.

$$Fe'''_{10}(NH_4)_{12}[Fe(CN_6]_{12} = 2Fe'''_2(NH_4)_6[Fe(CN)_6]_3 + Fe'''_4[Fe(CN)_6]_3$$

If in the preparation of the green ammonium salt above described, potassium salts are substituted for the ammonium compounds a green potassium compound is obtained when the alkaline solution is acidified with an organic acid, but, on washing, the colour of the compound changes to a dull blue which, when thoroughly washed and air dried, has the composition:

$$Fe'''_{10}K_{12}[Fe(CN)_6]_{12} 45H_2O$$

$$(= 3Fe'''_2K_6[Fe(CN)_6]_3 + Fe'''_4[Fe(CN)_6]_3)$$

When the precipitation takes place in the presence of free mineral acid a blue is obtained corresponding to the ammonium compound prepared under similar circumstances:—

$$Fe'''_8K_{12}[Fe(CN)_6]_9$$

$$(= 2Fe'''_2K_6[Fe(CN)_6]_3 + Fe'''_4[Fe(CN)_6]_3$$

Attempts to prepare similar sodium compounds did not succeed.

Compounds of the type:—

$$Fe'''_7M'_3[Fe(CN)_6]_6 (= Fe'''_4[Fe(CN)_6]_3 : 3Fe'''MFe(CN)_6$$

When dilute ammonium ferrocyanide is precipitated by an excess of a dilute solution of ferric chloride, a blue is obtained which, when washed and dried, has a composition agreeing with the formula:— $Fe'''_7(NH_4)_3[Fe(CN)_6]_6 50H_2O$ which, when heated to 100° C., loses half its combined water. A compound of exactly the same percentage composition may be obtained by adding ammonium ferrocyanide solution to a boiling excess of ferric chloride. If in the preparation of this compound by the latter method a considerable excess of potassium chloride is dissolved in the boiling ferric chloride solution, before the addition of the ferrocyanide, a blue precipitate is obtained of the same type, but with one equivalent of the ammonium replaced by potassium thus:—

$$Fe'''_7(NH_4)_2K[Fe(CN)_6]_6 50H_2O$$

which loses 26 equivalents of water at 100°C. This compound is also formed when ferric chloride solution is added in slight excess to a solution of potassium ferroeyanide, containing ammonium chloride.

The sodium compound of this series:—

$$Fe'''_7Na_3[Fe(CN)_6]_6 50H_2O$$

may be prepared by oxidising white ferrous sodium ferrocyanide by an excess of ferric

chloride solution, the former body being prepared by exactly precipitating sodium ferrocyanide solution with ferrous chloride, these two reactions may be represented thus:—

1. $Na_4Fe(CN)_6 + Fe''Cl_2 = Fe''Na_2Fe(CN)_6 + 2NaCl$

2. $6Fe''Na_2Fe(CN)_6 + 7Fe'''Cl_3 = Fe'''_7Na_3[Fe(CN)_6]_6 + 9NaCl + 6Fe''Cl_2$

A blue having the same proportion of alkali metal, but with less combined water, may be obtained by treating with an excess of ferric chloride solution, the still boiling solution of ferrous sodium ferrocyanide got by distilling sodium ferrocyanide solution with a dilute mineral acid. The compound obtained under these conditions has the composition

$$Fe'''_7Na_3[Fe(CN)_6]_6 24H_2O.$$

Compounds of the type

$Fe'''_{11}M_3[Fe(CN)_6]_9 = (2Fe'''_4[Fe(CN)_6]_3 + 3Fe''MFe(CN)_6$

The potassium salt of this series

$$Fe'''_{11}K_3[Fe(CN)_6]_9 100H_2O$$

is obtained by precipitating potassium ferrocyanide solution in the cold with a slight but permanent excess of a ferric salt; when heated to 100°C 33 equivalents of water are driven off: the same compound is obtained when a large excess of the iron salt is added to the ferrocyanide solution if an equivalent quantity of potassium chloride is previously added: and it may also be prepared by agitating freshly prepared ferrous potassium ferroeyanide with an excess of ferric chloride solution, but the blue obtained by the latter method on air drying, retains only 81 equivalents of water instead of 100.

The sodium salt

$$Fe'''_{11}Na_3[Fe(CN)_6]_9 90H_2O$$

may be obtained by oxidising ferrous sodium ferroeyanide with dilute chromic acid in the cold.

If potassium ferroeyanide solution is added to a boiling excess of a solution of ferric chloride, a blue is obtained containing much less combined potassium than these described above and having a composition agreeing with the formula:—

$Fe'''_9K[Fe(CN)_6]_7 78H_2O = (2Fe'''_4[Fe(CN)_6]_3 + Fe'''KFe(CN)_6)$

At 100° C. this compound retains 49 molecules of water.

When sodium ferrocyanide solution is added to a boiling excess of ferric chloride solution, a blue precipitate is obtained containing a very small proportion of alkali metal in combination, the composition of the compound agreeing with the formula:—

$Fe'''_{13}Na[Fe(CN)_6]_{10} 100H_2O = (3Fe'''_4[Fe(CN)_6]_3 + Fe'''NaFe(CN)_6$

A blue precipitate having the same composition may be obtained by precipitating sodium ferroeyanide solution with an excess of ferric chloride solution.

If twice the equivalent quantity of ammonium chloride or sulphate is dissolved in a dilute solution of sodium ferrocyanide, and the whole precipitated in the cold with a large excess of ferric chloride, a blue is obtained which has a composition agreeing with the apparently complex formula:—

$$Fe'''_{22}(NH_4)_6Na[Fe(CN)_6]_{10} 146H_2O$$

If we eliminate from the above formula the simplest portion of the molecule containing the sodium thus $Fe'''NaFe(CN)_6$ there is left the residue

$Fe'''_{21}(NH_4)_6[Fe(CN)_6]_{18} = 3Fe'''_7(NH_4)_3[Fe(CN)_6]_6 = 3[Fe'''_4[Fe(CN)_6]_3 + 3Fe'''NH_4Fe(CN)_6]$

agreeing exactly with the composition of the blue obtained when sodium free ammonium ferrocyanide is precipitated under similar conditions, but as the compound is precipitated in the presence of sodium salts, the $Fe'''_4[Fe(CN)_6]_3$ portion of the molecule is converted into $Fe'''_{13}Na[Fe(CN)_6]_{10}$ the compound obtained from sodium ferrocyanide in the absence of ammonium salts:—

$Fe'''_{22}(NH_4)_6Na[Fe(CN)_6]_{19}$
$9Fe'''NH_4Fe(CN)_6 + 3Fe'''_4[Fe(CN)_6]_3 + Fe'''NaFe(CN)_6$ or
$Fe'''_{22}(NH_4)_9Na[Fe(CN)_6]_{19} = 9Fe'''NH_4Fe(CN)_6 + Fe'''_{13}Na[Fe(CN)_6]_{10}$
or $Fe'''_{22}(NH_4)_6Na[Fe(CN)_6]_{19}$
$3Fe'''_7(NH_4)_3[Fe(CN)_6]_6 + Fe'''NaFe(CN)_6$

If freshly precipitated and washed cupric ferrocyanide is boiled for some hours with an excess of ferric chloride solution, the brown colour of the cupric salt changes to blue, and a compound is obtained which is a ferric cupric ferrocyanide of the formula

$$Fe'''_2Cu''[Fe(CN)_6]_2 28H_2O$$

as a rather light blue powder.

The blue compounds above described have been expressed as molecular compounds of one or more molecules of $Fe'''_4[Fe(CN)_6]_3$ with one or more molecules of the double salt $Fe''MFe(CN)_6$ general formula:—

$$xFe'''_4[Fe(CN)_6]_3 + yFe''MFe(CN)_6$$

A much more simple relationship between these complex compounds is seen if they are considered as molecular compounds of pure ferric ferrocyanide and the true double salt of the type:—

$$Fe'''_2M_6[Fe(CN)_6]_3$$

This relationship is seen very clearly when the formula of the above described compounds or a multiple of that formula is set out in the following manner:—

Formula.	Ratio of $Fe^{III}_4[Fe(CN)_6]_3$: $Fe^{III}M'Fe(CN)_6$
1. $Fe^{III}_2(NH_4)_6[Fe(CN)_6]_3$	
2. $Fe^{III}_{10}K_{18}[Fe(CN)_6]_{12}$ = $3Fe^{III}_2K_6[Fe(CN)_6]_3$: $Fe^{III}_4[Fe(CN)_6]_3$	3 : 1
3. $Fe^{III}_4(NH_4)_{12}[Fe(CN)_6]_6$ = $2Fe^{III}_2(NH_4)_6[Fe(CN)_6]_3$: $Fe^{III}_4[Fe(CN)_6]_3$	2 : 1
4. $Fe^{III}KFe(CN)_6 \times 6$ $Fe^{III}_2K_6[Fe(CN)_6]_3$: $Fe^{III}_4[Fe(CN)_6]_3$	1 : 1
5. $Fe^{III}_4M_3[Fe(CN)_6]_6 \times 2$ = $Fe^{III}_2M_6[Fe(CN)_6]_3$: $3Fe^{III}_4[Fe(CN)_6]_3$	1 : 3
6. $Fe^{III}_{11}M_3[Fe(CN)_6]_9 \times 2$ = $Fe^{III}_2M_6[Fe(CN)_6]_3$: $5Fe^{III}_4[Fe(CN)_6]_3$	1 : 5
7. $Fe^{III}_4K[Fe(CN)_6]_3 \times 6$ = $Fe^{III}_2K_6[Fe(CN)_6]_3$: $13Fe^{III}_4[Fe(CN)_6]_3$	1 : 13
8. $Fe^{III}_{13}Na[Fe(CN)_6]_{10} \times 6$ = $Fe^{III}_2Na_6[Fe(CN)_6]_3$: $19Fe^{III}_4[Fe(CN)_6]_3$	1 : 19

The ferric cupric ferrocyanide may be considered in a similar manner:

$Fe_2Cu[Fe(CN)_6]_2 \times 3$
$Fe^{III}_2Cu''_3[Fe(CN)_6]_4$ =
$Fe_4[Fe(CN)_6]_3$ ... ... 1 : 1

the very stable blue and green ferrosoferric ferrocyanides described by the author elsewhere* (Proceedings, Chemical Society, 29, 411) are undoubted molecular compounds of the double salt

$Fe^{III}_2M_4[Fe(CN)_6]_3$ with $Fe''M_2Fe(CN)_6$ or $Fe''_2Fe(CN)_6$

1. $Fe^{III}_2Fe''(NH_4)_2[Fe(CN)_6]_3$
$Fe^{III}_2(NH_4)_6[Fe(CN)_6]_3$:
$Fe''_3(NH_4)_2Fe(CN)_6$
2. $Fe^{III}_2Fe''K_2[Fe(CN)_6]_3$
$Fe^{III}_2K_6[Fe(CN)_6]_3$: $Fe''K_2Fe(CN)_6$
3. $Fe^{III}_2Fe''_2(NH_4)_6[Fe(CN)_6]_3$
$Fe^{III}_2(NH_4)_6[Fe(CN)_6]_3$: $Fe''_2Fe(CN)_6$
4. $Fe^{III}_2Fe''_2Na_6[Fe(CN)_6]_3$
$Fe^{III}_2Na_6[Fe(CN)_6]_3$: $Fe''_2Fe(CN)_6$
5. $Fe^{III}_2Fe''_2K_6[Fe(CN)_6]_3$
$Fe^{III}_2K_6[Fe(CN)_6]_3$: $Fe''_2Fe(CN)_6$

The latter compound may be prepared by adding some ferric chloride solution to a large excess of potassium ferrocyanide and boiling the mixture some time, some cyanogen is evolved and a dull blue precipitate obtained of a ferrosoferric potassium ferrocyanide of the above composition.

Analytical Details.

The analysis of the blue compounds was accomplished in the following manner.

Ferrocyanogen.—Half a gram of the purified and air dried compound was carefully weighed out into a beaker and treated with 50 cc. of a 5 % solution of potassium hydroxide and digested in a warm place until complete decomposition of the blue took place, then filtered and the precipitated ferric hydroxide thoroughly washed; the clear filtrate was then transferred to a 250 cc. distilling flask, acidified with an excess of dilute sulphuric acid, a small quantity of cuprous chloride added, and the whole distilled through a condenser into a dilute sodium or potassium hydroxide solution in an absorption flask, and the cyanide solution so obtained titrated with *N* 10 silver nitrate solution after the addition of a few drops of potassium iodide solution.—(H. E. Williams, *Journal, Soc. Chem. Industry*, 31st May, 1912.)

Basic Iron.—The precipitated and washed ferric hydroxide left on the filter paper was dissolved in dilute hydrochloric acid and either reprecipitated by ammonia solution and estimated as ferric oxide; or the acid solution was reduced, and the iron estimated by permanganate.

Alkali Metal.—One gram of the dried and ground blue was digested with distilled water and filtered, and the filtrate evaporated to dryness for the detection of incomplete washing; in the absence of any residue in the evaporated filtrate, the blue on the filter paper was ignited in an iron crucible in contact with the air, the ash was turned out into a beaker, dissolved in hydrochloric acid, the iron precipitated as hydroxide by ammonia and the filtrate evaporated to dryness in a platinum dish on the water bath. To prevent spirting on ignition due to the excess of ammonium salts, it was found advisable to add an excess of sulphuric acid and then to strongly ignite under a hood, the residue of sulphate of potassium or sodium being then carefully weighed and checked by solution in water and precipitated as barium sulphate.

As the alkali metal in each estimation was either all potassium or all sodium, the need of separating these two alkali metals did not arise.

* See also this *Journal*, Vol. XIII., June, 1913, p. 619.

Ammonium.—One gram of the sample after washing with water and an examination of the filtrate for the detection of free ammonium salts was placed in a distilling flask and distilled with dilute caustic alkali solution, the vapour containing the ammonia being conducted in a solution of standard acid and estimated in the usual manner.

In this manner the following analytical figures were obtained for the compounds described:—

Compound.		Fe (Basic)	Alkali Metal.	$^{\prime\prime\prime}Fe(CN)_6$	H_2O.
1. $Fe_2(NH_4)_6[Fe(CN)_6]_3 27H_2O$	Found	8·36	7·99(NH_4)	47·25	36·30
„ „ „	Calculated	8·32	8·07(NH_4)	47·38	36·22
2. $Fe_8(NH_4)_{12}[Fe(CN)_6]_9 52H_2O$	Found	12·57	6·12(NH_4)	54·40	26·91
„ „ „	Calculated	12·73	6·17(NH_4)	54·35	26·74
3. $Fe_{10}K_{18}[Fe(CN)_6]_{12} 45H_2O$	Found	12·34	15·4(K)	54·80	—
„ „ „	Calculated	12·12	15·24(K)	55·09	17·54
4. $Fe_7Na_2[Fe(CN)_6]_6 50H_2O$	Found	14·91	2·62(Na)	48·22	—
„ „ „	Calculated	14·81	2·62(Na)	48·19	34·37
5. $Fe_7(NH_4)_2[Fe(CN)_6]_6 50H_2O$	Found	14·97	1·98(NH_4)	48·07	—
„ „ „	Calculated	14·96	2·06(NH_4)	48·46	34·57
6. $Fe_7(NH_4)_2K[Fe(CN)_6]_6 50H_2O$	Found	14·94	1·39(NH_4) + 1·5(K)	47·91	—
„ „	Calculated	14·78	1·36(NH_4) + + 1·48(K)	48·07	34·31
7. $Fe_{11}K_3[Fe(CN)_6]_9 100H_2O$	Found	—	2·64(K)	42·70	—
„ „ „	Calculated	13·83	2·61(K)	42·97	40·55
8. $Fe_{11}Na_3[Fe(CN)_6]_9 90H_2O$	Found	14·66	1·60(Na)	45·06	—
„ „ „	Calculated	14·61	1·64(Na)	45·29	38·46
9. $Fe_9K[Fe(CN)_6]_7 78H_2O$	Found	14·96	1·10(K)	43·08	—
„ „ „	Calculated	14·61	1·13(K)	43·12	41·13
10. $Fe_{13}Na[Fe(CN)_6]_{10} 100H_2O$	Found	15·74	0·50(Na)	45·50	—
„ „ „	Calculated	15·55	0·49(Na)	45·39	38·55
11. $Fe_{22}(NH_4)_9Na[Fe(CN)_6]_{19} 146H_2O$	Found	15·66	1·98(NH_4) + 0·29(Na)	50·76	—
„ „	Calculated	15·43	2·04(NH_4) + 0·28(Na)	50·58	31·65
12. $Fe_2Cu[Fe(CN)_6]_2 28H_2O$	Found	10·01	5·66(Cu)	38·18	—
„ „	Calculated	10·11	5·75(Cu)	38·37	15·76

Contributions and Correspondence.

MEYER & CHARLTON CYANIDE SOLUTION.

With reference to Mr. H. E. Williams' note on the analysis of the above, I should like to remark that the value of the figures would be greatly enhanced if the author would favour us with a sketch of his methods, particularly as to what assumptions he made in doing the calculations. Why, for example, does free HCN not appear in the analysis, when it can be smelt in every cyanide solution except when the caustic alkali is very high, say 1%? How was sodium cyanate determined? If cyanate and ammonia were present, would not urea have been detectable? Is it not possible that the "free ammonia" is wholly or partly methylamine (from action of nascent hydrogen in zinc-boxes on cyanide)? Why does Mr. Williams think that the $AgNO_3$ titration does not give the correct "free cyanide?" I personally think the $AgNO_3$ figure is correct, and that Mr. Williams' figure (apparently obtained by difference) is quite wrong, since he assumes that none of the complex salts yields "free cyanide" by electrolytic dissociation, whereas, in fact, they nearly all do so to a slight extent, and the zinc compound to a great extent.

JAMES MOIR.

* See this *Journal*, Vol. XIV., 1914, pp. 475-476.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Determination of Carbon Monoxide in Air.—"The air, freed from carbon dioxide, unsaturated hydrocarbons and moisture, was passed through two vertical tubes packed with alternate layers of iodine pentoxide and glass wool and heated in a paraffin bath to 140—160° C., and then bubbled through mercury and collected in a special apparatus, consisting essentially of a dry gas vessel connected with a long vertical tube dipping into mercury. The gas vessel was exhausted, connected with the vertical tube, and the gas allowed to flow in at the rate of 1 litre per hour, the volume being calculated from the fall of the mercury. The carbon dioxide formed from the carbon monoxide by the oxodising action of the iodine pentoxide could be accurately titrated in the gas vessel by Pettenkofer's process. The analyses were checked by using red hot copper oxide in place of iodine pentoxide."—F. S. Sinnatt and B. J. Cramer, *Analyst*, 1914, *xl*, 163-170. *Journal of the Society of Chemical Industry*, May 15th, 1914, p. 499. (J. G.)

The use of Picric Acid in Standardising Solutions, and the Iodometric Determination of Acids.—"Picric acid is recommended as a standard substance for iodometry. It is not hygroscopic and gives a very sharp end-point without the addition of an indicator. The following equations illustrate the interactions involved :—

$$6C_6H_2(NO_2)_3OH + (or\ 6HCl,\ etc.) + 5KI = KIO + 6C_6H_2(NO_2)_3OK + (or\ 6KCl,\ etc.) + 3I + 3H_2O$$

Iodine is estimated by thiosulphate solution standardised by the picric acid (*N*/25 or *N*/100). The iodide-iodate solution should contain 40 grm. of potassium iodide and 10 grm. of iodate per litre. It is very stable, but becomes yellow on prolonged exposure to light ; sodium thiosulphate is then added until the colour disappears. Many coloured solutions of acids—and bases may be analysed iodometrically it starch solution be added towards the end of the titration of the iodine. A. Sander, *Z. Angew. Chem.*, 1914, *27*, 192-194. *Journal Society Chemical Industry*, April 30, 1914, pp. 411-412. (J. G.)

Assay of Zinc by Ferrocyanide Methods. "Sufficient of the ore to require about 25 cc. of the ferrocyanide solution (32·5 grm. per litre) is decomposed with nitric acid, potassium chlorate added and the mixture evaporated just to dryness ; oxidised ores are dissolved in hydrochloric acid, evaporated to dryness, and boiled with nitric acid before the addition of chlorate. The residue is treated with 100 cc. of boiling water, 7 grm. of ammonium chloride, 15 cc. of ammonia (sp. gr. 0·880) and 0·5 grm. of sodium peroxide, the liquid boiled for 5 minutes, filtered, the precipitate washed twice with a boiling 1% solution of ammonium chloride, re dissolved in 5 cc. of hydrochloric acid, re precipitated by addition of 3 grm. of ammonium chloride, 15 cc. of ammonia and 2—5 grm. of sodium peroxide, and washed three times with the ammonium chloride solution. *Titration in alkaline solution.* The combined ammoniacal filtrates (about 300 cc.) are heated to 70° C., 30 cc. of tartrate emulsion (200 grm. of cream of tartar, 2 cc. of neutral ferric chloride solution containing 10 % Fe, and 1 litre of water) added, and the solution titrated with ferrocyanide until a faint blue colour is obtained by adding a drop of the liquid to a drop of glacial acetic acid, with 0·2 % or more copper in the ore, the ammoniacal solution is acidified with hydrochloric acid and 1 cc. excess added, the liquid boiled with 20—30 grm. of pure granulated lead, decanted, and titrated after addition of 30 cc. of tartrate emulsion and 5 cc. of ammonia. With very low grade ores the excess of ammonia may prevent precipitation of the zinc by ferrocyanide, hence acetic acid is added until a precipitate commences to form, before the titration ; but the solution must not become acid. *Titration in acid solution* : To the combined ammoniacal filtrates, whether copper be present or not, 3 grm. of sodium thiosulphate and 5 cc. of hydrochloric acid in excess of that required to make the solution just acid to methyl orange are added, the solution boiled to expel most of the sulphur dioxide and coagulate the precipitated copper sulphide and sulphur, cooled to 70° C. and titrated (without removing the precipitate) using uranium acetate as external indicator as usual. In either method of titration the ferrocyanide solution is standardised against chemically pure zinc under similar conditions to those of the assay, a blank test being made to determine the volume of ferrocyanide required to produce the end-point in the absence of zinc. Results with the acid method on Broken Hill ores or concentrates, are accurate ; results for a concentrate containing 50% Zn should agree to within 0·5%. The alkaline method is preferred by the Sub-Committee as although the end-point is less sharp the process is more rapid than the acid method and equally accurate." R. T. D. Williams, *Mining and Engineering World*, 1914, *40*, 55-57. From Report of Assay Sub-Committee of Australasian Inst. of Min. Eng. Determination of Zinc by Ferrocyanide in Acid Solutions. Trans. *Australasian Inst. Min. Eng.*, 1914, *17*, 629-644. *Journal of the Society of Chemical Industry*, March 31, 1914, p. 316. (J. G.)

Determination of Free Caustic Alkali in Commercial Soda Compounds and Soaps by the Barium Chloride Method. "Precipitated barium silicate or borate is soluble in dilute aqueous solutions of barium chloride with the result that the method of determining free caustic alkali in soda compounds or in soaps which contain soluble silicates or borates, by precipitation of their aqueous solutions with barium chloride and subsequent titration of the clear liquid with *N*/10 acid gives inaccurate results. These salts, however, are insoluble in 50% alcohol and if, after the precipitation, a sufficient quantity of strong alcohol is added to the liquid to bring the alcohol concentration to 50%, accurate results are obtained except in the case of soaps containing oleic acid, since barium oleate is slightly more soluble in 50 % alcohol than in water."—A. Kling, V. Genin and D. Florentin, *Bull. Soc. Chim*, 1914, *15*, 200-205. *Journal of the Society of Chemical Industry*, March 31, 1914, p. 324. (J. G.)

Detection and Determination of Nitrites by Means of Diphenylbenzidine. "The authors confirm Wieland (*B. c.*, 1913, *46*, 3297) that in the diphenylamine reaction for the detection of nitrites and other oxidising agents the first step in the production of the blue colour is the formation of diphenylbenzidine, which is then converted by the oxidising agent into a quinonoid derivative, the

principal constituent of the blue solution. Diphenylbenzidine is hence a much more delicate reagent for nitrites than diphenylamine. It may be prepared by adding a solution of 8 grm. of diphenylamine in 20 cc. of acetic acid to 50 cc. of sulphuric acid (sp. gr. 1·842) previously diluted with 150 cc. of water and cooled. To this mixture a solution of 4·7 grm. of sodium bichromate in 200 cc. of water and 20 cc. of sulphuric acid (1 : 4) is added, constant stirring being maintained for about five minutes. The liquid is added to 100 cc. of sodium bisulphite solution (sp. gr. 1·231) and 400 cc. of water and the mixture heated on a water-bath until the dark green precipitate first formed changes to a light brown. After cooling, the precipitate is filtered off, washed with water and alcohol, dried, and the diphenylbenzidine extracted with boiling toluene. A yield of about 60% of the theoretical is obtained."—Marqueyrol and H. Muraour, *Bull. Soc. Chim.*, 1914, *15*, 186-197.—*Journal of the Society of Chemical Industry*, March 31, 1914, p. 335. (J. G.)

A Delicate Colorimetric Method for Detecting and Estimating Nitrates and Nitrites.—"The authors have used diphenylbenzidine (Kehrmann and Micewitz, *Ber.*, 1912, *45*, 2641 ; Wieland, *Ber.*, 1913, *46*, 3,300 ; compare also *Bull. Soc. chim.*, 1914, [iv], *15*, 186), for detecting and estimating small quantities of nitrates, the test being quite sensitive up to 0·00005 milligram of nitric nitrogen, the depth of blue colour depending on : (1) the temperature, the colour being deeper at higher than at ordinary temperatures ; (2) the time that elapses after mixing the nitrate solution with sulphuric acid and diphenylbenzidine ; and (3) the amount of nitrate present.

The estimation of nitrates by the new method depends on the depth of the blue tint, and is carried out in small porcelain crucibles. The method may also be employed for estimating both nitrites and nitrates in a mixture of the two.—Edmund Albert Letts and Florence Williamson Rea, *Proc. Chem. Soc.*, Vol. 30, No. 426, p. 72. (C. T.)

Molybdenum in Cyanide Solutions.—"The most satisfactory test for molybdenum in cyanide solution is the sulphocyanide reaction which is delicate enough to be applied directly to the solution. The test is conducted as follows : Take about 50 cc. of cyanide solution, add a drop of ammonia (most cyanide solutions contain sufficient ammonia and generally this addition is unnecessary), add a crystal of potassium sulphocyanide, then render acid with sulphuric acid, and add a strip of zinc. In the course of a minute, a beautiful carmine-red coloration is produced, which gradually changes to brown, and eventually a dirty green. This red coloration is different from the ferric sulphocyanide colour, and moreover is produced under reducing conditions where the ferric sulphocyanide coloration cannot exist.

Satisfactory quantitative estimations have been obtained by colorimetric analysis in comparing the thoroughly reduced cyanide solution with a correspondingly reduced solution of molybdic acid. The general colour of such solutions is dirty green by reflected light, and brown by transmitted light.

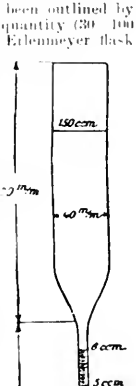
I appreciate Mr. Clennell's difficulties with identifications, as the cyanide chemist must ever bear in mind that he is dealing with solutions that bristle with organic derivatives : many of the metals, notably copper and mercury, have lost their ionic identity in combinations with such derivatives and hence our common inorganic tests do not apply. Or if the metals do respond to some tests, they fail in others, and one is led into great confusion. This undoubtedly explains why, in general, relatively small amounts of copper and mercury are precipitated in comparison with the amounts in solution. The Liberty Bell solutions carry even a greater amount of copper and mercury than precious metals, yet after passing through the zinc boxes where precipitation is so perfect that the tails contain only 1 oz. per ton in gold and silver, the copper, as well as the mercury, is but slightly diminished.

Mr. Clennell's investigation regarding the effect of molybdic acid upon cyanide is indeed timely. It is time that we were moulding into form some terse and definite ideas regarding acidity, humus acids, organic matter, reducing agents, cyanicides, foulness, etc., which for vagueness of meaning constitute a most remarkable collection of indefinite terms."—A. J. Weinig, *Engineering and Mining Journal*, April 11, 1914, p. 773. (H. A. W.)

Test for Nitrates.—"A rough test which serves to identify ½% of sodium nitrate in a mineral is described by Surr in the *Mining and Engineering World*. The suspected material is ground to 40 mesh, and a teaspoonful of it is mixed with rather less than half a teaspoonful of flour. In the absence of nitrates the mixture chars quietly, but if ½% or more of sodium nitrate is present bright sparks and eruptions are caused."—Surr, *Mining and Engineering World.—The Mining and Engineering Review*, March 5th, 1914, p. 174. (J. E. T.)

New Method of Analysis of Nitrates, Chlorates and Peroxides.—"The method of titrating ferric salts with stannous chloride in presence of methylene blue as indicator has been applied to the analysis of nitrates, chlorates and manganese dioxide. For the analysis of nitrates a flask of about 500 cc. capacity is used, provided with inlet and outlet tubes for a current of carbon dioxide ; the outlet tube passes to the bottom of a second flask of 300 cc. capacity containing 120 cc. of water to which a solution of decolourised methylene blue (2—3 cc. of water, 2 drops of a 10% solution of methylene blue, and 1 drop of *N*/5 stannous chloride, heated to boiling) has been added. After all the air in the large flask has been expelled by carbon dioxide, 3 grm. of ferrous ammonium sulphate and about 0·2 grm. of the nitrate are introduced, followed by 10 cc. of concentrated hydrochloric acid. The mixture is heated gently at first and finally to boiling and the water in the smaller flask is also heated to boiling When the reaction is completed, the flames are removed and the current of carbon dioxide is cut off : on cooling, the boiled water is sucked over from the smaller into the larger flask, and the solution in the latter is then titrated with stannous chloride. In the case of chlorates access of air has practically no effect on the results. The ferrous salt is added to the boiling solution of the chlorate, followed by 10 cc. of concentrated hydrochloric acid, added gradually. When the reaction is completed, the solution is titrated with stannous chloride. For the analysis of manganese dioxide a weighed quantity is suspended in 100 cc. of boiled water, then the ferrous salt and hydrochloric acid are added, the whole is boiled until a clear solution is obtained, and this is titrated with stannous chloride."—C. Russo and G. Sensi, *Gaz. chim. ital.*, 1914, *44*, I., 9-15 ; *Jour. Soc. of Chemical Industry*, March 16, 1914, p. 279. (J. G.)

DETERMINATION OF WATER IN SOLID AND LIQUID FUELS BY DISTILLATION WITH XYLENE.—"The author describes in greater detail his method of determining water which has been outlined by Huntley and Coste. A suitable quantity (30–100 gm.) of the fuel is brought into an Erlenmeyer flask of 500 cc. capacity with 200 cc. of xylene. The flask is connected by means of a tube, 0·5 to 1 cm. dia., with a vertical condenser which leads to the measuring vessel, shown in the figure. The distillation is conducted slowly whilst the first 30 cc. are being collected, and then more rapidly, the total 150 cc. being obtained in about 30 min. For larger quantities of water a tube of 12 mm. dia. in its narrowest part is used, the other dimensions being approximately the same. The cylinder is carefully cleaned by steam and chromic acid after each determination. The volume of water observed must be corrected for errors in the graduation of the tube, for change in the shape of the menisci by the pressure of the xylene (about 0·02 cc. for the narrow tube) and finally for water lost in the distillation which depends on the quantity present and the form of the apparatus. When once the last correction has been determined under known conditions, it is possible to make accurate corrections and to obtain results which are very reliable, except in the case of certain anthracites. For material such as briquettes with a volatile binding agent this appears to be the only reliable method of determining moisture, and it is also very convenient and rapid compared with other methods for liquids such as tar and viscous mineral oils.—P. SCHLAEFKE, *Z. angew. Chem.*, 1914, *27*, 52–56; *Journ. Soc. of Chemical Industry*, Feb. 28, 1914, p. 189. (J. G.)



DETERMINATION OF THE TITRATION VALUE OF ARSENITE SOLUTIONS.—"The use of volumetric solutions of iodine for the standardisation of arsenite solutions may lead to erroneous results owing to the presence of iodate, which in the standardisation of the iodine by thiosulphate in acid solution yields its total iodine content as free iodine, but, in the subsequent titration of the alkaline arsenite solution, is entirely unchanged. The measured volume of iodine solution should therefore always be previously acidified with hydrochloric acid to decompose iodate, before adding sodium bicarbonate and titrating with the arsenite solution. In preference to this method the iodometric standardisation with the help of permanganate is recommended. A measured volume of standard permanganate is run into acidified potassium iodide solution, and the liberated iodine, after the addition of bicarbonate, titrated with the arsenic solution in the usual way."—E. DITTE, *Chem. Zeit.*, 1914, *38*, 413–415.—*Journal Society Chemical Industry*, April 30, 1914, p. 412. (J. G.)

DETERMINATION OF HARDNESS OF WATERS BY THE METHODS OF WARTHA-PFEIFER AND BLACHER.—"In the Wartha-Pfeifer method the alkaline earths are precipitated from the hot neutral solution by a mixture of equal parts of sodium carbonate and hydroxide, and the excess of the reagent titrated back by means of acid. Lunge modified the method by acidifying the water, boiling, neutralising in presence of methyl orange, precipitating with the soda mixture, making up to a known volume, filtering and titrating back the unconverted alkali in an aliquot part of the filtrate. The authors have found in a long series of waters which they examined by the Wartha-Pfeifer method, that 92% of the results fell within a maximum error of 1·05 German degrees of hardness as compared with the gravimetric method, and in another series within 0·55 degree. They found also that Lunge's modification possessed more faults than the original and gave no better results. They conclude that the Wartha-Pfeifer method is suitable for exact work but not for a routine method."—J. ZINK and F. HOLLANDT, *Z. angew. Chem.*, 1914, *27*, 235–237.—*Journal of the Society of Chemical Industry*, May 30, 1914, p. 564. (J. G.)

METALLURGY.

UTILISATION OF HEAT FROM SLAGS.—"For many years past efforts have been made to utilise the heat contained in molten slags. Messrs. Bell Brothers and the Slag Power Co. have, after numerous trials, solved the problem.

The slag experimented with was ordinary Cleveland iron slag of the following average composition:

SiO_2	30·80
Al_2O_3	22·43
FeO	0·38
CaO	35·48
CaS	3·75
MgO	6·23
MnO	0·93

The temperature of the slag employed was about 1,500° C. On pouring this into water contained in a specially constructed vessel the slag granulates, whilst certain non-condensable gases are generated, having approximately the following composition:—

	Per cent.
H_2S	29·00
H	55·00
N	16·00

The amount of gas per volume of steam varies according to the quality of the slag, but on grey slag the average production may be taken as about 0·5%. Besides these gases a quantity of fine sulphur is produced. It was originally intended to utilise the steam generated from the slag in a low-pressure steam turbine for the production of power. It was found, however, that the presence of non-condensable gases referred to above rendered a high power vacuum impossible, and, although means were devised for successfully eliminating them, the sulphur difficulty still remained, and it was found impossible to completely separate it from the steam. Also when the sulphur was allowed to come in contact with the blades of a turbine, it caused their rapid destruction, and stopped up the steam entrances.

For these reasons the method of using the steam directly in the turbine was impracticable, but the difficulty has been overcome in the present system by employing a heat exchanger, the 'dirty' steam from the slag being used to generate clean steam, to supply the turbine.

The plant consists of the primary generator containing an upright revolving shaft with four blades, which keeps the water constantly rotating. With blast furnace slag a secondary generator is required, but not with copper slag.

Next to the generators is placed the calandria, a Kestner single-effect climbing film evaporator, consisting of two parts, namely, the calandria and the separator. The calandria is composed of a shell or casing containing the evaporation tubes. The tubes are about 23 ft. long and fixed into the upper and lower tube plates.

The water begins to boil in the tubes because they are surrounded by the steam, and as ebullition takes place a column of vapour rises up the centre of the tube.

This vapour travels at a high velocity, and at the same time draws up a film of water on the inner surface of the tubes.

Copper Slag.—By experiment it has been found that 1,000 kilos. of slag contain from 318—350 calories of available heat, and upwards of 500 kilos. of water have been evaporated per 1,000 kilos. of slag; Taking the lower figure, viz., 318 calories, then—

1,000 kilos. × 318 calories gives ...	318,000 calories	
From this must be deducted the heat taken away in the slag at 100° C.—		
1,000 kilos. × 100° C. × 0·18 specific heat	= 18,000	
And, say, 10% moisture—		
100 kilos. × (100° − 16°) =	8,400	26,400 ,,
Actual heat available =	291,600 calories	

Allowing 25% loss, which is very liberal, there is still available 211,700 calories, and this divided by 537 + 85 (water at 15° C.) = 622 = 351·6 kilos of steam.

From very numerous experiments it has been found that the calandria gives an average efficiency of 91%, but taking this at 80% there is 281·3 kilos. of clean steam available for the turbine.

The turbine makers will guarantee to use not more than 12·3 kilos. of steam per horse-power with 28·5 in. of vacuum and a vacuum of 7 in. over the water in the calandria. Taking the steam consumption as 13 kilos., there is per 1,000 kilos. of slag 21·5 h.p., say, 21 h.p.

Blast Furnace Slag.—By experiment it has been found that 1,000 kilos. of slag contain 550 calories of available heat, and upwards of 740 kilos. of water have been evaporated per 1,000 kilos. of slag.

1,000 kilos. of slag contain 550 calories per kilo. ...	... = 550,000 calories	
From this must be deducted—		
1,000 kilos. × 100 × 0·18 ... =	18,000	
And, taking the worst case where 50% of the water goes away with the slag, then—		
500 kilos. × 84 ... ... =	42,000	
	60,000	60,000 ,,
The available heat is		490,000 ,,
Allowing 25% loss there remains		367,500 ,,

And this divided by 622 gives ... 590·8 kilos. of steam. Again, taking the efficiency of the calandria at 8%, there remains 472·7 kilos. of clean steam, and with a consumption of 13 kilos. per horse-power, there is available 36·35, or, say, 36 h.p. per 1,000 kilos. of slag.

As the conditions at each plant are so diverse it is impossible to estimate the cost, but where slag is already being granulated, the only extra cost in the case of blast furnace slag is the second elevator; one man per shift is required to look after the running-in spout, and moving the trucks; this applies also to a copper plant.

Below are given figures showing the cost per kilowatt of a 1,000 kw. plant with 75% load using the heat from slag, and the cost of a similar plant using coal at 14s. 6d. per ton.

	Waste Heat Plant. d.	High-Pressure Plant. d.
Granulator—Wages ..	0·005	—
Repairs ...	0·005	—
Interest and redemption	0·010	Coals 0·210
Generator—Wages ...	0·092	0·092
Water, etc....	0·021	0·021
Repairs ..	0·057	0·057
Interest and redemption	0·100	0·100
Royalty	0·030	—
	0·320	0·480 per kw.

or a saving on the waste steam plant of 0·16d. per kilowatt.

On a plant of 1,000 kw. with 75% load the saving would amount to £4,300 per annum.

Apart from this saving there is a further advantage in this system. The water used in the primary generator need not be clean. The upkeep of the calandria and primary generator are small because the plates and tubes are not subject to great difference in temperature, as the slag goes directly into the water, and hence only steam at 100° C. comes in contact with any part of the plant.

The gases exert no action on the tubes or plates of the calandria, or heater, provided they are kept clean."—W. L. Johnson.—*Chemical World*, July, 1914, p. 207. (J. A. W.)

Platinum in Westphalia.—"The Deutschen Platinwerke meeting held at Dusseldorf was the occasion for the reading of a report upon the Westphalian platinum deposits, noted in our issue of January 31, 1914, by Mr. W. Hommel of the Clausthal mining academy. It is pointed out that the platinum occurs with lead, zinc, antimony, and nickel, and not in a colloidal state as previously supposed. As evidence of the successful treatment of these ores by a new process, there was exhibited a 500 gm. bar of platinum. This bar resulted from the treatment of 25 tons of ore. One ton of concentrate from the 25 tons of ore also yielded about 11% nickel. Mr. Hommel estimates that the cost of production, including mining, will be about 84.80 per ton."—Hommel.—*Mining and Scientific Press*, June 20, 1914, p. 1001. (G. H. S.)

The Embrittling of Iron by Caustic Soda.—*Conclusions.*—"Wrought iron when immersed in a concentrated solution of caustic soda corrodes slowly, becomes highly crystalline, and eventually brittle.

It has been suggested that the corrosion is due to the iron being constituted of two phases, a crystalline and an amorphous phase.

When a duplex metal of this nature is immersed in an aqueous solution of caustic soda, electrolysis sets in, iron going into solution at the anode and hydrogen being liberated at the cathode, whilst sodium ferrite is formed in the solution.

The hydrogen, after giving up its charge at the cathode, is partially liberated and escapes and partially occluded by the metal.

It is the occlusion of, and diffusion into the interior, of this hydrogen which brings about crystalline growth and brittleness in the metal, the hydrogen being first absorbed by the amorphous plate which exists between the crystals, thereby forcing the crystals apart.

When both the amorphous and the crystalline metal have become saturated with hydrogen, or partially saturated, the potential difference between the amorphous and the crystalline metal will drop, with the result that the rate of corrosion will diminish

The initial brittleness of the iron decreases with time, an equilibrium between the metal and the gas being finally established.

The brittleness is due rather to the molecular rearrangement brought about by occlusion or evolution of hydrogen than to the mere presence of hydrogen in solution.

The passivity of iron produced by immersion in caustic soda is due to the small potential difference between the hydrogenized amorphous and hydrogenized crystalline phases.

Precisely the same effects were noticed when using electro deposited iron. If, however, the metal be previously charged with hydrogen, the rate of corrosion is less, proving that the rate of corrosion is dependent upon the hydrogen content.

A 0·5 carbon steel is affected to a much less degree by caustic soda solution."—J. H. ANDREW, M.Sc.,—*Trans. of the Faraday Society*, March 1914, p. 329. (H. A. W.)

ZINC SHAVINGS.—"A small tube-mill, made from an oil tank such as oil is shipped in, has recently been installed at MacNamara mill, Tonopah, to grind the short zinc from the zinc boxes. The mill runs about 6 r.p.m., and is driven by a belt around the tube. On each end is bolted a 4½ in. flange, into which a short 4½ in. nipple is screwed and keyed forming the bearings, the feed, and discharge. The inlet nipple is reduced to 1½ in., through which is a 1 in. feed pipe for solution. A few pebbles are used in the mill to grind the zinc a little, and when working, rich solution is allowed to run through and discharge to a pipe leading to the first cell of a zinc box. One test showed 66 oz. of the metal in solution was precipitated in passing through the mill, and the precipitate coming out assayed 55 oz. bullion. Some very fine zinc is left in the mill, but the troubles with short zinc in clean-ups are over. The idea originated at the Tonopah Extension plant, where a barrel, containing a few pebbles set at an angle of 40°, was slowly revolved by gearing. Results show that there is no need of any acid treatment. It is used only a few days each month at the MacNamara, and prevents the short zinc from being coated with white precipitate. It also reduces the cost of cleaning the zinc boxes by about 25%."—J. B. TREGLOAN.—*Min. & Sci. Press*, Feb. 14, 1914, p. 287. (H. A. W.)

HARDNESS OF ORES.—"The writer has originated a simple and accurate method for comparing the hardness of ore sands judged from a tube milling standpoint. He would emphasise that the comparison does not extend to any other crushing machine. The principle of crushing in a tube mill is quite different from that employed in the grinding pan or the stamp battery; and what might be considered as excellent crushing material for one machine may prove quite unsuitable for another. This is illustrated by the fact that the most easily ground sand shown in the subjoined table is derived from an ore which in battery parlance is the 'toughest' in the series. The method of conducting the test is clearly explained, and it should be remembered that like all comparison tests the value of each result depends on how far the working formula has been adhered to by the operator. Every test must of necessity be carried out in an exactly similar manner, and the proportions must be strictly adhered to. I.M.M. screens are used because they are the only screens here whose aperture diameter can be guaranteed for a given mesh.

Stadler's excellent method for computing grinding efficiencies is used to illustrate the respective hardness of the ores tested, but the regrind factor is only an arbitrary term used for want of a better. Any standard method of computation would have given a similar comparison.

In the description of the method adopted it is stated that the sand tested must be obtained from wet-crushed samples. This principle can be safely deviated from at times, and if the sample to be tested is put through and through a sample cracker until the required weight of −20+40 mesh sand is obtained, the result will be accurate. The point is to keep the physical characteristics of the sand always the same. Particles of sand derived from the wet crushing of rocks, and those derived from dry crushing in rock breakers, are always angular, but the particles derived from crushing rock in ball mills or disc pulverisers are rounded. Microscopic examination of the sand would be found of great assistance in determining its physical character. Naturally, the grinding result on a sample of sand will vary greatly according as to whether the sand particles are round or angular; and this explains why it is necessary to prepare the sand by either of the processes above described. The capacities of tube mills vary a great deal on different ores and only experiment will indicate the work that will be performed in any given case.

In designing new plants the duty of a grinding machine causes the designing engineer a lot of anxiety, as the expected value of the residue is very often strictly proportionate to the lineness to which the ore is ground. At present the duty on an untried ore can only be guessed from its appearance which is often deceptive. A standard preliminary small scale test from which the capacity of a tube mill can be accurately estimated would be of great value, and the suggested scheme endeavours to supply that want. The attached 'Table of Hardness" gives the tabulated results of testing sand from some 15 mines, and also that derived from crushing an average sample of Danish Flints. Unfortunately, it is quite impossible to get correct tube mill crushing data in each case. The writer has not sufficient faith in any method yet propounded for comparing grinding efficiencies to attempt a comparison from the grinding results of tube mills working under different conditions. Successful comparison can only be attempted when the tube mills are working under similar conditions with regard to speed, pebble load, grade and tonnage of feed, etc., etc. In only two cases in the subjoined table has it been possible to accurately compare the work done by similar sized tube mills in the respective plants. In the case of No. 5 mine, whose sand has a regrind factor of 354·5, a tube mill has a 30% greater capacity than on the sand from No. 11 mine with a regrind factor of 266·3.

Tests on tube mills working under differing conditions on other mines indicate that this ratio will remain about the same, but these tests cannot be accepted as reliable. The writer hopes later to be able to record accurately tube milling data from some of the other sands in the series. This would greatly add to the value of the table. An elaborated table of this nature would allow of operators working in different parts of the world accurately comparing grinding efficiencies under similar conditions. Anything of this nature is at present impossible. Some standard should be established, and this is introduced as an embryonic test which can be made by any operator who can secure a set of I.M.M. screens and a 5-gallon glazed earthenware ginger-beer barrel. A study of technical literature dealing with grinding capacities and grinding efficiencies illustrate the fallacy of generalisation. It is quite useless comparing capacities and efficiencies of grinding machines unless the ores operated on can first be accurately compared.

Method of Computing Hardness.—A sample of ore to be tested (wet crushed) is graded (I.M.M. screens) until 8 lb. of −20+40 sand is obtained. A charge consisting of 8 lb. of flintstones, 1 in. to 2 in. diam., and 16 lb. of water, is introduced into a five gallon glazed earthenware ginger-beer barrel and ground for 90 minutes at 60 R.P.M. A sample of the ground product is then taken, dried and graded with a mechanical grader 40, 60, 100 and 150 mesh I.M.M. screens. Grading occupies 20 minutes. The value of the grading in energy units is then calculated by Stadler's method and this amount, minus the original value in energy units, represents the regrinding factor in the table.

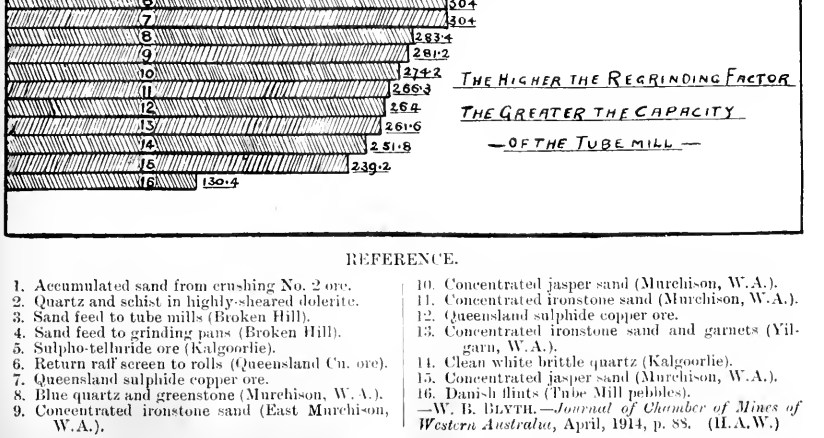
EXAMPLE.

Feed.				Tails.			
Screen.	Mean Factor Stadler.	Percentage.	Energy Units.	Percentage.	Mean Factor.	Energy Units.
+ 40	17·5	10	= 175	16·8	17·5	= 294·0
40/60	...	...	...	29·2	× 20·0	584·0
60/100	...	...	...	16·4	× 22·0	360·8
100/150	...	...	...	6·6	× 24·0	158·4
− 150	...	...	...	31·0	× 26·5	821·5
					Total...	2218·7

Total E.U. Tails 2218·7 Total E.U. Feed = 1750
Difference 468·7 = Regrinding Factor.

TABLE OF HARDNESS.

Based on Grinding Capacity of Tube Mills run under the same conditions. Regrinding Factors in Terms of Energy Units.



REFERENCE.

1. Accumulated sand from crushing No. 2 ore.
2. Quartz and schist in highly-sheared dolerite.
3. Sand feed to tube mills (Broken Hill).
4. Sand feed to grinding pans (Broken Hill).
5. Sulpho-telluride ore (Kalgoorlie).
6. Return raff screen to rolls (Queensland Cu. ore).
7. Queensland sulphide copper ore.
8. Blue quartz and greenstone (Murchison, W.A.).
9. Concentrated ironstone sand (East Murchison, W.A.).
10. Concentrated jasper sand (Murchison, W.A.).
11. Concentrated ironstone sand (Murchison, W.A.).
12. Queensland sulphide copper ore.
13. Concentrated ironstone sand and garnets (Yilgarn, W.A.).
14. Clean white brittle quartz (Kalgoorlie).
15. Concentrated jasper sand (Murchison, W.A.).
16. Danish flints (Tube Mill pebbles).

—W. B. Blyth.—*Journal of Chamber of Mines of Western Australia*, April, 1914, p. 88. (H.A.W.)

DOME MILL.—"Alterations and additions to the Dome mill at Porcupine, Ontario, will result in the following plant, it having been decided in 1913 to convert the existing process of all slimming to one of leaching sand and filtration of slime, according to the general superintendent, H. C. Mack. The ore, after being delivered from the rock crusher station, is to be crushed by 80 stamps (40 added) and five tube mills, the products from the tube mills being then distributed over 24 amalgamating plates, on which over 50% of the gold will be recovered. The product is then to be classified into sand and slime by means of hydraulic classifying and concentrating cones. The sand will be delivered to 6 vats, each 40 ft. diam. by 8 ft. 6 in. deep, for leaching by cyanide solution. The slime will be treated in the existing plant of four 8 by 10 ft. Pachuca vats, and three 90 four in. frame Merrill slime presses. The gold will be precipitated in 52 in. Merrill zinc-dust presses as during the past."—*Mining and Scientific Press*,—June 27, 1914, p. 1064. (G. H. S.)

CLASSIFICATION AT MIAMI. "To obtain the best possible results in the concentration of ores, it is generally accepted as standard practice that classification is essential, that the different grades of feed thus separated may be treated on different tables and to the best advantage. The methods of classification are generally hydraulic, and the classifiers are either mechanical or stationary in operation. Of the latter type, baffle and hindered-settling classifiers are being favoured by the operators. After considerable experimental work at Miami during which various types of classifiers were tested, the subject of tonnage, thickness of discharge, water required per ton of solids handled, proportion of coarse material on discharge, accessibility and ease in replacement of wearing parts were subjects of investigation. The Deister cone baffle classifier, a cross-section of which is shown in Fig. 1, was found adapted to conditions and has been installed. This classifier constitutes a cast iron barrel in which hollow cones or diaphragms with slots or openings sufficiently large to permit the free passage of the coarser material are placed in an inverted position. These pots or cones are staggered, the openings in one cone being out of line vertically with the adjacent cones, thus forming baffles. In operation, the crushed material which is mixed with sufficient water to insure its movement along the launder passes down into the barrel through the openings into the cones. At the same time sufficient water is admitted through one of the side inlets of the annular waterway, and of sufficient force to separate the fine from the coarse material passing through the cones, forcing these fines to the top and on to the next classifier, allowing the downward movement of the coarser particles.

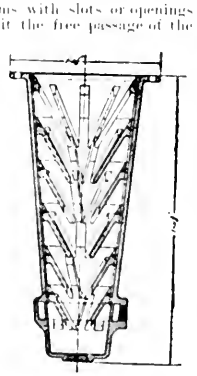


FIG. 1.

Classification does not result from the use of hydraulic water alone, but the pots act as baffles and the staggered path of the material allows the water to wash the fine particles from the downwardly floating coarse material, carrying them above and beyond to the next classifier. A nipple and wooden plug is used in the bottom of the classifier spigot. The opening through the plug may vary from ¾ in. to ¼ in.

The results obtained at Miami by the use of this classifier may be seen from the following representative tests:—

Tons solid per 24 hours	33·60	37·90	23·90
Ratio solids to water in product ..	1:1·77	1:2·38	1:2·64
Hydraulic water, gallons per minute ..	17·50	22·40	*
Screen analysis of product:			
−10 mesh ..	0·10	0·10	0·00
+20 ,, ...	4·70	6·10	12·70
+30 ,, ...	37·50	38·60	44·70
−40 ,, ...	24·30	19·80	19·20
60 ,, ...	22·00	19·40	14·50
80 ,, ...	5·00	5·70	3·70
−100 ,, ...	2·90	4·10	2·20
100 ,, ...	3·50	6·20	3·00

* Not taken.

From the results tabulated, one cannot fail but be impressed with the tonnage and ratio of solids to water in the product. From the screen analyses of the 10-mesh material it is to be remarked that in each test from 78 to 83% of the product is between 30 and 60 mesh. If curves were plotted with mesh and percentage as coordinates, the sharp rise between the +20 and +30 and sharp fall between the +60 and −80 would indicate graphically the degree of classification which is being effected.—*Mining and Scientific Press*, June 27, 1914, p. 1057. (G. H. S.)

THE VOLATILIZATION OF METALS AS CHLORIDES. "The author discusses the results obtained by Mr. Ben Howe at the Gwalia Consolidated Mine and points out the similarity to those obtained by himself, as described in *The Engineering and Mining Journal*, for August 29th, 1903. The degree of fineness to which the ore should be crushed and the relation of salt to sulphur to obtain the best volatilization are also mentioned. As much as 97.7% of the gold content of a Cripple Creek ore was volatilized in 60 min. in laboratory tests. Types of furnaces for roasting the mixture of ore and salt and the temperature required are discussed. Methods of recovering the metals from the fumes are dealt with, the author having obtained a recovery of 96% of the gold. In ores containing copper some difficulty was encountered in arresting the copper chloride fume. The author suggests woollen bags, as these gave the best results in laboratory experiments."—STUART CROASDALE. —*Mining Magazine*, March, 1914, p. 200. (J. E. T.)

TREATMENT OF CYANIDE PRECIPITATE.—"The elimination of the zinc from cyanide precipitate by an acid treatment or volatilising roast is now generally omitted, the material being melted down at once into dore bullion which is parted. Reverberatory lead smelting of the precipitate and cupellation of the lead bullion is practised on the Rand and in New Jersey, U.S.A.; smelting and cupelling may be carried out in the same furnace by using a different bottom for each operation. The introduction of oil fired tilting furnaces has resulted in a quicker and cheaper process; the Monarch-Rockwell

is a double-chamber furnace heated with an oil burner through a hollow trunnion upon which it turns for discharging the metal. In the Merrill process used at the Homestake Mill, S.D., the zinc is extracted by sulphuric acid; high-grade precipitate is briquetted and cupelled direct; low-grade material is melted into lead bullion in a circular water-jacketed blast furnace having a lead well; copper if present is recovered in a forehearth as matte by adding pyrite to the charge. Electric arc furnaces have been successfully applied to precipitate smelting; the melting is rapid, especially if the material has been previously dried. Working costs of the various methods of treatment are given."—H. S. MEGRAW, *Eng. & Min. Journal*, 1914, 97, 505-509, 606-612.—*Journal Soc'ty Chemical Industry*, April 30, 1914, p. 423. (J. G.)

MINING.

THE RAND BANKET.—"In certain Cambrian gold-bearing conglomerates in the Black Hills of South Dakota studied by J. D. Irving, the same problem is involved. There, as on the Rand, gold and pyrite, are generally distributed through the beds worked, and there are local enrichments. Mr. Irving found that much of the gold was detrital and held that the original sediments contained iron. The formation of the pyrite, according to him, represented action subsequent to deposition and was accompanied by enrichment of certain areas in the conglomerate. Original sedimentary process might, it is true, be appealed to in explanation of such enriched parts of the conglomerate. Take for example the gravels now being dredged at various points along the Sierra foothills in California. In these, gold occurs distributed as on the Rand; there are definite thin beds of gravel, corresponding to the 'reefs' in a thick series of sand and gravel, contain the workable gold; the profitable beds are miles in extent, and within each workable area there are rich and lean portions, often in fact definitely marked pay-streaks. The California placer deposits are not so rich as the Rand bankets, and they are not so thoroughly cemented, but at least they are analogous to the possible original fossil placers that have been assumed by others to have been what F. L. Ransome would call the 'prot-orebodies' of the Rand.

The California placers occur at the embrochure upon the plains of rivers traversing auriferous areas. Essentially they are delta deposits. In this connection it may be noted that Mr. Irving found the Cambrian conglomerates already mentioned to be littoral sediments. Examination of fluviatile placers, the usual type, shows that long, narrow, and usually crooked pay-streaks are the rule. The map of no river placer of which I know resembles the most interesting stope maps of the Rand which Mr. Horwood has given us.

T. A. Rickard has pointed out the differences between the Rand bankets and the beach gravels at Nome, and if T. M. Gibson's map* of the Third Beach line be compared with the Rand stope maps, it will be seen that there is practically no resemblance in form. Indeed, the Nome pay-streak is characteristically long and narrow, and is one formed by river action; it differs in being straight rather than crooked. It is to be remembered, however, that the Nome gravels contain gold concentrated and reconcentrated by active waves on a long gently sloping beach line with shallow water. These conditions would seem most favourable to bringing the gold into the narrowest possible streak along the shore. The California placers, on the contrary, were formed where there is little evidence of wave action. They are alluvial cones formed in quiet water, or even possibly in flood times above normal water level, just as material is being spread over their tops now. The point is that there are types of placer deposits that show just such distribution of gold as on the Rand, and further, that at least one fossil placer of this type is known. That there has been redistribution of the gold in the placers, proceeding furthest on the Rand and least in the very modern California deposits, is quite in accord with the ordinary history of rocks, and does not require, though it may be facilitated by, unusual geological agencies such as the intrusion of eruptive rocks.

Assume, on the other hand, that the gold was, as Mr. Horwood contends, introduced later as an incident of igneous activity, where is any analogous deposit? The nearest correlative that I have seen is, again, in the Black Hills, where there are certain 'silicious' deposits also studied by Mr. Irving. These are sandstones mineralized by solutions entering along transverse fissures and cracks. A brief visit to these mines, reinforced by a study of his admirable cross sections and plans, shows that the result is not at all the same as on the Rand. The gold is localized and is much more definitely related to the fissures through which the solution entered than the Rand stope maps indicate. Briefly, pay-streaks were formed rather than mineralized beds. In the Black Hills the two methods of origin are contrasted, the examples being in beds of possibly the same approximate age as on the Rand, and other conditions, including later intrusion of igneous rocks being the same. Mr. Irving has presented only one stope map of 'conglomerate' workings, and I am not personally familiar with these deposits, but so far as evidence at hand goes, the argument from analogy would seem to favour considering the Rand a re-worked placer. Argument from form, it must be confessed, is but one possible mode of approach and does not give a final answer. It may, however, justify an attitude of suspended judgment, even in the face of such evidence as Mr. Horwood has presented."—H. FOSTER BAIN.—*Mining and Scientific Press*, June 20, 1914, p. 1022. (G. H. S.)

THE RAND BANKET.—"Mr. Horwood summarises his list of arguments in favour of the infiltration theory as follows (p. 1005): (1) The great depth to which the profitable ore extends; (2) the dryness of the mines in depth; (3) the abundant water-ways by which mineralisers 'could have ascended'; (4) the crystalline character of the gold; (5) the occasional association of the gold with vein quartz; (6) the close association of the gold and pyrite; (7) the occasional presence of calcite, tourmaline, galena, blende, copper pyrite, pyrrhotite, iridosmine, and other metals of the platinum group; (8) the pseudomorphic origin of the pyrite pebbles and their shape and association with rich ore; (9) the distribution of the gold.

1. The first of these points is one of those in which Mr. Horwood's paper strengthens my doubts of the infiltration theory. He shows, from the evidence up to date, that a steady reduction in the grade of the ore with increasing depth has not yet been proved.

There is no doubt a fall in the grade milled, but that does not prove any fall in the actual grade of material in the mine; for owing to the remarkable success of the Rand mining engineers in lessening

Mining and Scientific Press, April 25, 1914.

working costs, lower grade material is now being mined which was formerly left underground. The connection of the fall in grade with the increase in the tonnage milled may be illustrated by the diagram. Mining, moreover, of course started where the richest ore occurs at the surface; and these mines would naturally become poorer as they approach the margin of these richer areas. Mr. Horwood's summary of the evidence shows that the Rand as a whole displays no such average decrease of actual grade in depth as there is in all great gold fields. On the infiltration theory, we should expect such a fall; for the deposition of the gold would be more concentrated near the surface where the physical and chemical conditions altered more rapidly.

2. The second point, the dryness of the mines in depth, would apply equally well to some coal mines, which might be represented as lodes due to bituminous infiltrations. In fact, all the arguments for the infiltration theory could be used to explain coal seams as due to bituminous infiltration.

3. Mr. Horwood tells us that there are abundant channels up which the mineralising waters 'might have come'; but we are still without actual evidence of channels up which such waters did come. Mr. Horwood quotes instances in which the dikes have had an apparent influence on the banket, and, as he says, I quoted various instances of the same sort. But I regarded those as coincidences; for assay plans from many parts of the Rand showed that in a very much larger number of cases the dikes were not associated with any change in the distribution of the gold. Kessler and others, after very careful consideration of this question, had come to the conclusion that, in spite of occasional exceptions, the overwhelming balance of evidence shows that the dikes have had no influence on the richness of the adjacent ore.

If the gold had been introduced by filtration, very many actual channels ought to have been discovered. But there is no evidence that any one of them has ever been found, except in the case of the small secondary veins which were formed after the banket.

One of the most important features of Mr. Horwood's paper in his series of valuable photographs showing the microscopic structure of the banket. They show that the banket, in its microscopic characteristics, is quite unlike any known lode. Lodes formed by infiltration, unless deposited in a fissure, usually show a gradual passage from the country rock through a silicified stage, into sheets of vein quartz; but in the banket many of the richest samples have been no more altered than barren samples. As the matrix of the banket is composed mainly of grains of silica, it would have been readily altered into vein quartz. The matrix of even the richest Rand ore is, however, often practically unaltered. I have studied microscopically the quartz veins from four continents, and I know of no lode of rich gold ore in which the matrix has not been greatly altered, although it may, under suitable illumination, show traces of its original structure.

4 and 5. The fourth argument the crystalline character of the gold is part of the placer theory, according to which the gold has been redissolved and redeposited during the cementing of the banket. The occasional gold quartz veins have no bearing on the main question, as they are secondary products of the same period. They are only of historic interest.

6. The close association of gold and pyrite is not constant some rich ore is poor in pyrite and some poor ore very rich in it.

7. The seventh feature, the presence of the materials enumerated, is fully consistent with the placer theory, for most of them must have been present as detritus in the pebble beds, and the rest (the diamonds, platinum, and iridosmine) are more easily thus explained.

8. The eighth, the pseudomorphic origin of the pyrite pebbles, was originally advanced to remove a difficulty from the placer theory.

9. The distribution of the gold, its limitation in paying quantities to the beds of water-worn banket, and its absence from the beds of equally permeable 'bastard' reef above and below the rich seams, are facts also consistent with its detrital origin.

Mr. Horwood's arguments for the infiltration theory appear to be not only consistent with the placer theory, but to tell in its favour.

The great quantity of gold in the banket is no doubt unique, but it is also unique for a lode field, and the classic theory most readily explains this feature. The formation of a goldfield is a process of concentration of a once diffused material. The placer theory adds an extra process by which gold, after a first concentration in the reefs of the metamorphic zone beside the Rand, was further concentrated during the denudation of the country.

The placer theory gives a simple, reasonable, and natural explanation of the origin of the gold, for it attributes its source to the vast quartz-lode series that existed on the Rand before the formation of the banket: the infiltration theory, on the other hand, introduces the gold from a hypothetical and improbable source, along solution channels of which we still have no evidence, and, according to Mr. Horwood's version, at a date long after the banket had received its gold." J. W. GREGORY.—*Mining and Scientific Press*, June 20, 1914, p. 1020. (G. H. S.)

ORIGIN OF RAND GOLD.—"E. T. Mellor, during the last four years in his capacity as Government Geologist, has been investigating the deposits throughout the length of the Rand. Mr. Mellor states that he started his investigations with a mental inclination toward the infiltration theory, but during his four years' work has not obtained any evidence in favour of the theory. On the contrary he found much evidence in favour of a placer origin. He states that those who compare the evidence in favour of the two theories will note that while the arguments for the placer theory are for the most part of a simple character, based upon direct observation of and comparison with similar deposits about the origin of which there can be no doubt, those in favour of the infiltration theory are frequently highly speculative or theoretical. That there has been considerable local redistribution of the gold content of the 'reefs' no one acquainted with them will be inclined to doubt. In many cases the redistribution has naturally produced effects which, though purely secondary, are suggestive of an infiltration origin for the whole deposit. The local character of some of these effects is often strikingly shown. With regard to the influence of dykes, Mr. Mellor says that against the few examples of such intrusions being associated with specially rich portions of the gold-bearing beds cited by Mr. Horwood, must be set dozens of examples in which either entirely opposite conditions are found or where nothing unusual is to be noticed. Though Mr. Mellor in his investigations has paid special attention to this point, he has not met with any instance where the presence of richer 'reef' was clearly dependent on the intrusion of a dyke except

as a purely secondary and extremely local result. He found several instances where the accidental disposition of a dyke with regard to a rich patch at sight suggested a causal relationship, but further examination never confirmed this view."—E. T. Mellor.—*Mining Magazine*, July, 1914, *Mining and Scientific Press*, May 6, 1914. (G. H. S.)

Knock-Down Sample Catcher.—"It is sometimes difficult to get a suitable receptacle in which to catch samples cut from an ore face. Various sample catchers made of cloth on metal frames have been devised. One I designed and have found satisfactory consists of a frame, a handle and a canvas bag sewed to the frame. The frame is made of ½ in. wrought-iron pipe in two parts joined together by two unions. Into a ½ in. tee bushed into one side of the frame is screwed a handle of ½ in. wrought-iron pipe, which can be any length desired and can be set in any position by turning the ½ in. tee. A convenient length of handle for ordinary sampling is 2 ft. For ease of carrying in baggage, the handle can be made of a number of pieces of pipe, 12 in. long, screwed together with couplings when in use. A ½ in. tee at the end of the handle gives a firm grip for the hand and makes the catch easier to hold.

A canvas bag is suspended from the frame by being sewed to it. Openings are left at the two unions so that they can be screwed together and unscrewed. The bag is made to slope toward the handle side of the frame, bringing the centre of gravity of the loaded bag as close to the handle as possible. The 6 in. opening at the bottom of the sack is closed by means of a pucker cord which is laced through 12 eyelet rings.

By means of the two unions, the frame can be taken apart and one-half folded over on the other half, so as to make a compact bundle taking up little space in the engineer's baggage."—E. J. Collins.—*Engineering and Mining Journal*, June 27, 1914, p. 1289. (G. H. S.)

MISCELLANEOUS.

Preparation of Eye-Preserving Glass for Spectacles. —"The main object of the research was to prepare a glass which would cut off those rays from highly heated molten glass which damage the eyes of workmen, without obscuring too much light or materially affecting colours of objects seen through spectacles made of glass. The screening properties of glass plates for ultra-violet, luminous, and infra-red radiations were investigated. Photospectrographs were taken, with exposures varying up to 3 hours, of the radiations from a glass tank in actual practice, the temperature at the melting end being 1500° C., and at the working end 1200° C. Infra-red rays being found in far greater abundance than ultra-violet, it is inferred that glassworkers' cataract must be ascribed principally to heat rays. A series of glasses was made by adding pure metallic oxides and earths as colouring or absorbing materials to a soda flux. Absence of striæ was obtained by repeated stirring with a platinum rod; and freedom from air-bubbles, by leaving the glass in perfect repose from 1½ hours at the maximum temperature. After cooling for 12 hours, plates were cut, ground and polished to a thickness of 2 mm. By superposing on the radiation from a Nernst lamp the light from a high-tension discharge between the poles of pure metallic uranium, a practically continuous beam extending from λ 2000 to λ 8000 was obtained, and the absorption of the glass plates recorded on a spectrograph. Biotite (black mica) from Norway and 'black amber' mica from Africa were found most effective in obstructing light while transmitting heat rays. Athermancy of various glasses by the readings of a therm. bulb corresponded with the results obtained from a radiometer balance. The prepared plates were first tested in the spectrum apparatus to ascertain the upper limit of transmission of ultra-violet rays. They were next put into the radiometer balance, then tested in an opacity balance to ascertain the percentage of luminous rays transmitted, and finally their colour was registered in a Lovibond tintometer. The composition of the glasses selected for practical use, in percentages by weight, is as follows: *150*: fused flux 90·00, cerium borate 8·13, $NiSO_4,7H_2O$ 0·07, U_3O_8 1·80; *158*: fused flux 89·75, cerium borate 8·13, Fe_2O_3 2·03, Cr_2O_3 0·09; *165*: raw flux 87·56, cerium borate 8·00, $FeSO_4,7H_2O$ 3·00, UO_3 0·35, NiO 0·09, Ce_2O_3 0·80; *187*: fused flux 83·0, $Ce(NO_3)_6,6H_2O$ 17·0; *197*: fused flux 79·00, $Ce(NO_3)_6,6H_2O$ 20·50, $NiSO_4,7H_2O$ 0·30, $CoSO_4,7H_2O$ 0·05, U_3O_8 0·15; *202*: flux 95·45, Fe_2O_3 4·75, $CoSO_4,7H_2O$ 0·40; *210*: fused flux 89·0, $FeSO_4,7H_2O$ 8·9, Cr_2O_3 1·3, carbon in fine powder 0·8; *217*: fused flux 96·80, Fe_2O_3 2·85, C 0·35; *221*: fused flux 80·0, $Ce(NO_3)_6,6H_2O$ 13·4, U_3O_8 6·6; *238*: raw flux 77·0, $Ce(NO_3)_6,6H_2O$ 23·0; *240*: raw flux 90, $FeC_2O_4,2H_2O$ 10; *240*: as 240 with a small quantity of red tartar and powdered wood charcoal to prevent oxidation; *247*: raw flux 92·00, cerium borate 6·30, NiO 0·04, Fe_2O_3 1·60, Cr_2O_3 0·06; *248*: fused flux 94·60; $Ce(NO_3)_6,6H_2O$ 4·72, UO_3 0·30 NiO 0·30, $CoSO_4,7H_2O$ 0·08; *249*: fused flux 88·47, Fe_2O_3 1·50, $CoSO_4,7H_2O$ 0·03, $Ce(NO_3)_6,6H_2O$ 10·00; *250*: raw flux 88·00, cerium borate 5·00, $FeSO_4,7H_2O$ 4·15, UO_3 2·75, Cr_2O_3 0·10; *251*: raw flux 92·0, $FeSO_4,7H_2O$ 8·0; *252*: raw flux 72·60, $Ce(NO_3)_6,6H_2O$ 24·90, $CuSO_4,5H_2O$ 2·10, NiO 0·40; *253*: raw flux 88·5, fused black biotite 11·5. The last mentioned glass has properties contrary to what might be expected from its biotite content; it offers almost complete obstruction to heat rays. The transmissive powers of biotite itself appears to be due to iron protoxide. In the foregoing recipes the flux used has the com-

No.	Heat Rays absorbed	Absorbs violet and ultra-violet rays of wave length shorter than:	Luminous rays transmitted	Colour.
150	37	3620	73	Pale Yellow.
158	63	3700	54	Pale greenish yellow
165	38	3680	42	Pale yellowish green
187	27	3650	99	Practically colourless
197	41	3800	45	Pale neutral tint
202	83	3830	25	Neutral tint
210	87	3620	30	Bluish green
217	96	3550	40	Bluish green tinge
221	39	3685	60	Faint yellow
238	34	3610	71	——
240	88	3950	36	Smoky green
246	98	3800	27·6	Sage green
247	29	3620	71	Faint green
248	47	3550	30	Neutral tint
249	51	3550	63	Pale blue
250	25	3685	74	Yellow, green tinge
251	37	3550	89	Faint yellow
252	47	3680	45	Faint bluish green
253	94	3610	30	Sage green

position: Sand 61·00, anhydrous sodium carbonate 25·50, recrystallised sodium nitrate 5·00, precipitated calcium carbonate 7·20, borax 0·77, arsenic trioxide 0·55%. The optical properties of the glasses are shown in the table.

Glasses Nos. 246, 217, 253, 240, 210, and 158 are most effective for cutting off heat rays; Nos. 240, 202, 246, 197, 158, 221, and 150 for absorbing ultra-violet rays; Nos. 187, 254, 250, 150, 247, and 238 for transmitting luminous rays; and Nos. 249, 197, 252, 165, 240, and 218 for reducing glare (of sun on chalk, snow, or sea.) SIR W. CROOKES.—Phil. Trans. 1914 (A. 509). *Journal Society Chemical Industry, XXXIII, 13, June 30, 1914, p. 646.* (J. A. W.)

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Abstracts of Patent Applications.

(C.) 339/13. Gaston Jacquier. Improvements in the precipitation of metals from solutions of their salts, and apparatus therefor. 16.7.13.

This application refers to a process for the precipitation of metals from solutions of their salts.

(C.) 388/13. William Holman James. Electrochemical treatment of metal bearing material. 15.8.13.

This application relates to a process for removing the refractory constituents of a gold ore prior to treatment by ordinary solvents. This is effected by electrolysing a solution of a salt of a mineral acid (such as sodium chloride) in such manner as to permit the withdrawal of the electrolysed liquid from each electrode separately. One portion of such liquid is passed directly to the refractory ore, thereby dissolving the constituents which render the ore refractory. The liquid has preferential action on the refractory substances—no precious metal being dissolved. Usually the anode product (acid) is used for treating the ore, but in certain cases, *i.e.*, where arsenic is the refractory substance, the cathode (alkaline) product is preferable.

(C.) 124/13. Henry Robertson Lawson. Improvements in assay furnaces. 4.9.13.

This application relates to assay furnaces comprising a muffle and a crucible chamber combined in one structure.

(C.) 108/14. Jack Yates (1), Charles Heyer (2). Improvements in safety devices or arrestors for colliery winding cages, hoists and the like. 12.3.14.

This specification relates to safety devices for colliery winding cages, hoists, etc., and consists of a gripping appliance to be applied either to the guide ropes or rods of the cage.

The grippers for use on guide ropes or rods have semi-circular faces which may be either serrated or grooved and the grippers for use with guides may have plain or serrated faces.

The grippers are brought into action as soon as the winding rope breaks, by means of weights at the ends of long levers or by springs, which force the gripping face on to the guides.

(C.) 113/14. James Rossiter Hoyle (1), Henry Warburton (2). Improvements in and relating to tappets for stamp batteries. 12.3.14.

This application refers to a sectional tappet for stamp batteries and is in every way similar to an ordinary tappet except in that it is in halves. The halves are held together by two dovetailed grooves, which engage them frictionally where the ordinary keys are driven in on the gibs.

(C.) 123/14. Charles Leonard de Wilde Reader. Improvements in or relating to air and like fluid compressors and exhausters. 19.3.14.

This application has reference to air compressors and the like, in the provision of one or more pairs of multiple non-return ball valves, each pair being secured together in a common recess in the machine or cylinder so that one serves for the inlet valve and the other for the outlet valve, substantially as described.

(C.) 131/14. Johan Hjalmar Lindholm. Improvement in process of producing calcium cyanamide. 26.3.14.

The applicant claims the patent rights of a process for producing calcium cyanamide from calcium carbide and nitrogen, consisting in using a calcium carbide containing a percentage of heat absorbing but otherwise inactive substances, heating the calcium carbide to a suitable temperature, and having the nitrogen act upon the calcium carbide at an over-pressure suited to the said percentage of heat absorbing substances.

(C.) 132/14. Johan Hjalmar Lindholm. Improvements in process of and means for producing calcium cyanamide. 26.3.14.

The applicant sets forth eight claims which vary very little from one another. The aim of the applicant is, while working at a high temperature in order to facilitate the reaction between the carbide and nitrogen at the same time to prevent the dissociation of the formed cyanamide, and obtain thorough contact of the carbide with the nitrogen, and for this purpose he uses a specially constructed furnace to which however he does not limit himself.

(C.) 134/14. Samuel Douglas McMaken. An improved air lift for circulating and mixing materials in a more or less liquid state. 27.3.14.

This application has reference to agitators of the Pachuca or Brown type which are fitted with a central tube or air lift pipe. This central pipe is fitted with a series of valves, which can open outwardly due to internal pressure but close automatically if the outward pressure is greater. These valves are fitted at intervals at different heights from the bottom of the tube. The circulation of the charge in the tank therefore commences as soon as the level of the pulp in the vat rises to the first valve and as the level of the pulp rises during the period of charging, the lower valve automatically closes and the next upper valve comes into action and so on until the vat is filled.

It is claimed that there is an improved circulation of the materials due to this taking place at various levels, so that there is no settlement at any time during the filling or discharging period, and special mention is made of the advantage of this when the whole product of sand and slime is treated together in the agitating process.

(C.) 172/14. Frederick van Duzer Longacre. An improvement in regulators for fluid compressors. 23.4.14.

This application refers to a regulating device for use with a compressor having a variable unloading mechanism consisting of an adjustable stop mechanism for preventing the unloader from loading the compressor above a desired maximum.

A regulating device for use with a compressor consisting of a series of clearance chambers, a valve mechanism for serially throwing said clearance chambers into communication with the compressor cylinder, as the receiver pressure increases, and a manually adjustable stop mechanism for limiting the closing motion of the said valve so that one or more of said chambers are always in communication with the compressor cylinder.

(C.) 189/14. Thomas Frederick Wall. Improvements relating to electrically operated drills and other reciprocating apparatus of the percussion type 30.4.14.

This patent application is as follows:—

(1) In electrically operated rock drills and other reciprocatory apparatus of the percussion type, the combination with a pair of series connected solenoids, of a pair of series connected non-inductive resistances arranged in parallel with the solenoids and having their intermediate or common point connected both to the intermediate or common point of the solenoids and to a controlling switch which alternately short-circuits each solenoid and the corresponding non-inductive resistance substantially as described.

(2) In electrically operated rock drills and other reciprocatory apparatus of the percussion type, having a series arrangement of solenoids with non-inductive resistances as characterised in the preceding claim the arrangement of the solenoids in separate cases of magnetic material, substantially as and for the purpose described.

(C.) 190/14. Josef Carl Steinbach. Device for automatically stopping electrically driven hoists when the speed deviates from the correct value during retardation. 30.4.14.

This specification relates to improvements and additions to the control apparatus, covered by the Inventor's existing Transvaal Patent No. 668/11. The object of the invention is to stop winding motors more gently than was possible with the device described in the earlier patent.

The more gradual stopping is obtained by slowly and automatically bringing back the control lever to the neutral position, whereby the rotor resistance is increased and, if necessary, the bringing back of the

control lever is carried beyond the neutral point so as to apply reverse current. In the earlier patent the winding motor was suddenly brought to rest by the cutting off of the main supply and the application of the brakes.

A small auxiliary generator driven from the hoist motor is used as in the former invention, and the automatic movements of the control lever are carried out by means of compressed air with valves controlled by an auxiliary circuit.

(C.) 197/14. William Armor Stedman. Improvements in and relating to filtering apparatus. 7.5.14.

The above patent refers to an improvement in filter leaves of the kind in which material is deposited on either side of a porous or cellular supporting member which prevents the filter from collapsing when the pressure between the sheets of filter material is reduced: more particularly the invention has reference to filter leaves of the type in which the slime cake is dislodged without the use of internal pressure tending to distend the filter material. Advantage is taken of the discovery that if the interior of the filter leaf is entirely filled with water so as to saturate the entire surface of the filter leaf when the slime cakes are thereon, these will quickly fall off. And as the cakes do tend to more closely adhere to the filter material if the external pressure is in excess of the internal pressure, it is the purpose of this invention to equalise the hydrostatic pressure on both sides of the filter material during the dislodging operation. This is attained by allowing the lower edges of the filter material to hang loosely down below the bottom frame member, thereby forming flaps which act as flap valves, closing under vacuum, but nevertheless enabling the liquid for dislodging the cakes to readily flow from inside outwards should the pressure inside exceed the outside, while preventing access of liquid from outside to the interior of the filter leaf, except through its surface during the time that suction is applied. As the filter material is open across the bottom any foreign matter depositing on the cocoa-mat filler will find a ready passage out of the leaf, thus greatly avoiding the choking troubles now being experienced with the leaves in use. The discharge of the cake is obtained by admitting water to the interior of the leaf by means of a funnel conveniently placed, the air escaping through a pipe provided with a ball valve, also conveniently placed. As the water rises in the leaf the entire surface of the filter medium is saturated, and when the inside pressure has become substantially equal to the outside pressure the flaps at the bottom open, and allow communication between the external and internal water, thus equalising the external pressure. The cakes will then be rapidly dislodged.

These leaves can be applied to any of the present stationary vacuum filters.

(C.) 198/14. Charles Butters. Improvements in and relating to filtering processes and apparatus. 7.5.14.

This application refers to a process of filtering slime, and an improved filter leaf. The process is based upon the effects produced during filtering by the adoption of a new construction of filter leaf by means of which it is possible to equalise the external and internal liquid pressure while the said leaf is submerged, thus facilitating the removal of the slime cake formed on the leaf.

(C.) 199/14. Charles Butters. Improvements in and relating to filtering processes and apparatus. 7.5.14.

The above application refers to the construction of a filter leaf for the purpose of filtering slime pulp, which will permit an easy cake dislodgment. The moisture of the cake is withdrawn as completely and rapidly as possible, by maintaining the maximum possible vacuum within the leaf for a time while the leaf with a cake thereon is suspended in air. After the cake is thus specially dried water is flooded down on to the cake while the vacuum is at the same time relieved, the cake being thereby dislodged from the filtering surface.

The filter cloths used in this leaf are comparatively light: the filler of the leaf is preferably coco-mat. The superimposed filter cloth is open across the bottom, and extends a short distance below the frame of the leaf. The open portions will close automatically when vacuum is turned on the leaf.

(C.) 219/14. Robert Farmer. Improvements in controlling means for winding, hoisting, hauling and similar engines, electric hoists or the like. 15.5.14.

This application is for the employment of a compound lever whereby the throttle valve or its equivalent, and the valves or their equivalent which control the brake gear, are simultaneously operated. These levers can also be operated independently as well. The application also provides for the closing of the throttle valve and the putting on of the brakes should anything happen to the driver during hoisting.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 179/14. Edward James Ryan (1), Thomas Quig Ryan (2). Improvements in means for enclosing lamp and like flames. 27.4.14.

(P.) 180/14. William Arthur Caldecott. Improvements in the treatment of acid or corrosive water. 27.4.14.

(C.) 182/14. Francis Henry le Sueur. A means for opening and closing gates and the like from a distance. 28.4.14

(P.) 183/14. William Jones (1), Cyril Heyland Browne (2). Improvements in cam shafts and the like. 28.4.14.

(P.) 184/14. Joseph Patrick Cunniff. Improvements in safety catches for use on skips, cages working in vertical shafts, lifts and the like. 29.4.14.

(P.) 186/14. Stephen Masters. Improvements in means for indicating movements of motor cars and the like. 30.4.14.

(C.) 187/14. Thomas Coleman (1), Frank Coleman (2). Improvements in the method and means for drying and heating sand, stone grit or other materials. 30.4.14.

(C.) 189/14. Thomas Frederick Wall. Improvements relating to electrically operated drills and

other reciprocating apparatus of the percussion type. 30.4.14.

(C.) 191/14. Josef Carl Steinebach. Device for automatically stopping electrically driven hoists when the speed deviates from the correct value during retardation. 30.4.14.

(P.) 191/14. James Howard Vivian. Improvements in means for supplying fluid to drill holes. 1.5.14.

(P.) 192/14. Frederick Charles William Ingle. Improvements in shovels and the like. 1.5.14.

(P.) 194/14. Henry Tyzzer. An improved device for ramming concrete, earthwork and the like. 4.5.14.

(C.) 195/14. Matthias Pier. Improvements in or relating to apparatus for carrying out reactions between hydrogen and other gases. 7.5.14.

(C.) 196/14. Guglielmo Marconi (1), Marconi's Wireless Telegraph Company, Limited (2). Improvements in transmitters for wireless telegraphy. 7.5.14.

(C.) 197/14. William Armor Stedman. Improvements in and relating to filtering apparatus. 7.5.14.

(C.) 198, 199, 200/14. Charles Butters. Improvements in and relating to filtering processes and apparatus. 7.5.14.

(C.) 201/14. George Moore. Improvements in processes for treating metal bearing material. 7.5.14.

(C.) 202/14. Walter Elwin Trent. Separating apparatus. 7.5.14.

(P.) 203/14. Walter Albert Sherill. Telescopic machine bar. 8.5.14.

(C.) 204/14. James Whytock. Improvements in elastic or resilient wheels. 8.5.14.

(P.) 209/14. Alexander Runciman Foyer. A new or improved shower bath apparatus. 9.5.14.

(P.) 210/14. Sigismund Rudolf James Maynard. Improvements in methods and apparatus for coal cutting. 11.5.14.

(C.) 214/14. Henry Thomas Durant. Improvements in or relating to the extraction of zinc from its ores or products. 14.5.14.

(C.) 215/14. Lewis Coolidge Bayles. Improvements in fluid operated percussive tools. 14.5.14.

(C.) 216/14. Friedrich Mausser. Process for the absorption of dilute nitrogenous gases by water. 14.5.14.

(P.) 217/14. Henry Candy (1), William Young (2). Improvements relating to the construction and arrangement of guides for stems used in milling batteries. 15.5.14.

(C.) 219/14. Robert Farmer. Improvements in controlling means for winding, hoisting, hauling and similar engines, electric hoists or the like. 15.5.14.

(P.) 220/14. Henry Samuel Potter. Improvements in devices for supplying fluid to rock drilling tools. 16.5.14.

(C.) 221/14. William Henry Gordon. Improvements relating to stamp heads for stamper bars. 16.5.14.

(P.) 222/14. Lancelot Ussher (1), Armour Hall (2). Improvements in compressing air. 18.5.14.

(P.) 223/14. Horatio Dunn. Improvements in the method of igniting safety fuses as used in mines, quarries and the like. 18.5.14.

(C.) 228/14. Alb Nurnberg. A Harz rock drill. 26.5.14.

(P.) 229/14. Seguin Fisher. Improvements in water connections for hollow rock drills. 27.5.14.

(P.) 230/14. James McKenzie Martyn (1), William George Holford (2). Improvements relating to baling tanks. 27.5.14.

(C.) 234/14. Daniel Shaw Wangh. Improvements in drilling machines. 28.5.14.

(P.) 235/14. R. van Powys. Improved chain drive for mechanical haulages, etc. 30.5.14.

(P.) 236/14. Daniel Corlett. A new type of rope grip. 3.6.14.

(C.) 237/14. Guglielmo Marconi (1), William Snowley Entwistle (2), Marconi's Wireless Telegraph Company, Limited (3). Improvements in transformers for high frequency currents. 4.6.14.

(C.) 238/14. Clemens Graaff. Chemical fire extinguishers. 4.6.14.

(C.) 239/14. Samuel Michael Schenkein (1), Otto Paul Richard Lehmann (2). Gem polishing machine. 4.6.14.

(P.) 244/14. Henry Lumley Harland. Improvements in stamp guides. 6.6.14.

(P.) 245/14. Henry Lumley Harland. Improvements in securing mortar box screens. 6.6.14.

(C.) 247/14. Edgar Arthur Ashcroft. Improvements in the production of alkali metal alloys by the electrolysis of alkali hydroxide in the molten state, and in the obtainment therefrom of alkali metals and alkali metal compounds. 8.6.14.

(C.) 248/14. Improvements in the electrolytic manufacture of alkali metals. 8.6.14.

(P.) 249/14. Arthur Edward Dowling (1), Arthur Edward Reynolds (2). Improvements in linings for the ends of tube mills. 8.6.14.

(P.) 251/14. Charles Hansen (1), Kinnear Reid Christie (2). Improvements in water supplying attachments for rock drills. 9.6.14.

(C.) 252/14. Herbert Frood. Improvements in and relating to brake blocks and the like. 11.6.14.

(C.) 253/14. George Wilbur Arnold. Improvements in ore concentrating machines. 11.6.14.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

BASKETT, E. G., *c/o* Rietfontein; Prestin Mines, Block A., via Sekondi, Gold Coast Colony, West Africa.

BELL, H. D., *to* 10, Dam Street, District Township, Krugersdorp.

DEGENHARDT, W. R., *c/o* Australia; *c/o* Bank of Adelaide, 11, Leadenhall Street, London, E.C.

JAMES, W. H. T., *to* Manor Lodge, 4, Grove End Road, London, N.W.

KING, A., *c/o* Nigel; Knights Deep, Ltd., P. O. Box 113, Germiston.

MACKAY, G. J., *to* 126, Albert Street, Ottawa, Canada.

MELVILL, G., *c/o* Cleveland; Village Deep, Ltd., P. O. Box 1064, Johannesburg.

MILLIGAN, J. T., *c/o* Johannesburg; Machavie G. M. Co., Ltd., Potchefstroom.

MURRAY, M. T., *c/o* Johannesburg; c/o T. J. Nicholson, Esq., Sunnymeade, Station Road, Hessle, near Hull, England.

NEAL, WALTER, *c/o* Mexico; 2233, Derby Street, Berkeley, California, U.S.A.

PARTITT, W., *c/o* Jeppes; Langlaagte Estate and G. M. Co., P. O. Box 98, Langlaagte.

PRICE, W. S. V., *to* P. O. Box 11, Maraisburg.

TOM, L., *to* P. O. Box 6034, Johannesburg.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. XV. AUGUST, 1914. No. 2.

Proceedings
AT
Ordinary General Meeting, August 15th, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines and Technology Building, Johannesburg, on Saturday, August 15th, Prof. G. H. Stanley (President) in the Chair. There were also present:

28 Members: Prof. J. A. Wilkinson, Messrs. E. Pam, J. E. Thomas, R. A. Barry, Prof. J. S. Cellier, J. Gray, H. Meyer, John Watson, H. A. White, R. G. Bevington, W. R. Dowling (Members of Council), H. R. Adam, A. S. Allam, S. Beaton, W. Beaver, H. D. Bell, J. Chilton, J. M. Dixon, C. J. Gray, B. J. Hastings, G. Melvill, J. M. Neill, S. Newton, J. F. Pyles, D. F. Scott, W. E. Thorpe, and E. M. Weston.

11 Associates and Students: Messrs. N. S. Arnot, M. J. Dennehy, O. A. Gerber, J. Gibson, W. Human, A. King, S. E. Willows-Munro, D. Nicholas, F. J. Pooler, H. Rusden, and P. Scatterty.

9 Visitors, and Fred Rowland (Secretary)

MINUTES.

The Minutes of the Annual General Meeting were confirmed.

FINANCIAL STATEMENT.

The President: I will now ask the Secretary to read the Financial Statement of the Society for the past year, which, owing to the Secretary's illness, had to be left over from the Annual Meeting. The Statement, printed on the next page, was then read.

The President: I am afraid there is not very much cause for congratulation on the Balance Sheet which has just been read, but it was a very exceptional year, and, on the whole, I think we can be glad it was no worse. A good deal of the apparent loss was due to writing off on the stock of Transactions. That will not occur again; so, with certain economies which we propose to effect in this year, the expenditure should at least be met. I would like to move the adoption of the Financial Statement.

Prof J. A. Wilkinson (Vice-President) seconded, and the Financial Statement was adopted.

NEW MEMBERS.

Messrs. Jas. Gray and F. J. Pooler were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected as follows:—

DOBBIE, THOMAS, 70, Becker Street, Yeoville, Johannesburg. Metallurgist.

MARTIN, HENRY ROSE, 73, Browning Street, Fairview. Cyanide Manager.

NETTLETON, STANLEY, P. O. Box 2701, Johannesburg. Mining Engineer.

The Secretary announced that since the last meeting of the Society the following Associates and Students had been admitted by the Council:—

As Associates—

BANKS, ROBERT, Westralia Mine, Mount Morgan, Western Australia. Metallurgist.

GOURLEY, JOHN M., Brakpan Mines, Ltd., P. O. Box 3, Brakpan. Ambulance Officer.

KEAY, ALEXANDER RAMSAY, Brakpan Mines, Ltd., P. O. Box 3, Brakpan. Assistant Assayer.

As Students—

CAMPBELL, JOHN JAMES, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

VIEUX, JEAN LE, S.A. School of Mines and Technology, P. O. Box 3911, Johannesburg.

GENERAL BUSINESS.

THE EUROPEAN WAR.

The President: It is only right that at this meeting some reference should be made to the war in which Great Britain and the Colonies are unfortunately engaged. You know

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 1st June, 1914.

Dr.

1912-13. £ s. d.	EXPENDITURE	1913-14. £ s. d.	£ s. d.
	To CHARGES (including Postages, Bank Charges, Insurance, etc.)	148 6 4	
[illegible]	,, PRINTING AND STATIONERY	198 15 2	
[illegible]	,, RENT, LIGHTING AND TELEPHONE	39 14 0	
[illegible]	,, SALARIES AND BONUS	627 0 0	
[illegible]	,, DEPRECIATION ON FURNITURE	10 0 0	
[illegible]	,, BALANCE COST OF TRANSACTIONS Written off	136 13 9	
[illegible]			1,160 9 3
	GRANTS, Seymour Memorial Library	50 0 0	
	Kr. Memorial Fund		
	Cons. S. C. Memorial Fund		
[illegible]	Prizes, School of Mines & Society	20 5 0	
			70 5 0
	,, LOSS ON ANNUAL DINNER		16 17 6
	,, SOCIETY'S JOURNAL		
[illegible]	Cost of Reporting, Printing, etc.	1,277 10 0	
[illegible]	*less* Received for Sub's. and Advts. after deducting commissions on Advts. and Collections	474 16 9	
			802 13 3
£2,1[illegible] 0 6			£2,050 5 0

Cr.

1912-13. £ s. d.	REVENUE.	1913-14. £ s. d.	£ s. d.
	By SUBSCRIPTIONS		
[illegible]	Members, current year	1,037 8 0	
	do. arrears collected	86 1 0	
	Associates, current year ...	438 12 9	
	do. arrears collected	65 17 9	
[illegible]	Students	6 16 6	
			1,634 16 0
[illegible]			
[illegible]	,, ENTRANCE FEES		30 9 0
[illegible]	,, INTEREST ON FIXED DEPOSITS		23 17 3
[illegible]	,, BALANCE, deficit on year's working		361 2 9
£2,1[illegible] 0 6			£2,050 5 0

BALANCE SHEET, as at 1st June, 1914.

Dr.

1912-13. £ s. d.	LIABILITIES.	1913-14. £ s. d.	£ s. d.
[illegible]	To SUNDRY CREDITORS	420 12 5	
[illegible]	,, SUBSCRIPTIONS paid in advance	29 8 0	
	MINING EXHIBITION FUND	53 18 0	
			503 18 5
[illegible]	,, REVENUE AND EXPENDITURE ACCOUNT		
(1,011 0 11	Balance at June 1, 1913	940 5 4	
70 17 [illegible]	*Less* deficit as per Revenue and Expenditure Account, June 1, 1914 ...	361 2 9	
			579 2 7
940 5 4			
£1,202 7 4			£1,083 1 0

Cr.

1912-13. £ s. d.	ASSETS.	1913-14. £ s. d.	£ s. d.
	By CASH AT STANDARD BANK		
(12 18 [illegible]	Johannesburg ...	88 3 10	
63 1[illegible] 11	London ...	26 13 8	
66 0 6)	,, On Hand, Johannesburg ...	87 1 0	
142 14 8			201 18 6
405 6 8	,, Cash on Fixed Deposit with accrued Interest	405 13 4	
326 19 1)	,, Cash in Post Office Savings Bank with accrued interest	102 9 8	
732 5 9			508 3 0
90 0 0	,, OFFICE FURNITURE ...		80 0 0
(64 9 8	,, SUNDRY DEBTORS ...	295 2 3	
2 2 9)	*less* Provision for Doubtful Debts ...	2 2 9	
62 6 11			292 19 6
£1,202 7 4			£1,083 1 0

We have examined the above Balance Sheet and accompanying Revenue and Expenditure Account with the books and vouchers of the Society, and have obtained all the information and explanations we have required. In our opinion such Balance Sheet is properly drawn up so as to exhibit a true and correct view of the affairs of the Society according to the best of our information and the explanations given to us, and as shown by the Books of the Society.

Sgd. ALEX. AIKEN and CARTER, *Hon. Auditors*

JOHANNESBURG, 13th AUGUST, 1914.

(Signed) JAMES LITTLEJOHN, *Hon. Treasurer*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

that the Imperial Government is asking that, as far as possible, everything shall go on as nearly as possible in a normal manner. We read in the papers that the motto in Britain is: "Business as usual"; and, so far as we can, with the assistance of members, this Society intends to act up to that motto. At the same time, I think we should assure the Government that any assistance which this Society as a Society, or as individuals, can give to the Government at this time will be rendered with all the willingness possible. (Applause.)

VISIT TO DE BEERS.

The President: There is one matter that we might attend to now, and that is the excursion to De Beers Mines at Kimberley. It will be remembered that we had a very cordial invitation, and I am only sorry that so few availed themselves of it. The few who did go had a very good time indeed; and I think that this meeting should pass a hearty vote of thanks to the management and Directors of De Beers for their invitation and for the facilities afforded during the visit. This was passed with acclamation.

BIRTHDAY HONOURS.

The President: There is another point which, I think, will give the members of this Society considerable pleasure. In the last list of Birthday Honours two of our honorary members—Mr. R. A. S. Redmayne and Dr. T. K. Rose—received the honour of Knighthood. Dr. Rose (who is the Chief Assayer of the Royal Mint) in particular, has been extremely kind to many of our members visiting England. He has done all in his power to make their visit enjoyable and to assist them in every way during their visits, and, further, he has taken a great deal of trouble in seeing that the medals which the Society has presented at various times have been satisfactorily executed in England. I propose, therefore, that a very hearty vote of congratulation be passed to Sir T. Kirke Rose and Sir R. A. S. Redmayne. This was agreed to unanimously.

TUBE MILL ENDS.

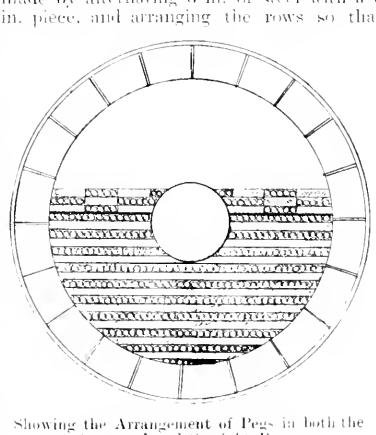
Mr. J. F. Pyles *(Member):* On mines where worn-out Osborn liner bars are carefully conserved for reinforcing other types of tube-mill liners, the following description of a wood and steel end liner may prove of interest.

The liner is built round a casting or sleeve, which should be 1½ in. thick. The annular outer portion is built of 6 in. chert blocks, the remaining space being filled with alternate layers of scrap steel cut to 6 in. lengths and 1 in. pine boards.

The procedure in building the liner is as follows:—First place the casting or sleeve in position, then build in the outer layer of chert blocks, which have been previously shaped to fit the mill. These blocks are placed in the mill with Portland or magnesite cement and keyed in position. A layer of 6 in. pegs is then put in with their axes parallel to the sleeve and their sides touching each other, and embedded in mortar. Next a board 1 in. thick and 6 in. wide, touching the chert circle at each end, is laid on the pegs and tapped into place. Mortar is placed on this board and the next row of pegs embedded in it, and so on until the liner is completed. The work of building this end liner will occupy one mason for half a day.

The end has the following points to recommend it. It is inexpensive, both as to material and labour, and is quickly and easily installed. Owing to the wood absorbing the moisture from the cement, the end dries in a much shorter time than if only steel or chert were used. The swelling of the wood wedges the whole so tightly that all danger of the end, or any part of it, coming out is obviated.

The life of the outlet end will be materially increased by staggering the wood and iron. This may be done by cutting the 1 in. board into pieces 6 in. square. Each row is then made by alternating 6 in. of steel with a 6 in. piece, and arranging the rows so that



Showing the Arrangement of Pegs in both the Staggered and Straight Rows.

each piece rests on and is covered by steel. The pieces are placed with the grain of the wood parallel to the pegs.

The writer does not claim that the idea of using wood and steel for tube-mill liners is in itself original, but, to the best of his knowledge, its practical utility has not been proved previously with regard to tube-mill end liners.

The adoption of this end liner was suggested, firstly, by the necessity of installing one that would not fall out, and, secondly, by the importance of economising steel.

The President: Notes of this sort are just the thing we want to enliven the Journal, and also to be of practical use to other workers in the same field.

Mr. J. E. Thomas *(Vice-President):* occupied the chair while Prof. G. H. Stanley delivered his Address.

PRESIDENTIAL ADDRESS.

By Prof. G. H. Stanley, A.R.S.M., F.I.C., M.Inst.M.M.

I have already expressed my very high appreciation of the honour which this Society has conferred upon me by electing me President; but during the two months which have elapsed since my election I have at times almost wished that the choice had fallen upon someone else. The reason for this, of course, is that an inaugural address is expected, and it is indeed a difficult matter to select a suitable subject, and, having done so, to do even scant justice to it in the time available for preparation. The events of the last fortnight, too, have not been conducive to that mental atmosphere in which such a task should be performed.

A general survey of the history of our Society and of recent progress in the allied sciences with which we deal, or any one of them, to be effective would occupy too much time; and if kept within reasonable limits in that connection, would be merely reiterating what must be already familiar to you, or at any rate to a large majority. I therefore propose to devote my address firstly, to a consideration of the laws of crushing and, secondly, to advocacy of a subject which I have very much at heart, namely, research work in connection with the mineral industry of this country.

THE LAWS OF CRUSHING.

During recent years there has been a very considerable amount of controversy between the advocates respectively of the "laws" of Rittinger and Kick.

Rittinger, on page 20, Chapter I. ("Das Zerkleinern) of his book, *Aufbereitungskunde*, published in 1867, said:- "*Die Arbeitsgrossen stehen daher nahezu im geraden Verhältnisse zum Zerkleinerungsgrade oder im verkehrten Verhältnisse zum Zerkleinerungsquotienten.*"

And also: -" *Die zum Zerkleinern erforderliche Kraft steht mit dem Oberflächenzuwachse im geraden Verhältnisse;* denn da aus jeder Theilung zwei gleiche Bruchflachen hervorgehen, so steht der Oberflachenzuwachs im geraden Verhältnisse mit der zahl der stattgefundenen Theilungen, also auch mit der dazu erforderlichen Arbeit."

I translate these statements as follows: "The work (in different cases) is nearly proportional to the degree of reduction or inversely proportional to the reduction quotients," and "The energy necessary for reduction is directly proportional to the surface increase; for since from each cleavage two equal fracture planes result, the surface increase is directly proportional to the number of cleavages taking place, and so also is the work necessary thereto."

Rittinger uses the term "Reduction-quotient" to mean the ratio of the edge of the resulting cube to the edge of the original cube, *i.e.*, $\frac{1}{2}$, $\frac{1}{3}$, or $\frac{1}{n}$ size, and the first statement is therefore equivalent to saying that "the work is inversely proportional to the reduced diameter," or "proportional to the reciprocals of the diameters crushed to," either of which forms carries a more obvious meaning than "Reduction-quotient."

In the second statement the word "Kraft" is employed, and this may mean either "power," "energy," or "force," but since in his discussion and proof he uses "Ft-lbs. of work," and in the first statement "Arbeitsgrossen" (amount of work). I do not think there is any room for doubt as to his real meaning, *i.e.*, that he meant power or energy, and not force.

Kick's Law, as stated by Mr. Stadler (and in this case I regret I have not seen

the original publication*) is that *The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volume or weights of those bodies.*" (Rittinger's bodies are not geometrically similar, the first fracture plane producing a flat slab.)

Much has been written, and by notable authorities, in support of one or other of these theories, accompanied by as much experimental proof on the one behalf as on the other, and it is consequently a matter for small wonder that most people view the whole position as one of considerable obscurity.

At first, comparison is somewhat difficult, because the two laws do not deal with the same set of circumstances. Rittinger starts with a piece of ore of a certain size and compares the work done in breaking it into pieces of other sizes, while Kick, on the other hand, compares the work required to break, in a similar manner, pieces of ore of different sizes. In short, Rittinger refers to a constant total quantity, and Kick to a variable quantity. It is therefore necessary, for comparison, to deduce from Kick's law an expression of the same nature as Rittinger's, dealing with a constant total quantity and referring the work necessary for reduction to the linear dimensions of the particles produced, which, since this is commonly defined as a result of sifting through screens of certain apertures, or by other direct linear measurement, seems to be most useful for practical employment. Such an expression I propose to bring before you to-night.

Die Arbeitsgröfsen stehen daher nahezu im geraden Verhältnisse zum Zerkleinerungsgrade oder im verkehrten Verhältnisse der Zerkleinerungsquotienten.

Erzeugt man z. B. aus zwei einzölligen Steinwürfeln kleinere Würfel, und zwar aus dem einen solche von $\frac{1}{2}$, aus dem andern von $\frac{1}{20}$ Zoll Seitenkanten-Länge, so wird zur Theilung im zweiten Falle nahe viermal so viel Arbeit benöthiget als im ersten Falle.

In der Praxis erfolgt zwar die Zerkleinerung nicht in regelmäfsigen, sondern in ganz unförmlichen Fragmenten, und es entfallen aufser den Stücken vom gewünschten Zerkleinerungsgrade noch eine Menge anderer von einem viel gröfseren Zerkleinerungsgrade; der zu einem gewissen Zerkleinerungsgrade erforderliche Arbeitsaufwand wird daher absolut genommen gröfser sein, als bei einer regelmäfsigen Zerkleinerung, aber verhältnifsmäfsig wird in zwei verschiedenen Zerkleinerungsfällen der Arbeitsaufwand gleichfalls nahezu dem Zerkleinerungsgrade proportional sein.

Man kann auch sagen:

Die zum Zerkleinern erforderliche Kraft steht mit dem Oberflächenzuwachse in geradem Verhältnisse; denn da aus jeder Theilung zwei gleiche Bruchflächen hervorgehen, so steht der Oberflächenzuwachs im geraden Verhältnisse mit der Zahl der stattgefundenen Theilungen, also auch mit der dazu erforderlichen Arbeit.

photographic reproduction of portion of p. 20 "Aufbereitungskunde."

Rittinger's proof, adopted also by Richards in his monumental work on "Ore-dressing," is, of course, familiar to all who have studied the subject, but it may be well for the sake of completeness to run over it again here.

If we consider a cube of rock of edge s, and adopt Rittinger's assumption that A ft. lbs. of work are required to rupture it along a plane parallel to two opposite faces. Then, according to Rittinger, $3A$ ft. lbs. are required to break it along 3 planes at right angles to one another to form 8 cubes of $\frac{s}{2}$ edge, and similarly

*) "Das Gesetz der proportionalen Widerstande," by F. Kick. Published in Leipzig, 1885, by Arthur Felix.

6A ft. lb. to break along 6 planes to form 27 cubes of $\frac{s}{3}$ edge.

9A ft. lb. to break along 9 planes to form 64 cubes of $\frac{s}{4}$ edge.

21A ft. lb. to break along 21 planes to form 512 cubes of $\frac{s}{8}$ edge.

$3(n-1)$ ft. lb. to break along $3(n-1)$ planes to form n^3 cubes of $\frac{s}{n}$ edge.

$3(m-1)$ ft. lb. to break along $3(m-1)$ planes to form m^3 cubes of $\frac{s}{m}$ edge.

The ratio of work required to break to $\frac{s}{n}$ or $\frac{s}{m}$ cubes is therefore $\frac{3(n-1)A}{3(m-1)A} = \frac{n-1}{m-1}$, or neglecting 1 when n and m are assumed to be large, $\frac{n}{m}$, *i.e.*, the work required is proportional to the reciprocals of the reduced diameters, or diameters crushed to.

The increase of surface involved with 3 planes $6s^2$

" " " 9 " $18s^2$

" " " 21 " $12s^2$

the edges being respectively $\frac{s}{2}$, $\frac{s}{4}$ and $\frac{s}{8}$, and work required 3A, 9A, 21A, the work is obviously proportional to increase of surface.

Rittinger is clearly comparing the work done in reducing from an initial size to various other sizes, as already stated, and bases his argument on the amount of work, A ft. lb., required to fracture along one plane, and assumes this to be constant per unit area of every other plane, and therefore proportional simply to the size of the plane, or the linear dimension squared in the case of a square.

Now, how is this "A ft. lb." arrived at? When a body is compressed as a result of the application of force, obviously work is done and power* is consumed, the magnitude of the work being, of course, obtained as the product of the average force and the distance through which it acts. As the force gradually increases the body becomes more and more compressed, and ultimately yields and breaks, force having been applied through a distance which may be represented by d, Fig. 1. The force required to break varies with the material, according to the different strengths, and is known in many cases as a result of experimental determination, and expressed as pounds per unit area (square inch), which is called the compressive strength: let it be F.

The total force required to break a cube of edge s will then be Fs^2, and the work done, A, will equal $\frac{Fs^2}{2} \times d$.

The absolute value of F may be unknown in the case of an ore, but obviously A will be proportional to s^2d.

The deformation or strain produced by a given stress is constant (or nearly enough so for the present discussion) and has a numerical value which is proportional in this case to s, or, in other words, d is a function of s, and since A is proportional to s^2d, it is therefore proportional to s^3, and this, in short, is Kick's law! Consequently, Rittinger's assumption appears to be incorrect, and Kick's appears to be correct.

Other than that the $\frac{\text{stress}}{\text{strain}}$ is constant up to the point of rupture, which is apparently nearly true for such brittle bodies as rocks, no assumption is made in the foregoing which is not amply confirmed by experimental evidence, as will be seen by reference to text-books on the subject of "Strength of Materials."*

The effect of this on Rittinger's argument and conclusions may now be examined.

Assume, as before, that A ft. lbs. are required for the first (say vertical) fracture; then to fracture also vertically—one of the halves thus obtained, supposing the force to be applied in the same direction, will require the application of half the previous force, but as it will be applied to half the area the intensity of stress will be the same, and d, therefore, the same, since the depth remains the same—s—as before. Consequently work done at fracture will be half that required for the whole cube, or, since there are two halves, the work done in producing the second fracture plane at right angles to the first will also be A ft. lbs. There are now four quarters to deal with, and each of these can also be shown to require one-fourth as much work as for producing the first plane of fracture (the piece supposed lying on a side, for d will be as in the second case, but d only half since the depth is $\frac{s}{2}$), the four together consequently requiring A ft. lbs. Up to this point, therefore, Rittinger is correct.

* Power in the popular sense as equivalent to work or energy.

Now consider one of the resulting 8 cubes of half the original edge, and therefore one-eighth of the original volume. And consider the work required to break it into halves, a procedure which has only to be repeated on *four* of them to produce a plane of the same area as one of the original three planes of fracture which produced the eight cubes. The intensity of stress at rupture remains as before, therefore since the area of the face to which the pressure is applied is one-fourth that of an original cube face, the total force (stress) is one-fourth, but the length of edge is $\frac{s}{2}$, and consequently the deformation (strain) will be $\frac{d}{2}$ and the work done will be $\frac{Fs^2}{4\times 2}\times\frac{d}{2}=\frac{Fs^2d}{16}$, and this multiplied by four gives the work required to produce a fracture plane of the original area s^2 under the new conditions as $\frac{Fs^2d}{4}$, or instead of being only A ft. lbs. it is only half as much! The work required is *not*, therefore, proportional to the increase of surface, though the force is.

According to Kick's law, a cube of edge s, and volume therefore s^3, will require an amount of work which is a function of s^3 to split it into two or halve it: each of these halves will require half as much work to again halve them, and so on: the work required for the various sizes is tabulated and by addition of these, taking into account the number of pieces to be broken, the total work to reduce to pieces of any fractional size of the original can be found. For instance, the total work required to break a cube of edge s into cubes

Of Edge.	Is Proportional to.	Total
$\frac{s}{2}$	$s^3+\frac{2s^3}{2}+\frac{4s^3}{4}=$	$3s^3$
similarly† $\frac{s}{4}$	$3s^3$ (from *) $+\frac{8s^3}{8}+\frac{16s^3}{16}+\frac{32s^3}{32}=$	$6s^3$
„ $\frac{s}{8}$	$6s^3$ (from †) $+\frac{64s^3}{64}+\frac{128s^3}{128}+\frac{256s^3}{256}=$	$9s^3$
„ $\frac{s}{16}$		$12s^3$

the total values being also entered in the table, together with, for comparison, the increase of surface exposed.

An inspection of the table will show that the work required to break from one size to another is not proportional to the ratio of the original to the final diameter, *i.e.*, reciprocal of diameter crushed to, nor to the increase of surface. It is, however, obviously proportional to the logarithm to the base 2 of the reciprocal of the diameter crushed to, and hence, of course, to the common logarithm also. The "law" may therefore be stated, following the form used by Rittinger, as follows: "The work is proportional to the logarithm of the reduction quotient."

I should, however, prefer to amplify this somewhat, *i.e.*, *The work required to reduce a given weight of rock from one size to another is proportional to the log of the* $\frac{initial}{final}$ *diameter.* I do not think this has been stated previously, in either form. It does not matter how many pieces there may be of the original size in the given weight, for an inspection of the table also shows that the work required to break the weight proportional to s^3 in any sized pieces, to pieces $\frac{1}{r^n}$ that size is constant. It will also be noted that from the first two figures of resulting increase of surface it appears as if the work would also be proportional to increase of surface, though based on Kick! and might be taken to justify Rittinger, but as the comparison proceeds the figures become more and more divergent. Practically, it shows that provided the degree of reduction is small, experimental results might almost be in accordance with the Rittinger theory, and this has undoubtedly given much support in the past. As the size of particles diminishes, however, the increase of surface is much more rapid than the work required to produce it, and as stated by Richards, this was experimentally found to be the case by Von Reytt; and though the latter's object was apparently to prove the correctness of Rittinger's law, his results, on the contrary, tend to confirm the deductions I have made from Kick's. My colleague, Mr. Le May, has been good enough to put all this into proper mathematical form as follows:—

Let W = work done at each reduction ($= ks^3$)

n = number or reductions.

$\frac{1}{r}$ = ratio in which linear dimensions are reduced at each reduction.

After n reductions, linear dimensions are reduced in ratio $\frac{1}{r^n}$

Size of Piece.	Volume and Work Required to Split. x	Number of Pieces To be Broken	Resulting	Total Work Done. x	Increase of Surface resulting	Ratio, Original, Final, Diameter.	Logarithm Original Final Dia. to Base 2	Size of Resulting Pieces.
$s \times s \times s$	s^3	1	2	s^3	$2s^2$			$s \times s \times \frac{s}{2}$
$s \times s \times \frac{s}{2}$	$\frac{s^3}{2}$	2	4 (2^2)	$2s^3$				$s \times \frac{s}{2} \times \frac{s}{2}$
$s \times \frac{s}{2} \times \frac{s}{2}$	$\frac{s^3}{4}$	4	8 (2^3)	$3s^3$	$6s^2$	2	1	$\frac{s}{2} \times \frac{s}{2} \times \frac{s}{2}$
$\frac{s}{2} \times \frac{s}{2} \times \frac{s}{2}$	$\frac{s^3}{8}$	8	16 (2^4)	$4s^3$				$\frac{s}{2} \times \frac{s}{2} \times \frac{s}{4}$
$\frac{s}{2} \times \frac{s}{2} \times \frac{s}{4}$	$\frac{s^3}{16}$	16	32 (2^5)	$5s^3$				$\frac{s}{2} \times \frac{s}{4} \times \frac{s}{4}$
$\frac{s}{2} \times \frac{s}{4} \times \frac{s}{4}$	$\frac{s^3}{32}$	32	64 (2^6)	$6s^3$	$18s^2$	1	2	$\frac{s}{4} \times \frac{s}{4} \times \frac{s}{4}$
$\frac{s}{4} \times \frac{s}{4} \times \frac{s}{4}$	$\frac{s^3}{64}$	64	128 (2^7)	$7s^3$				$\frac{s}{4} \times \frac{s}{4} \times \frac{s}{8}$
$\frac{s}{4} \times \frac{s}{8} \times \frac{s}{8}$	$\frac{s^3}{256}$	256	512 (2^9)	$9s^3$	$42s^2$	8	3	$\frac{s}{8} \times \frac{s}{8} \times \frac{s}{8}$
$\frac{s}{8} \times \frac{s}{16} \times \frac{s}{16}$	$\frac{s^3}{2048}$	2048	4096 (2^{12})	$12s^3$	$90s^2$	16	4	$\frac{s}{16} \times \frac{s}{16} \times \frac{s}{16}$
	$\frac{s^3}{2^{3m}-1}$	$2^{3m}-1$	2^{3m}	$3ms^3$		2^m	m	$\frac{s}{2^m} \times \frac{s}{2^m} \times \frac{s}{2^m}$
	$\frac{s^3}{2^{3n}-1}$	$2^{3n}-1$	2^{3n}	$3ns^3$		2^n	n	$\frac{s}{2^n} \times \frac{s}{2^n} \times \frac{s}{2^n}$

* $1/s$, reciprocal of diameter crushed to.

The work is proportional to the $\log \frac{\text{initial}}{\text{final}}$ diameters.

The total surface increases in G.P., and so, very nearly, does the increased surface (as the number of particles increases).

The total work increases in A.P.

Increase of surface is therefore much more rapid than that of the work required to produce it.

Work required is also proportional to the log. of the number of pieces resulting.

$\therefore \frac{\text{Initial diameter}}{\text{Final diameter}} = r^n \therefore \log_r \frac{\text{Initial diameter}}{\text{Final diameter}} = n$

Total work in n reductions $= nW = nks^2$.

i.e., Total work is proportional to n.

$\therefore$ ——————— $\log_r \frac{\text{Initial diameter}}{\text{Final diameter}}$

and since r is assumed constant, total work is proportional to $\log_{10} \frac{\text{Initial diameter}}{\text{Final diameter}}$

In an endeavour to prove one or the other law experimentally, tests were made by dropping flint pebbles from various heights, as suggested by Mr. H. A. White, to ascertain the heights of drop which would suffice to break them. Since the kinetic energy in a falling body is given by the expression $\frac{MV^2}{2}$ and this equals the weight × the height of drop, and the power required to break is proportional, according to Kick, to M (or D^3), the velocity^2 and also the height of fall required to break should be constant for all sizes, *i.e.*, K × weight = weight × height $\therefore$ height is constant. K being a constant depending on the breaking strength. According to Rittinger, the pebbles should require energy proportional to

D^2, *i.e.*, to $\frac{\text{weight}}{D}$, *i.e.*, $K \times \frac{\text{weight}}{D}$ = weight ×

height, whence height $= \frac{K}{D}$

or, the height should be inversely proportional to the diameter, and, therefore, the larger the pebble the less fall it should require to break it.

About 30 pebbles were tested, from about one inch to four inches diameter, and though they certainly did not all require the same height of drop to cause fracture, the small ones did not seem to require a markedly larger drop than those of large size. All broke at from 20 to 40 ft., thus at

Feet Drop	The Weights of Pebbles Broken were in Grammes.
20	125, 165, 180, 205, 300.
25	70, 130, 145, 165, 190, 270, 280, 330 and 350.
30	105, 115, 160, 290.
35	70, 85, 600.
40	80, 115, 120, 180, 670.

All the pebbles which broke at the falls stated had previously failed to break at lesser falls.

This certainly does not support Rittinger, and illustrates best perhaps the variable difficulty of breaking pieces of the same material of about the same sizes, doubtless due to small differences of structure and presence of inequalities, such as cracks and flaws, although care was taken to select them as uniform in appearance as possible.

Objection may be taken, on various grounds, to the deductions made, *e.g.*, that pieces dealt with are almost certainly not cubes, but of various irregular shapes, that they are not broken by halving, that breaking is effected not by crushing but by shearing, that in many cases breaking is effected by impact rather than by a slow or gradual application of force, and so forth. Some of these objections may next therefore be examined. I have already referred to the assistance rendered in a consideration of the laws of crushing—in the ore-treatment sense—by that branch of engineering science known as "Strength of Materials." The term crushing has not quite the same significance as in our subject, for as soon as the material is broken as the result of a certain application of force it is considered to be crushed, while we in ore-treatment may have to apply force many times to reduce the size of the resulting particles to the required extent. "Strength of materials" is concerned with resistance and work done, etc., mainly before fracture occurs and chiefly within the elastic limit, since the material under test is required to resist fracture in use, while in connection with ore crushing problems it is only the work required to rupture which is of moment—any work which merely deforms without fracturing being so much waste. Nevertheless, the first principles underlying the consideration in each case are the same, and as much experimental work has been done towards the elucidation of the laws it will be well to know some of the conclusions arrived at.

Martens* states that resistance to crushing per unit area is dependent upon shape of cross section and length of prism and for similar section prisms

$-s = \alpha + \beta \sqrt{\frac{a}{l}}$ in which $-s$ is the crushing resistance, α and β are constants, area is a, and l is the length.

Experimental determinations gave the value of α as several times β. In the case of a cube $\sqrt{\frac{a}{l}}$ of course becomes 1, and for

* Handbook of Testing Materials (Translated by Henning) Wiley & Sons, New York, 1899.

cubes resistance to crushing, per unit area, is constant, for other shapes it increases to some extent as the s area and inversely as the length (*i.e.*, height). This, then, introduces a slight divergence from the assumption made, but since s is relatively unimportant, the effect will be small. The effect of cleavage planes, which of course are planes of weakness, has also been mentioned as introducing a great factor of uncertainty in theoretical deductions. Martens has a statement which is of importance in this connection also. "If in crushing tests a monolithic cube has strength 1, the strength falls in a composite block of eight cubes to '88." Of course, between these eight there are no bonds at all, but this absence of bonds will be similar in effect to the presence of cleavage planes in a block of mineral, though presumably the strength will not be reduced to '88." It tends to show, though, that smaller particles ought to sustain less crushing force—per unit area—than large ones, as before, though the difference is not very large. Morley* shows that the shape of pieces has a great influence on the work required to break them, for instance, consider two pieces of of the shapes shown, of the same area at the top, Fig. II. Both will break under the same application of vertical force (2, obviously, in the upper part), but even if the distortion in 2 is as great as in 1, which it would not be, obviously the smaller body

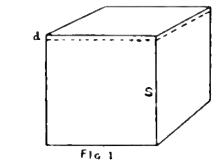


FIG I

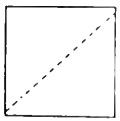


FIG III

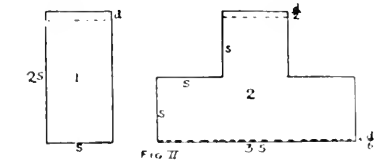


FIG II

has not required the less energy. Theoretically, it can be shown that in the case of 2 the work required is only $\frac{1}{2}$ that for 1. At first sight this is somewhat disconcerting, but from the metallurgical view point two more fractures of the lower part of 2 are still required to break the resulting pieces of 2 to the same size as the two resulting pieces of 1, this using (by calculation) a total of rather more than three times the power in the case of 2 than in 1, consequently the deduction still holds approximately.

Morley also says that in brittle materials the strains up to the point of rupture are small, and if the material is homogenous and isotropic the distribution of stress is probably nearly the same as within the limits of elasticity. And, further, that the fracture of brittle metals and other brittle materials under compression generally takes place *partially or wholly by shearing* or sliding in a direction oblique (*ca* 45°) to that of the direct stress, Fig. III. This seems to be most important, for the total force required is still proportional to the linear dimensions squared, and although the body fails by shearing, as it is alleged that ores do, the work done is still proportional to the volume, *vide* Kick, and the possibility of shearing as a valid objection seems to disappear.

Morley shows how the application of a compressive stress to a body results in the production of shear stress, and says, further, that a shear stress in a given direction cannot exist without a balancing shear stress of equal intensity at right angles to it, and, again, that shear stress in any plane is always accompanied by shear stress at right angles to it.

Martens says that shear stress is somewhat affected by the dimensions of the piece, but that the forces required for shearing bodies of identical material and geometrically similar will be proportional to the surfaces to be sheared. This recalls Rittinger, but Martens gives *force* and not *work*, and Rittinger's law is not true for pure shearing, unless it were possible to produce shearing without in any way deforming the remainder of the body, which is impossible, inasmuch as it must be supported in some way to resist the force at all. Consider two cubes of edges s and s_1. The force required to shear is proportional to s^2 and s_1^2, therefore the compressive force per unit area is constant, and its distance of action will be proportional to s and s_1. Therefore, work is again proportional to s^3 and s_1^3. Applied to a piece of rock in a crushing machine (Fig. IV.), the application of force in the direction shown will obviously produce shearing force, but ac-

* "Strength of Materials." Longmans, London, 1911.

companied by balancing shearing forces at right angles, the resultants placing different parts in either compression or tension, and the piece will fail along some surface or surfaces of weakness.

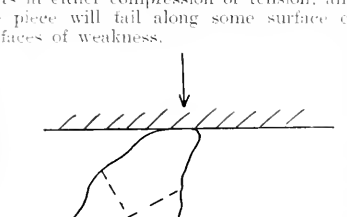


FIG IV.

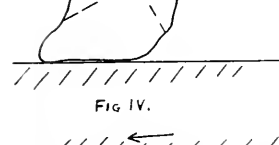


FIG V.

With regard to impact, Martens says that under identical total impact heavy blows produce greater upsetting (deformation) than light blows. Morley says stress at the moment of fracture is higher under impact or very rapid loading, and elongation (deformation) also much higher. Consequently the work done in fracturing by impact would be more than by slow crushing. Both refer, apparently, to metals and not to brittle materials, but there seems to have been very little work done on this point, and the effect of the manner in which the breaking is done must therefore be left open for the present. If brittle materials do behave similarly, though, crushing by impact would not seem to be advisable from the point of view of power consumption. Lastly, if the pieces are not broken by halving, but by dividing into 3, 4, or x pieces each time, the number of pieces formed by repetition of the divisions will still be the second, third, fourth, and so on powers of 3, 4, or x, as the case may be, and the total power would still be proportional to these powers or logarithms, and still increase in arithmetical progression.

A distinction has sometimes been drawn between the action of grinding machines by stating that some of them work by abrasion, and this may be held to involve a different amount of energy. But if the action be examined it will be seen that the amount of real abrasion taking place—except on the wearing surfaces of the machine—is small, and that the action is really crushing or shearing. A particle between two grinding surfaces, which are *never* really smooth, is not merely held by the one and worn away by the other, but owing to friction between it and the surfaces is immediately subjected to compressing or shearing forces, and yields accordingly. I have shown flat surfaces, as in a pan, but the action between rolling and sliding tube-mill pebbles would be similar. It will be admitted then, I think, that none of the objections mentioned are of sufficient importance to adversely affect the general accuracy of the rule.

I do not consider that the data available are sufficient for the close comparison of our crushing practice with theory, and, hence, for an exact calculation of efficiency, since we have not available, for instance, the real average size of the final product, or indeed any of them, but by making some assumptions some interesting results are obtained.

Suppose, for example, in local practice that the average size of the breaker feed is 6 in. diameter, and that of its product and of the battery feed is 1½ in., while that of the battery product is ·1 in. and tube-mill feed slightly more, say, ·15 in., and of its product ·001 in. Taking the same weight in each case

the breaker will reduce from ...	6000 to 1506	ratio of $\frac{\text{original}}{\text{final}}$ dia. is 4·0	and log ·602
the stamps reduce from ...	1500 to 100	„ „ „ 15·0	„ „ 1·176
the tube-mills reduce from ...	150 to 4	„ „ „ 37·5	„ „ 1·574

The efficiency of the three types is expressed as $\frac{\text{actual}}{\text{theoretical}}$ tons per h.p. The theoretical can only be determined from the result of measurements of the work required to break known cubes, and this is

not available; it may, however, be taken as a function of a constant K. In the three cases cited the theoretical tons per h.p. will obviously be inversely proportional to the theoretical work required, *i.e.*, as $\frac{1}{0{\cdot}602}$: $\frac{1}{1{\cdot}176}$: $\frac{1}{1{\cdot}574}$ and is then a product of these figures and K, *i.e.*, $\frac{K}{0{\cdot}602}$ $\frac{K}{1{\cdot}176}$ $\frac{K}{1{\cdot}574}$, in other words, the breaker should crush more tons per h.p. than the stamps, and the stamps more than tube-mills; but unless the practical figures obtained are in the same ratio the efficiencies will not be equal. Assuming that the tons per h.p. per day in the three cases are 12, 1·5, and 2, the efficiency will be proportional to $12 \times 0{\cdot}602$, $1{\cdot}5 \times 1{\cdot}176$, and $2 \times 1{\cdot}574$ or $7{\cdot}22 : 5{\cdot}29 : 3{\cdot}15$, *i.e.*, efficiency is proportional to output per h.p. $\times \log \frac{\text{initial}}{\text{final}}$ diameter. Admittedly these figures can only be rough approximations for want of accurate data, and the subject is important enough to justify an extended investigation with the object of obtaining reliable figures. I think the average diameter is probably less than I have assumed, the error being probably greater in the finest size, which would tend to show the tube-mill, in the calculation, in a less favourable light than it deserves. Nevertheless, assuming they are not very far from the truth, it indicates that the tube mill is considerably more efficient than it is considered to be in some quarters. There is another factor which adversely affects the tube-mill efficiency figure and renders comparisons on this basis unfair, namely, that the material it deals with is more siliceous than the ore as whole, it consists, in fact, of the more difficultly reducible portion of the ore, which theoretically requires more work per unit volume (or weight) than the original ore. Taking this into consideration still further improves the tube mill efficiency. In any case, a stamp mill for the same work would be so inefficient as to be impossible, just as it would be impossible to use a breaker to crush to ¼ in. Each machine, as is well known, has a limited range through which it is efficient, and a tube-mill should only be compared when doing certain work with other machines which are suitable for the same class of work.

RESEARCH IN THE MINERAL INDUSTRY.

The absence of certain data to which I have drawn attention indicates one of the many directions in which further research is necessary. I am not, of course, the first to call attention to the need which exists for such work to be done. Dr. Caldecott and Mr. Stadler, notably, have urged its importance, and it is understood that a large amount was done by the Mines Trials Committee, now disbanded, but not for publication.

With regard to other necessary work, it is only needful to glance at the position with regard to certain "refractory" gold occurrences to realise how necessary such work is and how difficult, if not impossible, it is for it to be undertaken by individuals. Rather should the Government, a potential partner in profits, come to the assistance of the Industry and undertake the duty. Our enormous deposits of easily treated banket so overshadow in importance any other known gold occurrence that we are apt to think lightly of other occurrences of different nature in "outside" districts. Some of them, of course, are being successfully worked, but there are others of sufficient magnitude to have received, in other countries, more attention than they have done in this. I do not, of course, wish it to be inferred that they have not in many cases had a lot of money and trouble expended upon them, but it has often been a very futile expenditure from causes which need not be unduly elaborated, but which include the erection of plants for stamp milling and cyaniding as on the Rand, for ore which is totally unsuited to such treatment. After such an experience, shareholders and all concerned are apt to become discouraged, the property acquires a bad reputation, and it is difficult or impossible to attract fresh capital and make a new start, not necessarily to work the mine, but to first find a suitable process. There *are* some ores which seem to resist all known methods, and others for which known methods, though they might extract the gold, would not be possible under the existing circumstances. Nevertheless, I believe there are properties for which a profitable treatment could be devised, and the fact that they are lying idle obviously means also that labour they might employ is not being employed, there is no distribution of wages, circulation of money and trade; no profits tax, and often no claim licences being paid to the national revenue, and no dividends either.

When a new discovery is made and certain exploratory and development work carried out, any capable metallurgist would be able to suggest and test methods of treatment and recommend accordingly. But it might be found that the ore was not amenable to any of the ordinary treatment methods, and a long and expensive investigation with respect to the causes of difficulty and solution of the treatment problem might be indicated. Many, I might say most, owners at this stage, particularly the smaller ones, would decide to drop the venture, and consider themselves fortunate that they had not discovered the true character of the ore through the failure of an expensive plant. At this point, then, I am of opinion that the Government should step in, as an interested party, and cause the required investigation to be made. It would perhaps be possible to commission a practising consultant to carry out the work, but two difficulties at once become apparent which would prevent this being always feasible. First, the work to be done is so indefinite as to nature and probable time occupied that it would be almost impossible to quote a fair fee, and, secondly, certain appliances and apparatus which might conceivably be found necessary are only to be found in a more or less complete testing plant. In other cases the real difficulty might be due to the limited financial resources of the owners of the property, and then the Government might assist financially in finding a suitable process.

Similar cases arise in connection with other metals than gold, and also with non-metallic minerals, of which many occurrences are known. Surely in connection with the exploitation of oilshale, asbestos, graphite there will be work to be done, not to mention the better utilisation of coal and its by-products.

The Government has not been entirely unmindful of its duty in this respect in the past, as instanced by the grant from the "Prospecting Vote" in aid of the equipment of the ore-treatment laboratories of the South African School of Mines, which are available under certain conditions for testing parcels of ore. These, in conjunction with the remainder of the metallurgical and assaying department and the other laboratories of the Institution, provide most of the essential apparatus and appliances required for the conduct of such investigations, though admittedly certain extensions and additions would be desirable. But with this contribution the direct Government provision ceased: any investigation undertaken is by private individuals or by the instructional staff, whose time for such work is all too limited. It would be too much to expect that the Government would establish an entirely new set of laboratories for the purpose I have outlined, though in Canada this has recently been done, and indeed the present laboratories are suitable: but obviously a suitably trained staff would be necessary, and, as a commencement at any rate, the provision of this staff and necessary running expenses would be all that would be required. The financial outlay would be therefore very small in comparison with possible or even probable increase of revenue to the Government in the directions I have indicated, without considering the concurrent advantages of increased employment.

At the same time, there is work of somewhat different character to be done, not directed entirely to financial gain, but primarily of value for its scientific interest, though, as in many cases heretofore, such work often has unexpected results in the former direction: for example, the elucidation of the laws of crushing and the efficiency of crushing machines, and the causes of failure of stems and cam shafts, which might very conceivably result in considerable practical advantage.

The work undertaken might therefore be considered as falling under either of two heads (a) for the elucidation of ore treatment problems, and (b) for affording information of value to the industry in connection with existing methods of treatment. In either case the work undertaken would be selected so as to be, as far as possible, of value to the industry as a whole, though at the same time, the position of the Government as a "partner" in profits of successful mines would have to be kept in view. Details, of course, would require to be settled but I trust that something will eventuate as a result of this suggestion, with advantage to the mineral industry and the nation.

Mr. J. E. Thomas (*Vice-President*): I have much pleasure in proposing a hearty vote of thanks to our President for his Address. It must have entailed an enormous amount of work on his part, and it is very evident that it will entail a considerable amount of study on the part of members who read it as it should be read. It

is [illegible] difficult to follow a paper of this sort as it is read, but we shall all have the opportunity of thoroughly studying it in the *Journal*. I believe it is not customary to discuss the Presidential Address, but I expect our President would be pleased if anyone who likes to say a few words on it now will say something about it. Gentlemen, I have much pleasure in moving a vote of thanks to our President for his Address.

Mr. H. A. White *Member of Council*: I have very much pleasure in seconding the vote of thanks to our President for his very able Address, and I think, as a Society, we certainly can support the second portion of the Address, which draws the attention of the Government to making investigations into something which will be of benefit to the Country. With regard to the first portion of the Address, I think the President is to be congratulated on the fact that discussion has arisen. I do not think I would like to say very much about it, except to just point out the fact that the impression referred to in that diagram before you is the real reason of contention between the two laws. That is impression before fracture, we agree, in practice is absolutely necessary; in theory it is not. Therefore, when we are dealing with the referred results or questions of this sort we can assume that the power is applied at the actual fracture plane, and it is curious that the experiments which were made on behalf of the Mines Trials Committee by dropping a weight thirteen times, by the mere test which the President has referred to, showed that Kick's law was nearer the truth, and that Rittinger's law was further away from it. One gets nearer the effect by utilising the power at the plane of fracture. This implies that the "distance" factor is always the same in the same material depending upon the range of molecular forces, and not upon dimensions of the body fractured.

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

Read at May Meeting, 1914.

By G. H. Stanley, A.R.S.M., M.Inst.M.M. *President.*

DISCUSSION.

Mr. J. M. Neill *Member*: The author has very ably demonstrated the difficulties of classification, deserves the thanks of operators, who, without quite knowing where trouble originated, have had to contend with just such difficulties. Much will now be made clear to them. He is also to be congratulated upon having devised a practical method of achieving the results shown, because of the influence it may have on the treatment of many base metal ores.

The following sentence, however, is rather contradictory: "This imperfect classification has obvious disadvantages, more particularly when it is desired to separate sand particles of the same kind, *i.e.*, specific gravity, into groups of different sizes."

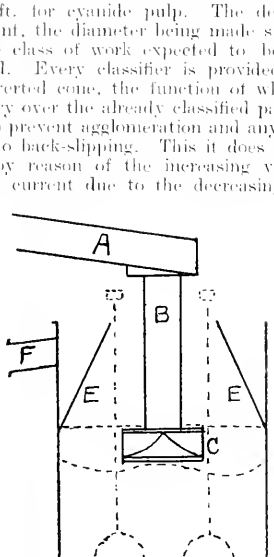
According to the notes, imperfect classification results in a mixture of coarse and fine particles; separation "into groups of different sizes" would give a similar result. May I suggest that separation into different groups of equal sizes would probably convey the meaning intended and be more in accordance with the author's notes of results achieved, *i.e.*, all + 60 underflowing, all − 60 overflowing. So far as the metallurgy of a gold ore, capable of ordinary mill, tube, and cyanide treatment is concerned, and upon which trials and results presented are based, classification into groups of equal sizes does not seem to offer any advantage. On the contrary, classification into groups of equal hydraulic value would seem to be more reasonable. In this way much of the fine heavy mineral which requires extra comminution or longer and stronger cyanide treatment, will appear in the classifier underflow instead of in the overflow. For example, take the following grading analysis of pulp to cyanide treatment, assaying 2·53 dwts. per ton.

Percentage		Value per Ton dwt.	Gold Contents dwt.	Percentage of Total Gold in Sample
+60	3·3	1·2	3·96	1·53
+90	18·6	1·92	35·712	13·81
+200	20·5	2·8	57·4	22·20
−200	57·6	2·8	161·28	62·43
	21·9			
	78·1			

Examination of the above would lead one to expect, classification notwithstanding, that any percentage of −90 underflowing with the sand would likewise have a value of 2·8 dwt. per ton, and so, on a classification basis of 50 sand and 50 slime, the values would be respectively 2·367 dwt. and 2·8 dwt. This proved not to be the case, however, as the actual charge values on this particular pulp were, sand 3·41 dwt.

and slime 1·74 dwt., consequently the 28·1 ot +90 taken down with the sand had a value of 4·71 dwt. per ton.

Consideration of the foregoing, and a desire to participate in the discussion which the author so kindly invites, encourages me to present a few hitherto unpublished facts relative to the classification at the Benoni Reduction Works. Stadler classifiers were installed throughout the plant, and generally from 18 to 21 were in daily use. The body of the classifier is circular in shape, and can be provided with from 1 to 5 discharge openings. The depth is 4 ft., the diameter varies from 3 ft. for tube mill pulp to 5 ft. for cyanide pulp. The depth is constant, the diameter being made suitable to the class of work expected to be performed. Every classifier is provided with an inverted cone, the function of which is to carry over the already classified particles and to prevent agglomeration and any tendency to back-slipping. This it does admirably by reason of the increasing velocity of the current due to the decreasing area



A. Launder; B.—Down Pipe; C.—Shoe; DD.—Discharge Openings (Underflow); EE.—Inverted Cone; F.—Outlet (Overflow).

through which it has to pass. It also forms a launder through which the overflow reaches the outlet. A white-iron shoe is used to give the pulp a rotary motion from centre of classifier outwards. Change of direction, so inimical to good classification, is by this means overcome, and hydraulic grouping allowed freely to take place. The shoe is somewhat similar to the impeller of a centrifugal pump, but, being firmly attached to the down-pipe conveying the pulp to the classifier, it does not move. The outlets for underflow are bolted to the bottom of the classifier.

The classifying area is the space from the top of the thick pulp (composed of particles too heavy to remain in the rapidly revolving stream issuing from the shoe) to the bottom of the inverted cone. The shoe lies within this area, which is roughly indicated by the dotted lines. Herein lay the first difficulty, the classifying area could not be maintained. It either choked, or channels formed right through to the outlet, and many of the difficulties set forth in the author's notes were encountered. To overcome these, the scheme I devised was to make use of the channelling by suspending therein, in such manner as to permit of free movement over the outlet or outlets, a hollow cast-iron ball or balls, of suitable diameter, shown dotted, filled with the material being classified, and securely plugged. The movement of the ball was demonstrated by self-recording chart to be as much as 1¼ in. Its specific gravity was somewhat greater than that of the pulp being classified, and its use made possible a classifying area of any desired depth, so long as the ball was not actually uncovered. The outflowing material being so largely governed by this area, little more trouble was now experienced. With a fairly continuous feed the height of classified pulp within could be maintained at any desired point. The underflow carried a constant moisture, tube mills 19 , final pulp 27½ ; breakaways were quite unknown. By further experiment I am satisfied that the moisture could be very much reduced if necessary. The overflow still required a little attention, as, owing to the vortex formed by pulp rushing down the pipe from launder to shoe, air was carried down, and, in escaping, carried fine sand over with it. The down-pipe was therefore cut off below the launder and the shoe attached to a larger pipe in which the pulp discharged. The air freely escaped from within this pipe. Results were then as tabulated below, and as money was a consideration further experiments were not tried. The screen tons dealt with per tube classifier per hour were 8·66 minimum solids, not including returns.

Gradings (average of one month's run 18,000 tons)

Screens (61 mesh)			
+60	+90	+200	−200
16·7	12·2	9·4	31·7

Tube Classifiers — Underflow				Screens + Returns — Overflow			
+60	+90	+200	−200	+60	+90	+200	−200
76·9	14·2	4·7	4·2	8·2	18·9	18·3	54·6

Cyanide Pulp			
+60	+90	+200	−200
4·5	16·4	21·9	57·2

Value 2·41 dwt.

Underflow to Caldecott table				Overflow to slime			
+60	+90	+200	−200	+60	+90	+200	−200
15·33	32·35	46·22	77	0·14	0·65	3·56	95·65

3·13 dwt. 1·72 dwt.

A series of check gradings for another period of eight days gave the following results:

	Sand to Caldecott Table			
	+60	+90	+200	−200
Averages	6·975	37·6	37·825	17·6

	Slime to Settlement Cones			
	+60	+90	+200	−200
Averages	0	0·375	2·6	97·025

There was no added hydraulic water used and no secondary classification of the slime. The sand fell direct on to a Caldecott table, which ran from March 10th to May 1th without changing cloths. The vacuum remained about 5" m. Hg. In this case the vacuum and water pumps were approximately 27 ft. lower than the filter bed of the table, which may have had a beneficial influence. In any event, the underflowing sand was a very leachable product, notwithstanding the quantity of −200 it contained.

The grading averages were influenced by certain experiments in crushing involving the whole plant, and by frequent stoppages also, which are detrimental to good classification. Nevertheless, 92 to 95 +90 were common returns in tube classifier underflow, and 97 to 98 −200 in slime overflow.

I cannot conclude without paying a tribute to the assay department of the Klein-fontein Mine, where all the grading and assays were carried out. The relationships, under and overflow, both by grading and value, are remarkably close to the grading and value of the total pulp. For instance: In total pulp 20·9 −90 becomes 41·77 in sands underflowing classifier. Sands 5 practically 50.

100 tons total pulp 50 @ 3·13 212·5 dwt.
241 dwt. 50 @ 1·72

Considering that those are the results of many hundreds of samplings, gradings, assayings, etc., they seem to me to be excellent.

The President: I think we can congratulate Mr. Neill, and I congratulate myself, on his contribution to the discussion. When one writes a paper the object is not only to give information, but to elicit information, and Mr. Neill has come forward with quite a lot of useful information in this connection.

RELATION OF FALLS OF ROOF IN COLLIERIES ON THE MIDDELBURG COAL FIELD TO WEATHER CHANGES

(Read at March Meeting, 1914.)

By Chas. J. Gray, A.R.S.M., F.G.S., M.I.M.E. (Member).

REPLY TO DISCUSSION.

Mr. Chas. J. Gray *(Member):* While I regret that my paper has called forth so little discussion, I should perhaps congratulate myself that it has escaped serious adverse criticism.

The two points which Mr. Dewar brought up are all that require attention. His first point was that the location of accidents should have been given, showing whether they occurred at the face or back from the face. I would reply that if I had separated accidents from falls which occurred at a distance from the working face, those due to falls of sandstone, those due to recent blasting, and those due to other causes obviously having no connection with the drying action of the air on shale or coal roofs—I could, as I satisfied myself by rough trial, have shown still more striking correspondence between the remaining accidents and dry weather, than was given by the statistics, which included all accidents. I did not do so because full information was not available in all cases, because such manipulation, which could not have been checked by members, would have opened the door for suspicion, and because it was unnecessary. The case was strong enough without such additional elaboration. I may, however, now say that out of the 25 deaths in the years 1910 to 1912 only four were caused more than twenty or thirty feet back from the face, and that all those accidents occurred far from the mine entrances.

Mr. Dewar's second point was that on winter mornings the white men may not go

into the mine till after the natives, and that their absence may increase the number of accidents from falls in the winter. It is quite true that a certain number of accidents occurred in collieries through the natives reaching working places before they had been examined, but even if they all occurred in winter, their number would have been quite inadequate to account for the preponderance of accidents actually observed. In the year 1912 two fatal accidents occurred before the white miner had visited the places, but as one occurred in the winter and the other in the summer, they did not cause the marked excess of winter accidents. That the remissness of white miners in that respect is not the explanation, can also be seen from the correspondence of changes from year to year in accident rates, with changes in the reverse direction in rainfall. There is no reason to believe that a man will be more prompt in examining his working place in a wet year than in a dry year.

I consider that I am justified in adhering to my view that the increased drying action of the air in winter on the coal and shale roofs in the Middelburg coalfield is the cause of the greater number of accidents from falls of roof.

ROCK TEMPERATURES.

(Read at May Meeting, 1914.)

By E. J. Moynihan *(Associate).*

DISCUSSION.

Dr. Jas. Moir *(Past-President):* I think it is regrettable that a paper of such economic importance to the Rand as this should go undiscussed, and this must be my excuse for intervening in what is mainly an engineer's affair. It is particularly unfortunate that Messrs. Whitehouse and Wotherspoon have not defended their work from the author's attack. My own view of their work is that it was ridiculous of them to state their average gradient as 1° in 253·9 ft., when this value is in doubt by at least 50 ft. on their own evidence, as Mr. Moynihan shows. Their work has an accuracy of about 1 in 5 instead of 1 in 2,500. I do not think, however, that Mr. Moynihan has succeeded in upsetting the common opinion that the Rand temperature gradient is 1° F. in 200 ft., more or less. I think myself that the evidence shows it to lie between 1 in 175 and 1 in 225, and the Government Mining Engineer, in his 1911 Report, held the same view by giving the formula T 60 + h 200. It is true that very many observed temperatures are higher than this formula would predict, but I think it could be shown in every case that the abnormal temperatures were obtained in close places where overheating by breathing takes place. Mr. Moynihan's method of taking the highest reported air temperature and comparing it with its depth, leads to an average gradient of about 1° F. in 150 ft. from the 26 deepest mines* (over 2,000 ft.) of the Rand, which supports his argument well, but is not evidence that the true rock temperature gradient is the same. Nevertheless, most of us will accept the author's view that temperatures taken in shallow holes will not give the true rock temperature, and that in most cases the true temperature is higher than the borehole reading. This argument, however, applies all over the world, and Mr. Moynihan is not, therefore, entitled to state that the Rand gradient is "not abnormally low" as compared with European gradients. It *is* abnormally low, in spite of the observational errors to which he calls attention. Again, I am by no means convinced that a freshly made borehole is necessarily cooler than the rock. It takes a lot of compressed air to counteract the effect of breathing in a close space; and as for the drill water, it is used only in small quantity, and is practically preheated in the pipes up to the prevailing temperature of the end and further slightly heated by the drill. I agree, of course, that the thermometer should be left sealed in for a long time (weeks) to correct the initially abnormal temperature of the hole and overcome the poor conductivity of the rock.

It has to be admitted with regret that most of the temperatures in the Mines Department Report quoted by the author are unrepresentative. They are mostly supplied by mine officials who have no particular scientific interest in getting them right, and none whatever in getting them consistent. Thus at about 4,000 ft. depth we get 65° reported by the Turf Mines and 86 by the Cinderella, both probably quite true and both easily explainable by ventilation, but quite inconsistent unless explained. The New Modderfontein figure of 90,

* Omitting the Village Deep, the figures of which are obviously unrepresentative in the opposite direction.

quoted by the author, is, I find, a clerical error for 70° in the original mine return, but it is a great pity that it found its way into print without being detected by some one of non-clerical training. Other years' returns are 75°, 65° to 75°, and 60° to 70°.

In conclusion, I hope that the author's stimulating criticism will lead to proper borehole observations being made at an early date, since so much depends on knowing when artificial refrigeration will become necessary. At present it seems to be somewhere between 5 and 15 years hence for the ten deepest mines, though Mr. Moynihan evidently thinks it will be sooner. In any case, close places in deep mines will always show temperatures higher than the rock temperature, and this will always have to be allowed for.

Mr. H. Meyer (*Member of Council*): It is not proposed to discuss Mr. Moynihan's criticism of Messrs. Whitehouse and Wotherspoon's paper read in May, 1911; no doubt Mr. Whitehouse will do this himself when he returns in a few months' time. A good deal of practical work on this subject has been done recently by Mr. Townsend, on the Village Deep Mine, and he has allowed me to make use of his observations to discuss some of the points of contention raised by Mr. Moynihan.

The first point made by Mr. Moynihan is that the method of taking rock temperatures by means of shallow drill holes underground is untrustworthy. He, however, assumes that rock is so uniform that depth is the only factor governing temperature. The presence of streams of underground water might have an important bearing on the subject, and there is no certainty that uniformity would be obtained by a system of very deep holes. In order to predict the greatest depth to which mining can take place without recourse to refrigeration, is it not more important to ascertain the conditions of increase in temperature of the rock at open-end mine workings? The temperature of undisturbed rock is scarcely of the same importance. Our practice is therefore to obtain the temperature of rock in shallow holes in dead ends and to compare these results with those obtained after connection has been made and when good ventilating currents are passing through the same place. As long as the temperature of rock is not inconveniently high in the neighbourhood of the work, it is not particularly important to know what it is in the undisturbed ground.

These remarks somewhat overlap Mr. Moynihan's second main point, viz., that reduction in temperature at the Village Deep is not due to the installation of a fan or better ventilation, but to a seasonal fall in the surface temperature. This is not supported by the following evidence. In May, 1913, temperatures were taken of the rock in the 18th level, 4,015 ft. below the surface. The temperature in this dead end was 81·6° F. Subsequently, the level was connected with those above and below, with the result that a strong ventilating current passed through it. In May, 1914, the rock temperature was again observed, and found to be 77·1° F., or 4½° lower. The mine authorities consider that improved ventilation has produced this result, and not a seasonal variation of surface temperature between May, 1913 and 1914.

In the deeper mines of the Rand it will be found that seasonal changes in the temperature of the air at the surface are reflected very slightly underground. The temperature of the air at the surface and at the bottom of the Turf Mines vertical shaft was read at noon and at midnight. This shaft is a strong downcast, the air passing direct from the surface to the bottom. While there was a difference of 7° in the temperature of the air at the surface, there was less than ½° difference at the bottom of the shaft. This being the case, the temperature of air and rock in the dead ends is not likely to be affected by seasonal variations on the surface, as has been proved by actual observations.

I consider that the cooling effect of a good current of air passing through the mine workings is much underestimated by Mr. Moynihan. As an instance, I may state that the temperature of the air was taken on the 19th level, 4,180 ft. below the surface. One observation was made in the ventilating current and the other in a dead end 120 ft. away. The first reading was 76·9° F., and the second 82° F. This instance is cited because of the proximity of the two points of observation to each other; in every case where the temperatures were compared it was found that stagnant air was much hotter than the ventilating current. It is therefore difficult to forecast the temperatures that will be encountered at a depth of 5,000 ft. At the lowest point in the Village Deep within reach of the ventilating current where any reading was taken, viz., 4,575 ft. below the surface, the air temperature was found to be 79·6° F.

The large amount of artificial watering and spraying that is now done underground has the effect of increasing the wet bulb temperature, and calls for greater quantities of cool air. Whether it will indicate 85° F. at a depth of 5,000 ft., regardless of the efforts in ventilating, is, in my opinion, doubtful. Mining men hope to disprove this figure by drawing into the mine and circulating as large volumes of cool air as possible. Should Mr. Moynihan be correct, we hope to artificially cool the air and make working conditions possible at a considerably greater depth. Before this last resource is required an improvement can be looked for in other directions with the object of the better utilisation of the volume of air drawn into a mine. With clean, dry intake roads, carefully laid out development, and every attention given to leading the air through channels where it does useful work, I think it will be found that reasonably comfortable and healthy working conditions will be possible for a considerably greater depth than has already been attained.

In conclusion, it will be of interest to know that Mr. Townsend's latest observations of temperatures taken in 6 ft. drill-holes in dead ends at various depths indicate an increase in temperature of 1° F. for 258 ft. of vertical depth down to 1,575 ft.

The President: This is certainly an effective contribution by Mr. Meyer, and I hope that other people who are in a position to know the figures will come forward with details, either in support of, or controverting, Mr. Moynihan's remarks.

THE UNION PATENTS BILL.

(Read at April Meeting.)

By A. L. Spoor and W. E. John (Members).

REPLY TO DISCUSSION.

There has been rather more discussion on this paper than we expected, and we wish to thank all those who have taken part in it. Two contributors have tolled the passing bell over the subject of our paper. The obsequies are, however, premature, as the new Draft Bill to which they refer is in substance the old one with copyright provisions added.

Mr. Donaldson's suggestion that the proposed sub-offices might be used for keeping copies of specifications for public inspection is sound in principle, and would certainly invest such offices with at least one useful function. The ultimate object of all patent legislation is to increase the knowledge of useful arts among the people of the country, and unless the records of inventions are made completely and readily available that object is not fully attained. We believe, however, that the spread of knowledge by this means could be better accomplished, as Prof. Wilkinson suggests, viz., through the medium of the existing public libraries.

Mr. Donaldson and Mr. White are in favour of compulsory working, while Mr. Cullen and Prof. Wilkinson agree that it would be a mistake; the voting on this point being therefore equal. The contributors who support it have mentioned about all that can be said in its favour. They will see that their objections apply to relatively few cases, and are fully met by compulsory license without jeopardising the validity of practically every patent, which would be the result of compulsory working.

Mr. Cullen is in error when he says that "even to-day there is no Transvaal Register of Patents." The Pretoria office at the present time keeps the following records:—

(1) A general register in which applications are entered in numerical order. This gives the name of the applicant, the names of assignees and licencees if any, the title of the invention, date of application, etc.

(2) A separate name index of applicants, assignees, and registered licencees. This is in book form, and requires to be converted to the card system.

(3) A classified title index, which contains the title of each published application grouped (by its title) into one or more of the 146 subject matter classes.

(4) A resumé of the subject matter of each complete specification, following the English system of classification. This useful material is at present in manuscript. It embraces applications from 1906 to date. In conjunction with index (3) it materially facilitates a subject matter search over the period which it covers; and ultimately if it is extended to embrace the published specifications of all the Union Provinces it will provide the material for what is undoubtedly the most urgently required record, *i.e.*, a detailed index of subject matter. The preparation of the latter will, of course, be a big task, but fortunately the framework of it is already available in the English classification system, which could be adapted with some modifications.

Mr. H. G. Wells has written a pleasing tale of a future period when the utilisation of intra-atomic energy has so multiplied human wealth that there is enough and to spare for all. When this time comes Mr. Moynihan's ideas will no doubt be realised; but in the present state of society the producer of a new industrial idea is reasonably demands some security for it as he does when building a house. True, none can positively correlate cause and effect, but if patent protection does not foster inventions and industries, the case of Switzerland presents an extraordinary coincidence. The Swiss Legislature, considering that Switzerland was in communication on all sides with industrial countries, decided that everyone should be free to adopt and exploit without restriction any invention which they wished to. Switzerland remained what she had been for centuries, a pastoral country. Then a patent law was adopted, and immediately there commenced that industrial awakening which has made Switzerland, with practically no natural asset but its waterfalls, a potent factor in modern industrial life. We agree that there are those who will continue to invent under any circumstances; but without a Patent Law their best ideas would, in nine cases out of ten, be still-born. To reduce an invention to practical concrete form and adjust it to its proper place in the complex fabric of modern industrialism is certainly no less a matter than its conception; and few will be found to expend the necessary time and money, if anyone is to be free to step in and appropriate the fruits of their labour.

Mr. Moynihan quotes advertising schemes which, without protection, bring profit to their authors. It is clear, however, that an advertisement is essentially ephemeral, and its value lies in its individuality, so that the mere copyist is likely to benefit the originator as much as himself. In the case of a technical invention, the more nearly an infringer can copy the inventor's perfected work the better for the infringer. However, Mr. Moynihan having shown us the promised land, leaves us yet many years wandering in the wilderness. In palliation, he suggests an unassailable patent undoubtedly an ideal to be striven for—and if he would show how to achieve it he would earn world-wide thanks; but the only means to that end which he puts forward has been in daily operation in this country and others for very many years. As to his second item, let it be supposed that someone invents, let us say, an effective and simple system of wireless telephony which brings the telephone within the reach of every farmer in the country. According to Mr. Moynihan, he should not profit by it: but must allow all rivals to utilise it without fear of an action for infringement provided only they take the trouble to invent and patent an improved form of, say, receiver or any other detail. His third point is somewhat in accordance with the general tendency of to-day. But whilst the more technical knowledge a tribunal can have the better it will be, is not the judicial mind always necessary, and would not a pure technician surely make a sad failure as a judge? His charity in the matter of costs is charming. No matter what unwarrantable expense a rival might put him to in defending his rights, he would be delighted to bear it all himself.

Mr. White has put forward such an exhaustive and interesting criticism that one only regrets that he has seen fit to colour it so highly with what he refers to as the other side of the bargain. The bargain between the patentee and the public does exist, but as it is fundamentally one in favour of the public, we doubt if the maxim *caveat emptor* needs to be applied so stringently as our critic suggests. The inventor on his side discloses his idea, and the public makes the purely negative contribution of refraining from appropriating it for a limited period. If Mr. White will permit us to interpret him, we think his position would be clearer if it is understood that he is championing the public as against an applicant who is to a greater or less extent not a true and first inventor. The discrepancy between Government fees and inventors' profits, which Mr. White refers to, is doubtless due to the excessively high fees imposed by some unenlightened foreign Governments, and emphasizes the moderation of the present Transvaal fees, to which of course we referred. They are, as a matter of fact, far below those of any European country. Mr. White's bold exposition of the parentage of an invention ("if necessity is the mother of invention, adaptation is the father") nearly takes us out of our depth. The well-known proverb suggests a parthenogenetic origin; and if one forsakes this theory, the difficulty of identifying the other progenitor is very considerable. Even Mr. Cullen in his discussion claims the distinction in certain cases. Our own feeling favours the claim of the inventor's ingenuity.

We fully agree with Mr. White's claim to have all patent records properly arranged and accessible. This is one of the most important functions of a patent office. We have already mentioned what is at present done in the Transvaal office; Prof. Wilkinson has indicated the general lines on which further steps should be taken. To carry these out would require the appointment of such a number of examiners that each could confine himself to matters in which he was expert, together with clerical assistants.

The views expressed as to the necessary qualifications of the Commissioner of Patents leads us to mention that the principal change in the new draft of the Bill is one referred to by Prof. Wilkinson: that all the judicial functions which were formerly possessed by the Transvaal Commissioner are to be vested in the Supreme Court.

Mr. White girds at the futile amusement of definition-making, but he must surely see that the present is not a case in point. The definition of invention is the very foundation of any Patent Law; and simply because an invention is naturally of the indefinite nature he indicates, it is necessary to prescribe an arbitrary standard in order to produce something sufficiently concrete to work with. How, otherwise, is one to know to what the protection of the Patent Law applies? We thank him for pointing out a deficiency in our suggested definition of invention.

We come now to the vexed question of whether a Union patent should be defeated by mere knowledge or use abroad. Several contributors have discussed it; and we think Mr. White, in arguing for it, happily illuminates the case against it. As we understand him, he says that supposing a device to be in use abroad, that fact could only be used to defeat a patent here, if it (the fact) were known here. This, presumably, infers that the device itself is known here, in which case its use to defeat the patent is fully provided for by the other clauses of the definition. Furthermore, although the use abroad might be prior to the application here, the local knowledge of that use might be later than the application. This is really the crux of the objection; in other words, mere knowledge or use abroad is not a thing which enlightens the public of the Union, but it would simply be used to defeat a patent when the disclosure of the invention by the patentee had set rivals in search of an anticipation. We think that in discussing this point, both Mr. White and Mr. Cullen have overlooked the common phenomenon of independent invention by different people. The ban to which Mr. White refers applies, of course, only to introducers who are not themselves originators.

We have perhaps not made ourselves clear in our criticism of the ground of objection "that the invention is not novel or not capable of being patented." This, of course, could not be eliminated, but it is simply nonsense to say that an invention is not patentable because it is not patentable. The clause at once becomes rational if it reads "not novel or not patentable in terms of the statutory definition of an invention."

Objection (d) is one designed to meet a special and very rare case. Any other interested parties cannot have the special grievances here provided for.

With regard to objection (g) (that based on a defective title for the invention), does Mr. White seriously think that the matter is one of sufficient importance to warrant High Court proceedings? Are inventors so persistently deceptive or culpably negligent in labelling their inventions as to warrant every applicant being threatened with total loss of his rights because of the inaptness of half a dozen words to describe his invention? We agree with Mr. Cullen that this ground of objection, as well as (g) and (i), could well be left to the examiner.

It seems certain that the Bill will not come before Parliament until next year; so that for the present the separate provincial patents will continue, and meanwhile the Society will have ample time to make any representations it sees fit.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

PRECIPITATION OF GOLD AND SILVER IN CYANIDE SOLUTIONS BY CARBON. — "In treating tailings, which also contained half-burned coal, etc., from power plants, by fine grinding in a weak cyanide solution for the recovery of gold and silver, it was found that the coal caused first the selective precipitation of the gold and later of the silver. Experiments showed that ground clinker, half-burned coal, and charcoal were all effective, but that unconsumed coal-dust had no action. The action appeared more rapid with fresh carbon than with carbon that had been saturated with water. The trouble was overcome by passing the tailing over vanners to remove the carbon."—R. K. COWLES, *Trans. Australasian Inst. Min. Eng.*, xvi., 58.—*Journal of the Franklin Institute*, May, 1914, p. 532. (J. G.)

NEW METHOD FOR THE DETERMINATION OF SULPHUR IN IRON PYRITE.—"Treat 0·5 gm. of finely ground pyrite in a 150 cc. flask with 25 cc. *aqua regia* (equal pts. HCl and HNO_3) prepared a few minutes in advance. Allow to act several hours (preferably overnight) in the cold, add about 1 gm. NaCl, evaporate almost to dryness on a sand bath, take up with 2 cc. HCl and sufficient water, then add 50 cc. boiling water, and little by little 4 gm. Na_2CO_3. The precipitate is not bulky and does not retain sulphates when washed with hot water. To the filtrate add 2 drops methyl-orange (0·1 %) and neutralise exactly (rose tint) with HCl. Boil off CO_2, add 60 cc. of $Ba(OH)_2$ (15 gm. per litre) and a drop of phenolphthalein. Pass CO_2 (carefully washed), stopping the flow of gas the moment the rose colour just disappears. Filter, wash with hot but not boiling water, cool, and titrate the filtrate with $N/2$ HCl to a rose colour. Multiply the cc. HCl used by 0·008 to find the wt. S in the sample, or by 1·6 to find % S. To determine S in burnt pyrite, weigh out 3 gm. and proceed as before, but use 8 gm. Na_2CO_3 and 25 cc. $Ba(OH)_2$. To determine Pb, Cu and Zn in pyrites, mix 3 gm. in a porcelain crucible with 2 gm. S and 8 gm. $KNaCO_3$, cover with a little Na_2CO_3, put the lid on the crucible and fuse for 15 minutes. Extract the cooled mass with 300-400 cc. hot water, add 20 cc. HCl (22 ° Be), allow to digest several hours, remove and wash the crucible. Add 25 gm. crystallised sodium acetate, allow to settle, filter and wash with H_2S water containing acetic acid. Separate the ZnS by dissolving in HCl, precipitate as ZnS and weigh as ZnO. Burn the filter CuS PbS in a porcelain crucible, dissolve in HNO_3 and electrolyse; or, if preferred, separate the Pb as $PbSO_4$." ERNST MARTIN, *Min. Sci.*, *3*, 886-9. *Chemical Abstracts*, Feb. 10, 1914, p. 473. (J. G.)

HEAT TEST AS APPLIED TO EXPLOSIVES; FIRST REPORT TO THE HOME OFFICE OF THE DEPARTMENTAL COMMITTEE, APPOINTED APRIL 21, 1909.—"The report deals with the best means of standardising, with greater accuracy than at present, the apparatus and materials employed in the Abel heat test. The following are some of the chief alterations proposed by the committee. A copper heat test bath is to be provided with a constant level apparatus, with seven holes in the lid at equal intervals from each other and equidistant from the centre, one of the holes being for the thermometer. A cage of two crossed wires is fixed to the underside of each test-tube hole, and unoccupied holes are covered with small discs of copper. The French chalk used for incorporating with gelatinous explosives should not absorb more than 0·5 % of moisture under certain defined conditions, and 50 cc. should weigh 23·5 ± 1·0 gms. A Wedgwood mortar and pestle are recommended for the incorporation of gelatinous explosives. Specified times are suggested for the incorporation of blasting gelatin (1 minute), gelatin dynamite (6), and gelignite (3 min.) with French chalk. The new mill recommended for grinding cordite can be readily cleaned and the cutting edges can be accurately adjusted and easily repaired. A new design has been adopted for the water oven for drying gun cotton, etc., so as to allow a current of air to pass over the sample, and aluminium trays are recommended in place of paper trays. Detailed directions are given for the preparation of nitrocellulose pulp for heat testing. After preliminary pressing the sample is rubbed through a sieve on to an aluminium tray, and, after drying, should be sifted in the nest of sieves as used for ground cordite. The same paper should be employed for filtering operations in connection with the heat test as that used in the preparation of heat test paper. The chemical constitution of the paper and its physical properties, such as thickness, weight, texture, surface and length of fibre influence the result when the paper is sensitised for heat test purposes. Several samples of paper were specially prepared, but, with cordite, did not give results concordant with those given by Waltham Abbey batch 8 which was adopted as a standard. It is not sufficient merely to specify a filter paper, and the method of preparation of a heat test paper from it. It is essential that the sensitiveness of the finished test paper be verified by a test. At present, batches of heat test papers are tested by the Chief Inspector, Woolwich, for acceptance into the service, by means of a selected sample of guncotton and in comparison with a standard heat test paper. This is not entirely satisfactory, because of the possible variations in both the guncotton and the standard heat test paper used as a control, and the committee recommend the introduction of a test in which the heat test paper to be standardised is exposed to nitric peroxide, under standard conditions. This 'diffusion test' should be employed concurrently with the standard test. Standard tint papers should be prepared by using a permanent pigment. As light has the effect of reducing the time of the heat test, and as this effect is considerably more marked in the case of cordite than in other explosives, the exposed portion of the heat test tube should be covered with a ventilated cap of black paper. The heat test tubes should be of the best quality soda lime glass, free from lead and without streaks, and should, after heating for three hours at a steam pressure of 50 lb. per sq. in. show no more cloudiness than a standard sample so treated." *Journal Society Chemical Industry*, April 30, 1914, p. 441. (J. G.)

DETERMINATION OF CARBON MONOXIDE BY MEANS OF YELLOW MERCURIC OXIDE.—"Carbon monoxide can be oxidised quantitatively by slowly passing it over about 20 gms. of yellow mercuric oxide contained in a U tube, and maintained at 100° C. in a water-bath, and the resulting carbon dioxide absorbed in standardised baryta water. The mercuric oxide must be precipitated and dried at a low temperature, and protected from the prolonged action of light; oxide precipitated in hot solutions, or the red oxide, is inactive. About 45 mins. is required for the combustion of 100 c.c. of gas. Methane is unacted on by the yellow oxide at 100° C. and hydrogen, though partially oxidised, does not influence the titration, so that the method is available for the determination of carbon monoxide in presence of either of these gases."—L. MOSER and O. SCHMID, *Z. Anal. Chem.*, 1914, *53*, 217-233. —*Journal Society Chemical Industry*, April 30, 1914, p. 442. (J. G.)

EVOLUTION OF ELEMENTS. "The unknown element *nebulium* detected spectroscopically in the gaseous nebulæ by means of its violet lines, is now stated to have the atomic weight 3 or thereabouts.* The temperature of the Orion nebula is about 15000°. —BOURGET, FABRY and BUISSON, *Compt. rend.*, and *Chem. News*, 1914, 109. (J. M.)

* See Mott on nebulium, this *Journal*, Vol. X., 1909, p. 98; and Sir J. J. Thomson on the gas X_3.

DETECTION OF GOLD AND PLATINUM. — "The method of Pertusi and Gastaldi* for the detection of HCN with Cu acetate and benzidine acetate may, with a few changes, be applied to the detection of Au and Pt. With quite dilute solutions of Au salts the reagent (a solution of 1 g. benzidine in 10 cc. AcOH + 50 cc. H_2O) gives a deep blue coloration which gradually changes to violet. The coloration is green in the presence of free AcOH and is changed to blue with addition of benzidine in excess. A blue, flocculent precipitate (which settles readily and may possibly be utilised for quantitative determination) is produced in a short time after addition of the reagent to quite dilute solutions of Pt salts; the deposition of the precipitate (which is then darker in colour) is promoted by heating. Dilute solutions of Fe''' salts give with benzidine acetate a blue coloration which is only stable in the presence of excess of benzidine. Free mineral acids have no influence on the Au reaction, but retard the Pt reaction in the cold. When Au and Pt are to be detected in the presence of Fe, the latter should be removed by previous treatment with dilute HCl or HNO_3. The limit of sensitiveness of this reaction is 0·00000035 g. per 1 cc. for Au and 0·00000125 g. per 1 cc. for Pt." — G. MALATESTA and E. DI NOLA, Milan, *Boll. chim. farm.*, *52*, 461-3.—*Chemical Abstracts*, April 20, 1914, pp. 1397-8. (J. G.)

DETERMINATION OF TITANIUM AND THE INSOLUBILITY OF TITANIFEROUS ORES. — "Ilmenite is readily soluble in concentrated HCl. When ilmenite or iron ores which after treatment with concentrated HCl give residues which contain Ti, there is present in the ore or ilmenite mineral other Ti minerals (*e.g.*, rutile). The Ti can be precipitated from solution by heating and the following is proposed as a quantitative method: Dissolve 2·5 g. sample in 50 cc. concentrated HCl and heat ½—1 hour. Stir to prevent formation of insoluble precipitate on the bottom of the flask. Cool and dilute with 2 vol. water. Filter and wash with 2% HCl. Add about 20 g. Na_2CO_3 and carefully neutralise with NaOH solution, taking care that no permanent turbidity forms. Then add 25 cc. 2 *N* HCl and water containing SO_2 (about 50 cc.). Dilute to 1 litre and boil for half an hour and add more of the SO_2 solution. In one to two hours the precipitate formed will settle, then pour the supernatant liquid through a CSS 589, 15 cm. white band paper. Wash free from HCl and in doing so avoid caking of the precipitate to the paper. Ignite while still moist and weigh the TiO_2. The insoluble residue from the concentrated HCl solution should be examined for Ti. Fe and Al are not carried down by the precipitate. P, on the other hand, comes down quantitatively." —P. FARUP and E. SCHREINER, *Tidskrift Kem. Farm. Terapi*, *11*, 65-73.—*Chemical Abstracts*, May 20, 1914, p. 1722. (J. G.)

"FREE CARBON," ITS NATURE AND DETERMINATION IN TAR PRODUCTS.—"Results of experiments are described showing that benzene, toluene, chloroform, carbon bisulphide, and glacial acetic acid in contact with tar form insoluble compounds, the amount increasing with the period of contact: the residues obtained with chloroform and carbon bisulphide contained chlorine and sulphur respectively. No such insoluble compounds appeared to be formed with pyridine and aniline respectively, but these liquids are considered unsuitable as solvents for the determination of "free carbon," for which purpose digestion with toluene on a steam bath for not more than 30 mins. and extraction of the insoluble residue with benzene under standardised conditions is recommended. For asphaltums extraction with cold carbon bisulphide, using a Gooch crucible, is more rapid and gives results equally accurate."—J. M. WEISS, *Journ. Ind. Eng. Chem.*, 1914, *6*, 279-285.—*Journal of the Society of Chemical Industry*, May 15th, 1914, p. 474. (J. G.)

*This test is really due to Moir: see Proc. Chem. Soc., London, May 5, 1910, and this *Journal*, 1910, p. 542.

METALLURGY.

LOSSES OF DISSOLVED GOLD.—"The subject of losses of dissolved gold occurring in cyanide plants has received only a small proportion of the study and attention to which it is entitled from those persons who are in charge of the operations of cyanide plants. Visits to some of the larger of those plants of Central America, Mexico, and the United States have caused the truth of the previous statement to be forcibly impressed upon me.

Discrepancies due to Losses.—These losses are in some cases the cause of serious discrepancies occurring between the actual and theoretical recoveries, when the theoretical recovery is based upon the difference between head assays and tail assays. An example of such a case is taken from a well known mill in Honduras some years ago. At this place a usual difference varying from 3% to 5% occurred monthly between the actual and theoretical recovery. The theoretical recovery was based upon the difference between the head and tail assays, and it was usually the 3 to 5% greater than the actual recovery, mostly due to the unestimated soluble loss. The tail sample was taken from the Butters filter-leaf holding the washed cake, by means of a scoop attached to the end of a stick, the sample being scraped from one or two leaves of each charge by means of the scoop. The sample from each charge was dumped into a pan which held all of the charge samples for 24 hours. When the pan would be nearly full the sampler would pour off any solution from which the pulp had settled, in order, I presume, to make room for more of the charge samples. The worthlessness of such sampling is obvious, especially in a mill where the solution entering the zinc-boxes would assay over 20 oz. silver per ton, yet nevertheless I have seen the same or nearly the same method practised in more recent times in other well known mills. One can well imagine the extent of the undetermined loss which occurred at the mill in Honduras upon considering the losses as shown in the examples which follow hereinafter.

Tailing Sample.—A somewhat common method of sampling filter tailing is to take a cut or so from the washed cake before it is discharged from the filter-leaf. I have always found this method to be incorrect, more especially so with the Butters or Moore type of filter. Some reasons for its unreliability are that in taking the sample the sampler usually selects a cake which is not frozen, that is, one which does not adhere to the opposite cake contained on another leaf; coarse or heavy particles of ore which have a tendency to concentrate close to the filter medium during cake making, are not caught in the sample owing to the unevenness of the filter surface caused by slats or coco matting; and the failure to obtain solution in the sample which was discharged with the cake and which remained in the pipe connections and spaces of the filter-leaf at the end of the wash before discharging.

A number of carefully conducted comparative tests which I have made have proved in each and

every case that the leaf sample was lower than a properly taken flume sample.

The following tests were made at a mill treating about 1,000 tons of ore per day using a Butters filter which made a 60 minute cake with a 60 minute wash, and are examples of the foregoing statement. The unwashed pulp entering the agitators rarely assayed over 8d.

Charge	Sample	Tail Assay Washed.	Tail Assay Unwashed.	Solution. Soluble loss by difference.	Solution. Assay.	Solution. Extract Per Cent.	Solution. Soluble by Sand pump Assay.
A	leaf	0·035	0·035	none	0·063	1·04	0·0034
A	flume	0·035	0·05	0·015	0·027	0·91	0·0245
B	leaf	0·027	0·03	0·0033	0·0045	1·08	0·0019
B	flume	0·035	0·045	0·01	0·013	1·0	0·013

The method used in sampling at this mill prior to the operators learning of the unreliability of the leaf sampling, some few months, was very similar to the method described in the first part of this article. According to the difference between the solution solubles of the leaf samples and the flume samples as shown by charge B, the company operating the mill was losing over £61,000 per year more than it was aware of. The loss was mostly due to frozen cakes, and a few simple changes in the filter and filtering overcame most of the loss which had theretofore been unrecognised. The following figures show the smallest difference found between the soluble of the frozen and non frozen cakes:—

Charge	Sample	Tail Assay Washed.	Tail Assay Unwashed.	Solution. Soluble loss by difference.	Solution. Assay.	Solution. Extract Per Cent.	Solution. Soluble by Sand pump Assay.
C	leaf	0·027	0·03	0·0033	0·0045	1·08	0·0019
C	frozen leaf	0·027	0·05	0·023	0·036	0·92	0·033
C	flume	0·035	0·045	0·01	0·013	1·0	0·013

A considerable difference is shown in the foregoing figures between the soluble loss as shown by solution assay and that as shown by the difference between washed and unwashed sample assays. This difference is partly due to the difficulty of obtaining close assays when charges of only one assay ton size are taken for assay. The main reason for the difference is probably due to the method used in drying the unwashed sample. The possibility of error in soluble determination by the differential method arising from the method followed in drying samples containing soluble metal is shown by the following:—

Flume sample C was thoroughly mixed and divided into four portions. Portion 1 was dried in a clean tin pan in the customary manner of ordinary practice. Portion 2 was enclosed in oiled paper and dried, the paper was reduced, and its gold content found and calculated. Portion 3 was treated with acid copper sulphate followed by sodium sulphate and a little sodium sulphide to precipitate the metals, then it was dried and assayed. Portion 4 was mixed with water, filtered, and washed in a filter press, the filtrate measured and assayed, and the residue dried and assayed. The following results were obtained:

Soluble by difference between washed and No. 1 10.00c. per ton.

Soluble by difference between washed and No. 2 19.63c. per ton.

Soluble by difference between washed and No. 3 27.69c. per ton.

Soluble by solution assay (No. 4) 26.87c. per ton

Another example is shown by the following:

Sample dried as ordinarily = 16.33c. per ton soluble.
Sample dried in oil paper = 20.95c. per ton soluble.
Soluble by solution assay = 24.80c. per ton.
Washed sample = 28.68c. per ton.

The above figures clearly show in the case of the particular ore that drying samples in clean tin pans does not give correct results. This is probably due to the concentration or precipitation of the soluble metal on the pan. In these cases an electric stove was the heat medium for drying.

I have noticed the same effect in drying silver ore in Honduras, but as I cannot find record in my notebooks I am unable to give figures.

The subject of drying samples containing dissolved metal is quite interesting from the evidence shown by the foregoing examples, and it should be investigated more thoroughly than I have investigated.

The soluble loss of cyanide varies considerably, and the rate does not follow any fixed rule. In a mill where water is none too plentiful and where solutions are not wilfully discarded, the soluble loss of cyanide through a Butters type of filter averages about 0·6 lb. KCN per ton when the circulating solution averages about 1·2 lb. KCN. The soluble loss in cyanide is quite an item, and I believe that it will be overcome to a great extent within the near future.—H. R. Layng. *Mining and Scientific Press*, May 30, 1914, p. 801. (H. A. W.)

Balanced Impeller for Sand Pumps.—"While the introduction of sand pumps was a step forward in the lifting of solutions carrying solids of varying size, their use has often been attended with some troublesome features. Among the latter, the most prominent, perhaps, was the excessive thrust from the impeller, for heads over 30 ft. This resulted in loss of power and caused a leak around the shaft, through the stuffing box, which in turn rapidly cut the stuffing box and the shaft. The Traylor Engineering and Manufacturing Co., Allentown, Penn., has developed what it calls a balanced impeller, for use in its sand pumps, and which is designed to overcome these difficulties. This impeller has slots in it, near the centre, which permit the solution to pass through it and thus to equalise the pressure on each side. The vanes are cast on the impeller in such positions that when rotating, they help to deflect the water or solution into and through the slots, the latter having edges bevelled in the proper direction to aid in this purpose also. It is claimed that the result of this design has been to remove the thrust on the impeller and to make unnecessary the use of a thrust collar or thrust bearings. The pumps containing these impellers remain single suction pumps, as before, but the suction action on each side of the impeller is the same, resulting in the stuffing box being used to keep air from entering the pump, and not, as before, to keep the solution from getting out.

In the pump itself, some improvements have also been made in the method of holding the linings in place. The latter are now held and adjusted by set-screws from the outside, instead of by bolts, making it possible to cast the liner castings without any openings in which sand might eddy."—*Engineering and Mining Journal*, May 16, 1914, p. 1069. (G. H. S.)

The De Lassaloe Leaching Process used by the Rhaisdell Cocotitlan Syndicate at Pachuca, Hidalgo, Mexico, is a hyposulphite cupric chloride leaching process. It is especially adapted to leaching tailings containing organic material that would cause excessive consumption of cyanide." *Engineering and Mining Journal*, May 9, 1914, p. 954 (G. H. S.)

MINING.

FACTOR OF SAFETY.—"How many of those who habitually use the term 'factor of safety' clearly understand its meaning?

Mr. C. E. Stromeyer, before the Staffordshire Iron and Steel Institute, points out that the expression, 'factor of safety,' was originally introduced for the purpose of inspiring confidence. It is generally understood that a machine or appliance such as, for example, a haulage rope or a bridle chain, with a factor of safety of say five, had been tested to five times the maximum stress for which it was designed, and in this general sense the term implies a margin of safety which seems to be an efficient safeguard even against occasional overloading.

On the Continent it is more usual to employ the expression 'permissible working load,' or some such term, instead of 'factor of safety,' and there is much to be said in favour of this practice. The factor of safety in its relation to winding ropes has recently been widely discussed in the German technical press, one of the main points raised being the influence of depth, and, consequently, of the increased weight of the rope itself upon the normal factor of safety. Col. Blackett alluded to the same subject when the general regulations were before Lord Mersey. The regulations provide that the strength of cappings, in the case of haulage ropes, shall be based upon the ultimate strength of the rope, where as the owners contended that the guiding factor should be the maximum load, as in the case of winding ropes. The distinction is subtle, but not immaterial.

The term 'permissible working load' possesses the advantage that a definite limit of safety is guaranteed by the manufacturer's tests. 'Factor of safety' virtually means nothing at all. It is a figure obtained in part by laboratory tests, in part by mathematical calculation based upon assured knowledge which is frequently wide of the truth. As Mr. Stromeyer says, 'the mathematical formulæ used by engineers are of the crudest, and do not correctly determine the actual stresses produced by external forces.' It is generally assumed that the factor of safety is, for example, a piece of metal, is the ratio of ultimate stress to elastic limit. Thus if the ultimate breaking stress is 20 tons, and the elastic limit is 4 tons, we say that the factor of safety is five. But it is not by any means an easy matter to determine either of these figures with accuracy. The utmost that can be done is to take the mean of a number of tests often widely divergent. Mr. Stromeyer, indeed, goes so far as to say that in certain cases the figure called 'factor of safety' is made up of 50% of mathematical convenience and 50% of ignorance as to the true value of the elastic limit. The truth is that the elastic limit is a somewhat inconstant quantity. As we now know from the results of fatigue experiments, the elastic limit is reduced by repeated stress, and this is a consideration which possibly assumes great importance in connection with such appliances as wire ropes.

Then we have the influence of the law of probability. There are innumerable small causes which may influence elastic limit one way or the other to a slight degree. These may generally tend to neutralise each other, but under certain conditions they may be cumulative. Thus if there are ten of these minute influences at work, each capable of influencing the elastic limit either positively or negatively to a slight extent, say, 1 ton, this will give 2^{10} or 1,024 possible combinations of chances, giving a total range of 10 tons in the break down point, and there is one chance in 1,024 that all the influences will be adverse. Similarly we may calculate other chances, with the result that there is always a possible error, and also a probable average error, in the experimental determinations upon which the 'factor of safety' figure is based.

Let us take another case of the greatest importance in mining operations, viz., the difference between static and dynamic stress, that is, between dead loads and live loads. In the text-books it is usually shown that a load which is applied suddenly results in a stress which is twice as great as that which a gradually increasing load will produce. Mr. Stromeyer believes that this is not true in all cases, and that suddenly applied loads often produce a much greater stress than double the static effect. We are here apparently in a state of profound ignorance, and have been too ready to deduce a general law from insufficient data. This question came recently into prominence in Mr. W. Walker's report on an accident at Rufford Colliery, due to the failure of a spring hook under a sudden jerk. The factor of safety as shown by testing was about 4·3, and it was admitted to have been insufficient under the circumstances of dynamic stress. It was, in fact, recommended in this report that ten should be a minimum factor of safety in connection with all ropes, chains, hooks, etc., used in such work as sinking operations. Even this margin is quite uncertain when questions of momentum have to be considered. Very similar conditions prevail in connection with the strength of winding ropes and more particularly of cage cappings. Breaking strains in such cases, as found by testing under a gradually increasing load, give but little information as to the results of a violent jerk, such as may readily be experienced in practical use.

The above examples go to show how mythical the 'factor of safety' figure may be. Permissible safe load would seem to be a far more trustworthy guide in practice; but even in this case the term has only a limited application on account of changes due to the ageing of materials in use. A steel rope may start from the factory with a high breaking limit, but this becomes continually lower in use, partly as a result of continued and repeated stresses inducing fatigue, partly by reason of deterioration of the material. The latter factor is especially serious in the case of certain metals and alloys entering now somewhat largely into the construction of machinery of all kinds. Mr. Stromeyer remarks that the risk of ageing is so great that some engineers are becoming shy of using certain nickel-steel alloys and some brasses and bronzes, because these sometimes become absolutely rotten after a time, even although they may be perfectly satisfactory when new. There is much yet that remains to be investigated in the mysterious diseases to which some metals seem to be liable."—*Colliery Guardian*, May 29, 1914. (H. M.)

CYLINDRICAL ORE CHUTES OF WOOD STAVES.

Synopsis—Reasons of economy forced a change at the South Mine, Broken Hill, from ore chutes of square section to a cylindrical type with vertical staves. These are longer lived, cheaper and can often be recovered when a stope is finished. Round timber for staves preferable to sawed. Methods of preparing staves and assembling chutes described. Four types of chute evolved with varying diameters and thicknesses adapted to varying conditions. Inclined chutes receive metal reinforcement. Special protection at point where vertical chute changes to inclined. Methods of repair. Costs. Special applications.

"The ore chutes formerly used in the South Mine at Broken Hill, New South Wales, were either square sets lined inside and out in the timber stopes, or squared timber cribbing in the open stopes with 2 in. spaces between adjacent pieces. They wore so rapidly that they did not last for the life of the stopes unless built at frequent intervals. Their repair either by replacement or by relining was difficult, slow and expensive.

The fact that these defects existed and that it was necessary to provide chutes for 150 ft. stope lifts forced the adoption of a cylindrical chute, made of a stringy-bark timber set on end, in spite of prejudice against it.

In this design each section of the chute is practically a barrel, or a small vat without bottom or top, made of staves with bevelled edges. The position of the grain of the timber being vertical, cutting action is a minimum. The staves are bound together with double hoops of fencing wire until the filling surrounds and holds them, and are built up in sections as required.

After experimentation, four standard types of chutes were evolved: (1) Chute of 4 ft. internal diameter, and 10 in. thickness; (2) chute of 3 ft. internal diameter and 10 in. thickness; (3) chute of 3 ft. internal diameter and 6 in. thickness; (4) chute of 3 ft. internal diameter and 5 in. thickness. The length of staves is in all cases 4 ft. This length is light and handy, even in the thickest sizes, and does not hang up when thrown down a winze into a stope; it is furthermore an aliquot part of the average height which the chute must be raised for each filling operation. While sections of this length are rather short, they have not proved unstable in practice, and in case of any distortion through a fall of ground, only a 4 ft. section has to be restored. The three thicknesses are adapted to varying conditions, 5 in. being the minimum permissible for sufficient bearing surface between sections and for resistance to collapse. The 4 ft. diameter type is partucularly available for the bottom portions of chutes, as it minimises the danger of 'hanging up.'

Applications of Types. Type 1 is used up to a height of 35 ft. in open stopes for both vertical and inclined chutes. Type 2 is used in open stopes for vertical chutes for the sections from 35 ft. to 70 ft., when the stope lift is 150 ft.; it is also used in modified style for all heights above 35 ft. in inclined chutes. Type 3 is used in square set stopes for the section from 35 ft. to 70 ft., where the stope lift is 100 ft. and from 70 ft. to 120 ft., where the stope lift is 150 ft. Type 4 is used for effecting repairs to old crib chutes which have failed. The fact that no type of chute is specified for the last 30 ft. of stope lift is due to the fact thatth is portion of the deposit is removed by underhand stoping requiring no chutes.

The staves are made from either round or sawed stringy bark, the latter, 10 in. × 5 in.; 10 in. × 6 in., or 10 in. × 10 in. For green sawed timber the templates are used, according to the cross section. The 10 in. × 10 in. piece, when cut, is reversed and put together. If the stringy bark is seasoned on the surface, an advisable proceeding, it shrinks so that different templates have to be provided. The shrinkage causes the use of an extra stave in a chute, 10 in. in thickness, but makes only a slight alteration in size in the thinner sections." ANDREW FAIRWEATHER, Proc. Aus. Inst. Min. Eng. (*Engineering and Mining Journal*, June 6, 1914, p. 1139. (G. H. S.)

COAL DUST EXPLOSIONS BY FRICTIONAL ELECTRICITY.—"A suggestion was made by Mr. W. A. D. Rudge in a recent issue of *Nature*, that a possible cause of explosions in coal mines lay in the fact that a cloud of coal dust when projected against an insulated conductor might give rise to sparks several centimetres in length. Experiments in the laboratory with various kinds of dust had shown that the potentials might easily rise high enough to make this possible. During tests made in Ludlow Pit, at Radstock, with a Wulf electrometer provided with a radium-tipped collector, potentials of 289 volts or more were observed behind a train of wagons, but no sparks were noticed. It should be pointed out, however, that the coal dust in mines is rarely pure, and that coal dust mixed with stone dust does not give rise to a high charge; as the sign of the charge depended on the dust material, and coal dust and stone dust imparted opposite charges to the conductor, the mixture was comparatively harmless. These results seem to supply a strong argument in favour of the use of stone dusting in coal mines."—W. A. D. RUDGE. (*Indian Engineering*, June 27, 1914, p. 350. (J. A. W.)

GREAT SALT LAKE PRESERVES TIMBER.—"In replacing a railroad trestle recently burned along the north shore of Great Salt lake, engineers have just found that the piles are still perfectly sound after 43 years of service. Looking for the cause, since these were only of local pine and fir, they found the timbers were impregnated throughout with salt from the lake.

At another point on the lake, 18 in. piles, set 29 years, are similarly preserved with salt which has penetrated to their very centre. Timbers in the Southern Pacific trestles across Salt lake, placed in 1902, appear to be as good as on the day when the piles were driven. They have been preserved well above water line by the salt dashed onto them by the waves, a fact apparently anticipated by the engineers who built the trestles.

The first transcontinental telegraph line, built before the railroad, extended west from Salt Lake City through the prosperous mining camps of Eureka, Austin and Virginia City. When the railroad was built, the telegraph line was transferred to follow its right of way and the old poles sawed off at the ground. An engineer who recently examined the butts left in the ground in the salt desert near Fish Springs found that, although 50 years had passed since the poles were cut off, the old butts were perfectly sound.

Telephone and electric companies in the Salt Lake valley have used the local salt for preserving poles. When set up, about 75 lb. of salt is placed around the pole on the ground. This method cannot be used, however, when the pole is on or near a lawn, or in any place where vegetation is desired.

It is pointed out that the reason why the waters of Salt Lake act as a strong preservative, as distinguished from ocean waters, is because the lake water is so much saltier, being practically a saturate solution. Preservation with salt is of no use in ocean piling against the attack of teredos and other marine borers.

Experts in the forest service who have been investigating the preservative treatment of timber offer the suggestion that ties and poles which have been immersed for some time in the waters of the lake ought to be impervious to decay if the salt is not leached out by the action of the elements. It has been suggested that this can be guarded against, for

example, by painting the butt of the pole with a coat of creosote which will keep out the moisture and keep in the salt." — *Mining Science*, June, 1914, pp. 34-35. (J. C.)

MISCELLANEOUS.

Radium and Its Sources.—"The rays of radium are divided into three classes, according to the physical properties exhibited by each.

Alpha Rays.—These have little penetration, and are stopped, or absorbed, by an ordinary thick sheet of paper. They have very little photographic effect. They are electrically positively charged particles. They are the strongest ionizers, *i.e.*, when allowed to pass through a non-conducting gas, they render it conducting to electricity—this effect is called ionization. In a hundred radium particles ninety of them are alpha rays.

Beta Rays.—These have more penetration than alpha rays, but are absorbed by a thin sheet of copper or tinfoil, or 5 mm. thickness of aluminum. They have the greatest photographic effect, but less power of ionization than alpha rays. They are negatively charged particles. In a hundred radium ray particles nine of them are beta rays. These rays are found to stimulate the growth of cancer.

Gamma Rays.—These have the greatest power of penetration, and will pass through 30 cm. of iron, or a ½ in. steel bar, without apparent diminution in intensity. These rays travel through space at the rate of about 185,000 miles per second. They have little photographic action, and little power of ionization. They do not appear to be charged with electricity. These are the rays in the cure of cancer. By wrapping metal foil around the tube of radium salts the alpha and beta rays are excluded from passage, while it is no obstruction to the gamma rays. In a hundred radium particles one of them is the gamma ray, but it is the most wonderful of all.

Of the radioactive elements radium has the strongest rays—about 1,000,000 times more active than uranium.

Radium exhibits the wonderful power of continuous emission of energy in the form of heat without diminution of its substance, and from no source except itself. It is found that one-half grain of radium bromide gives out heat at the rate of two calories per hour, and radium therefore doubtless contains a greater store of energy for the world's use than any other supply known to-day.

Radium is usually placed on the market in the form of a radium bromide, or radium chloride—the latter containing 76% radium, and the former less than 59%.

The ratio of radium present to uranium in any unaltered mineral, as pitchblende, is constant, and stated by Rutherford and Boltwood to be 0·0000003 gm. of radium to 1 gm. of uranium—or, in other words, 1 ton of 60 % uranium ore will contain two-tenths of a gram of radium. An exception to this rule has recently been discovered in the case of some Colorado carnotite.

It therefore takes about 215 tons of 2 % uranium ore to produce 1 gm. of radium—if 100% is saved.

More than forty different uranium minerals have been discovered of which pitchblende, carnotite, autunite and torbernite have been of commercial importance—pitchblende and carnotite having thus far produced practically all the radium we have.

Pitchblende is the richest ore of uranium—frequently carrying between 50% and 80% uraninm. It is essentially the black oxide of uranium, but always carries many impurities. Uraninite is a crystalline variety of pitchblende.

Pitchblende in commercial quantities has quite invariably been found in fissure veins in igneous rocks, as granite, gneiss, schists, etc., and always associated with gold, silver, copper, lead, zinc, tin, and various sulphides, or some of the foregoing. Its colour is pitch black to brownish, sometimes with greyish or greenish cast; brittle; lustre dull, metallic, pitch-like; fracture, conchoidal to uneven; hardness 5 to 6; specific gravity 9 to 9·7 in crystals, or 6·4 and higher in massive form.

Carnotite is essentially a hydrous vanadate of uranium and potassium, but without standard chemical formula—probably a mixture of minerals, the exact nature of which analysis fails to reveal. Colour, bright canary yellow. Found in a powdery form, or in slightly cohering masses, or as impregnation between the sand grains, or as yellow stain in rock crevices, or in calcareous sandstone.

Until very recently buyers have refused to handle ores containing less than 2 % U_3O_8—but there is now some talk of reducing the limit to 1½ %. Carnotite ores, as shipped from Paradox Valley, have averaged about 2·5 % U_3O_8, and 3·5 % V_2O_5.

While prices have been variable, yet as based upon the uranium and vanadium content, perhaps, the following scale would be an average:

2 % U_3O_8	...	81.30 per lb.
2½ % U_3O_8	...	1.40 ,,
3 % U_3O_8	...	1.50 ,,
V_2O_5	30 to 50c.	,,

If scale based on U_3O_8 alone, then 82.00 to 82.50 per lb., but on 5 to 10 % U_3O_8 82.75 per lb., and for 10 to 20% U_3O_8 83.00 per lb.

The probable maximum cost in Europe of extracting radium from carnotite is 820 per mgm., corresponding to 820,000 per gm.

The average price of radium bromide is perhaps 870 per mgm., corresponding to 870,000 per gm., or 82,000,000 per oz., and for radium chloride 890,000 per gm.

During the year 1912, for which approximate statistics are available, it is certain that American ores produced more than twice as much radium as all other sources combined, and practically all this ore was mined in Colorado.

That the United States have produced such a large proportion of the radium ores does not necessarily imply a paucity of such ores elsewhere; but is rather in the nature of a handsome compliment to the superior energy and enterprise of American prospectors and mines, as, even at this early stage of the industry, it has become apparent that such ores are widespread and abundant, having already been found in South Dakota, Montana, Washington, Idaho, Utah, Colorado, New Mexico, and Arizona; also in Germany, Bohemia, Austria, Norway, Sweden, Russia, Turkestan, Portugal, Peru, England and Australia."—Chester T. Kennan.—*Mining Science*, June, 1914, pp. 43-45. (J. C.)

Hardening the Surface of Concrete Floors.—"A new hardening material for the surface of concrete floors is described as containing 95 % of iron dust or iron flour, and it is added to the dry cement in the proportion of 15 to 25 pounds to each 100 pounds, one part of the mixture being used with two parts of sand. This preparation is applied as a top coat to a thickness of ½ to 1 in. It forms a hard and durable floor, claimed to be waterproof and not slippery, and is also used for making new concrete adhere to old concrete in repairing."—*Chemical Trade Journal*, April 18.—*Indian Engineering*, June 27, 1914, p. 359. (J. A. W.)

New Books.

Burns, Daniel. Electrical Practice in Collieries. 4th Edition, Revised and Enlarged. Cr. 8vo., pp. 364. *C. Griffin.* Net 7s. 6d.

Cranjon, R., and Rosenberg, P. A Practical Manual of Autogenous Welding (Oxy-Acetylene). 2nd Edition, Revised. 8vo., pp. 256. *C. Griffin.* Net 5s.

Davies, G. Humphrey. The Rating of Machinery. 2nd Edition. Cr. 8vo., swd. *Manufacturers' Association.* Net 2s.

Thorns, A. H. The Principles of Metallurgy. 2nd Edition. Cr. 8vo. *Macmillan.* 6s.

Smith, H. G. Minerals and the Microscope. An Introduction to the Study of Petrology. Illustrated. Cr. 8vo., pp. xi.-116. *Murphy.* Net 3s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 254/14. Carl Travers. Improvements in respirators. 12.9.14.

(P.) 255/14. James Wilshir (1), The Denver Rock Drill and Machinery Co. (2). Improvements in devices for passing water into rock drills. 13.6.14.

(P.) 256/14. Donald McLachlan. Improvements in detachable bits for hand and machine rock drills. 13.6.14.

(P.) 258/14. Thomas Patrick Stanton. Improvements in mining and reducing ore. 16.6.14.

(C.) 259/14. Paul Selby. Improvements in spraying apparatus. 17.6.14.

(P.) 260/14. Frank Hill. Improvements in rollers for mine shafts, haulage tracks, and the like, for supporting hoisting, hauling or other ropes or cables. 17.6.14.

(P.) 262/14. James Henry Ruthven. Improved automatic emergency safety gripping brake for use on skip guide rails, ropes and the like, and on winding drums, wheels and the like. 18.6.14.

(C.) 263/14. Emily Grace Wilson. Improvements in and connected with internal combustion engines. 18.6.14.

(C.) 266/14. Benjamin Graemiger. Improvements in driving machines by electro motors and the like. 18.6.14.

(C.) 267/14. Leon Hirschberg (1), Potential Developments, Ltd. (2). Improvements in or relating to the treatment of hydrocarbons. 18.6.14.

(C.) 268/14. John Emery Bucher. Improved process for producing cyanides. 18.6.14.

(C.) 270/14. Adrien Armand Maurice Barriot. Process for the treatment of ores and solid salts by electro-chemical reduction. 18.6.14.

(P.) 272/14. James Bennett (1), William Edward Elbeshausen (2). Improvements in closure devices for making bottles and the like non-fillable. 18.6.14.

(P.) 276/14. Richard Morrison McDonald (1), Frederick Claude Stuttock (2). Improvements in means for preventing the dissemination of dust from rock drills. 23.6.14.

(P.) 277/14. William Fyfe. An improved method of fastening pulleys to cam shafts in stamp batteries. 23.6.14.

(C.) 281/14. George Tom Cannon. Improvements in rope grips or jockeys for mechanical haulages and the like. 26.6.14.

(P.) 282/14. John William Birtley Craggs (1), William Campbell (2). Improvements in jockeys or rope grips for mechanical haulages and the like. 26.6.14.

(P.) 283/14. Lancelot Ussher (1), Armour Hall (2). Improvements in pipe and tube manifolds and the like. 27.6.14.

(P.) 285/14. Fabian Mercer Cox. Jig concentrators. 27.6.14.

(P.) 287/14. Harold Sharpley (1), Kenneth Norman Lambert (2), Richard George Hillier (3). An improved means or apparatus for transporting granular and like materials. 27.6.14.

(P.) 288/14. John Christian Reul. Carboline, a compound powder to prevent the formation of carbon in motor engines. 27.6.14.

(P.) 289/14. John Conrad Lind. Improvements in and relating to the treatment of waste dumps of collieries to recover the products of combustion therein. 29.6.14.

(P.) 291/14. John Christian Reul. Carbolons. A compound powder for preventing the formation of carbon in motor engines. 1.7.14.

(C.) 292/14. Gotthilf Ausgarius Betulander (1), Nils Gunnar Palmgren (2). Improvements in automatic or semi-automatic telephone exchange systems. 2.7.14.

(C.) 293/14. Aubrey George Eneas. Improvements in or relating to spray nozzle and method of distributing liquids. 2.7.14.

(P.) 295/14. Seguin Fisher. Improvements in passing fluids into reciprocating rock drilling tools. 4.7.14.

(P.) 296/14. Ambrose Spinnley. Improved clamp for boring water mains. 6.7.14.

(P.) 297/14. John Pountney Udal. Improvements in blasting cartridges. 7.7.14.

(P.) 298/14. James West (1), John Edward Jones (2). Improvements in valves used in jigging and other similar machines. 7.7.14.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in addresse otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non receipt of Journals and Notices.

DONALD, H. S. B., *to* Rose Deep, P. O. Box 6, Germiston.

LESLIE, H. M., *Ho* Scotland; Roseneath, Empire Road, Parktown.

MACKENZIE, J. D., *f.o.* Johannesburg; P. O. Box 26, Randfontein.

NEILL, J. M., *f.o* Benoni; Geduld Proprietary Mines Ltd., P. O. Box 41, Springs.

STEELS, S. H., *f.o* Roodepoort; Prestia Mines, Ltd., via Sekondi, Gold Coast Colony, West Africa.

STUDOLEH, N. H., *to* Concession Hill Mine, Hartley, Rhodesia.

TILLARD, M. O., *to* 58, Schmidt Street, Bellevue East.

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THE JOURNAL

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Vol. XV SEPTEMBER, 1914. No. 3.

Proceedings AT Ordinary General Meeting, September 19th, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building on Saturday, 19th September, Mr. W. R. Dowling (Past-President) in the chair. There were also present:—

23 Members: Messrs. R. A. Barry, Prof. J. S. Cellier, F. W. Watson, John Watson, H. A. White (Members of Council), W. Beaver, J. A. Campbell, W. M. Coulter, J. M. Dixon, H. W. Gill, J. T. Guiney, B. J. Hastings, W. E. John, G. Melvill, P. T. Morrisby, S. Newton, E. A. Osterloh, W. S. V. Price, E. Roberts, D. F. Scott, A. L. Spoor, J. A. Taylor, and I. Tom.

7 Associates and Students: Messrs. D. L. Allan, J. Gibson, J. M. Gourley, A. King, F. J. Pooler, W. J. Sutherland, and A. Willcox.

6 Visitors, and Fred Rowland (Secretary).

THE PRESIDENT.

The Secretary: Prof. Stanley has asked me to apologise for his unavoidable absence this evening owing to the illness of his wife.

MINUTES.

The Minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBER.

Messrs. H. W. Gill and P. T. Morrisby were elected scrutineers, and after their scrutiny of the ballot papers the Chairman declared that the candidate for membership had been elected, as follows:—

DEVEES, A. F. H., P. O. Box 5598, Johannesburg. Scientific Instrument Maker.

The Secretary: Since the last meeting the following Associates have been admitted by the Council:—

RUGGLES-BRISE, HAROLD RALPH, Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Mining Student.

LINDBERG, ANDERS WILLIAM, Ferreira Deep, Ltd., P. O. Box 1021, Johannesburg. Chief Electrician.

PRENTICE, THOMAS KENNEDY, Victoria Cottage, 12th Avenue, Mayfair, Johannesburg. Reduction Works Employee.

VOTES OF CONDOLENCE.

The Chairman: We have to record with sincere regret the death of Lord de Villiers, the Chief Justice of the Union, and, at the time of his death, Acting Governor-General. Lord de Villiers, with his high qualifications as a lawyer and judge, in addition to fine qualities as a man, took an important part in the deliberations at Capetown when the various Colonies of South Africa decided on forming the existing Union. He was a man who could be ill spared in these troublous times, and it will be difficult to fill his place.

You are also asked to express your sympathies at the unfortunate deaths of General de la Rey, Dr. Grace, and the police officers who met their deaths in the recent tragic events, by rising from your seats.

Those present signified their sympathy by rising.

GENERAL BUSINESS.

THE EUROPEAN WAR.

The Secretary: You will remember at the last meeting it was agreed that the Society should offer the Government any assistance possible in its power during the present crisis. That offer was conveyed to the Government through Mr. Malan, the Minister of Mines and Industries. Mr Malan writes to us that he wishes to convey his sincere thanks to the Society and

to say that the Government highly appreciates the Society's offer of assistance. As one of the methods of assistance to the Government at this time, the Council has given its office, and my services to a large extent, to the work of the S.A. Red Cross Society. (Applause.) I might say that the S.A. Red Cross Society has organised the making of comforts for the troops throughout the Transvaal, in Natal, and in portions of the Free State, and the Cape. (Hear, hear.) A large number of parties of ladies are busily engaged in preparing comforts, and we have to gratefully acknowledge the services of not only the ladies but also the gentlemen who have assisted from the Rand and elsewhere in the matter. A large number of parcels has been sent out. The Red Cross Society is unable to distribute articles to combatants in the field, but consequently it is posting them, as well as being done by various institutions which are affiliated to the Red Cross Society, namely the Defence Force Comforts Association. The first week it was sent out the Society posted 1,350 parcels of socks, scarves and mufflers to the value of £1,000. In the second week 108 parcels of socks, scarves and mufflers were posted to the Red Cross Depot by 106 parcels of socks, mufflers and scarves posted direct to the men at the front. At present socks are still required, and plenty of them, as the men are not fully equipped in that respect by the Government when they leave. (Applause.) In spite of requirements in other directions, and the fact that the Society has not yet posted to the men at the front the number of articles which will be ultimately sent, we are, I understand, in a position very well to address ourselves to the task. (Hear, hear.)

Mr. H. A. White *Mr. President, Gentlemen.* We are here to-night to-day to hear the Red Cross Society's case put forward in an exceedingly good way by the representatives of the Society. I feel sure that all of us present here to-day will be pleased to assist the Society in every possible way that we can, and I understand that the committee will be glad to receive any suggestions that we may make. I therefore have much pleasure in moving a vote of thanks to the Red Cross Society's representatives for the very able manner in which they have put their case before us. (Applause.)

SOME NOTES ON TREATMENT OF ZINC SLIMES.

Mr. H. A. White *Mr. President, Gentlemen.* This subject has been so fully threshed out in the past by various members of our Society that the object of the present notes is merely to place on record some recent improvements and to give an opportunity for bringing the whole subject up to date in our *Journal*.

1. The use of bisulphate of sodium has now become so general that the Modderfontein supply (which is now about 28% free acid) is not always adequate to meet the demand. It is, however, possible to obtain supplies from either of the other two dynamite factories in this country, that from Natal containing up to 40% sulphuric acid. The price is naturally somewhat higher than that of the local product, but shows a great saving over C.O.V.*

2. White precipitate is present in the boxes in considerable quantity, special precautions are advisable in the acid treatment. These include (a) emulsification of the white product taken from the filter press and the avoidance of the formation of greasy lumps which are very slowly acted upon by the acid. (b) Owing to the fact that the action of sulphuric acid is somewhat slow, especially in the presence of heavy nobles, it is advisable to allow at least twelve hours' contact with hot acid in excess and to receive some of the coarser zinc shavings, to be dissolved, for subsequent neutralisation of the remaining acid. A considerable improvement in the value of the final slime is easily effected in this way.

3. The consistency of the clay borers now in common use may be observed to be very rapidly increased in the vicinity of cracks, and a penetration may take place in a liner still showing considerable thickness in other parts. In order to avoid the formation of these cracks it is essential that the liner should extraneously be evenly supported and heated. To attain this object, as the liner never fits the pot exactly, it is well to fill up the space with powdered graphite, using crucible pots for the purpose. First pour a little of the powder in the bottom of the new crucible and press the liner into position, then with a circle of zinc sheet cut to fit over and protect the inside of the liner, the powder may be poured in and rammed with a stout wire till the crucible and liner to the top of the liner. Care must, of course, be taken to remove any graphite left inside the liner, which, however, rarely happens. Liners properly supported in this fashion

* Common Acid of Natal.

will give up to 50 more "fusions" than are usually obtained.

(4) On some mines it is not found necessary to add any flux or oxidiser whatever to the well calcined gold slime as it is self-fluxing and contains sufficient ferric oxide to keep the value of the bullion produced well above the level of the usual battery bars. The slag thus produced is fairly low in gold, but owing to unavoidable variations in the composition of the calcined slime, even on the same mine, trouble may be caused by insufficient fluidity of the slag or by undue corrosion of the liner.

A method allowing a little more elasticity in conditions is to fill the liner within half an inch with the gold slime and to pour on this, as a protective cover, a mixture of 1½ parts of fine sand and 1 part (by weight) of calcined borax. This may amount to 15 or 20 by weight of the calcined gold slime, and gives a fluid non-corrosive slag at a reasonable heat. I would suggest that preliminary experiments in this direction be made with two of the pots in a "pour," as any trouble can easily be rectified (by adding borax or otherwise) in the next melt, and the conditions are strictly comparable.

It is clear from comparative experiments made with slime ranging in value from 10 to 35 of bullion that, as Mr. Dowling has pointed out (*Rand Metallurgical Practice*, Vol. I., p. 271, and *Chemical, Metallurgical and Mining Society's Journal*, Vol. VII., p. 212, January, 1907), the intimate mixture of flux and slime is not necessary, and this practice, which is conducive to loss of valuable dust, should by now be completely obsolete.

By the above methods the value of the gold slag produced after panning may be reduced below 10 oz. per ton, and as the quantity is smaller the total value of such by-products as pots, liners, and slag should not exceed ½ of the cyanide gold produced, and, in fact, in some cases is below ¼%, while the buttons of gold from each pot have averaged over 100 ozs. for some clean-ups. In this manner also a considerable saving in cost is shown.

The Chairman: The paper is open for discussion now and at the next meeting, and I hope therefore that members will come forward with contributions. There is one point I would like to touch on in these notes while proposing a very hearty vote of thanks to Mr. White, because this is an extremely useful paper, and suggests economies in many ways, which it [illegible] in these times to practice. That is the author's method of overcoming the cracks in the clay liners. There is often corrosion right through the liner, following evidently some line of weakness probably due to a crack, and whether this crack happens in the previous transport of the liner or whether it is due to uneven expansion at heating up does not matter. The manufacturers, when I was in London in 1910, told me that they thought this happened in transport, and that it could be detected by ringing. Whatever might be the cause of the crack, Mr. White's method seems to be a very useful and economical way of getting over it, as clay liners amount to an appreciable charge in smelting.

The vote of thanks was agreed to unanimously.

A METHOD OF DETERMINING GOLD IN BY-PRODUCTS CONTAINING PLATINUM AND IRIDIUM, ETC.

Mr. H. R. Jolly (*Member*): In the following contribution the part of the process which it is intended to describe most fully is the precipitation of the gold after it has been separated from the metals of the platinum group.

On receipt of the sample the most suitable flux can only be determined by trial fusions. The assays are fluxed, fused and the lead buttons cupelled. The silver and platinum in the resulting bullion beads are estimated in the usual manner. In the process to follow, the treatment of one button will be described. The bead containing the gold and iridium, etc., is heated carefully with dilute aqua regia 1–1 until all the gold is dissolved. The iridium, etc., is filtered off and the filtrate collected in a 600 cc. beaker. The filter paper is washed free from gold and the filtrate is neutralised with a 20 caustic soda solution. The beaker is then put on a hot plate, the contents warmed gently, and about 5 to 8 cc. of a warm 10 solution of potassium cyanide added, and the solution allowed to stand for about 10 minutes. 30 cc. sodium sulphite 20 solution are then added, and the beaker allowed to stand for about 10 to 15 minutes longer on the hot plate, after which it is removed and the contents diluted to about 300 cc. The process to follow is the same as the precipitation of a cyanide solution by Mr. White's method, which has been described before, but to make this method absolutely clear it is advisable to give it again. The reagents are

added to the solution in the following order and quantity:—8 to 10 drops of 10% potassium ferrocyanide, 30 cc. of 20% copper sulphate, 30 cc. 20% sulphuric acid, and 10 cc. sodium sulphite. It is absolutely necessary to stir the solution after the addition of each reagent. The precipitate is allowed to settle and collected on a filter paper; after draining, the filter paper and its contents are put into a No. 1 assay pot, fluxed and fused in the ordinary manner. The lead button obtained is cupelled, the bullion bead parted and the gold weighed.

Chemical gold was dissolved up in dilute aqua regia and treated as above. The following table shows the results:—

Original Weight. mgm.	Weight after Treatment. mgm.	Loss. mgm.	Percentage Extraction.
24·54	24·50	0·04	99·83
34·74	34·73	0·01	99·97
30·44	30·44	-	100·00
10·22	10·20	0·02	99·80
26·45	26·40	0·05	99·81

According to Watts' *Dictionary of Chemistry*, potassium auricyanide is formed by adding neutral gold chloride solution to a hot concentrated solution of potassium cyanide, and it is on this reaction that the foregoing method is based. It is, however, necessary to reduce the gold to the aurous state in order to get complete precipitation, and this is accomplished by adding sodium sulphite. (See Dr. Moir's discussion on Mr. Whitby's Method for the Assay of Cyanide Solutions, *Journal* for December, 1909; also the writer's discussion on the same paper, *Journal* for February, 1910.)

Mr. John Watson (*Member of Council*) moved a hearty vote of thanks to Mr. Jolly, which was adopted *nem. con.*

GOLD AND PRICES.

By Prof. R. A. Lehfeldt, D.Sc.

Currency consists of metallic and paper money and bank deposits; for the purpose of this paper we may express its amount as the product of the amount of gold (G), by a factor which will be called the "currency factor" (F). Then the total payments made can be put, on the one hand, as the amount of currency multiplied by the number of times it is used (in a year) the so-called velocity of circulation (V), and, on the other, as the total sales of goods (T), multiplied by a factor representing their average price (P). Or

$$GFV = PT$$

This is known as the equation of exchange. On differentiation it gives

$$\frac{dG}{G} + \frac{dF}{F} + \frac{dV}{V} = \frac{dP}{P} + \frac{dT}{T}$$

In order that the value of money should be invariable, or, in other words, that the average level of prices should not rise or fall, we must have $dP = 0$. Hence, in words, the percentage increases in G, F, and V taken together must be equal to the percentage increase in T.

(1) Statistics of trade on consideration lead to the conclusion that the total amount of trade T in the commercial world is increasing at from 3% to 3½% per annum.

(2) Statistics of the supply of gold are given by the United States Mint. They show that (at the end of 1911) the banks and public treasuries of the world possessed about 7,770 tons;[1] adding to this the best estimates for the circulation among the people (but excluding India and the East), the total comes to 12,410 tons. The output for 1911 was 695 tons; but from this must first be deducted what is used in the arts—say, 172 tons—as practically none of it is subsequently used for monetary purposes. There is also a demand for India; large quantities of gold are taken by India, and though not always melted down, practically disappear from circulation. In 1911 the net absorption was 132 tons (apart from the banks and treasuries). There remained 391 tons available to add to the existing stock, or a little over 3%.

(3) In estimating the currency factor, the numerator should be the total amount of currency in actual use among the people, *i.e.*, outside the banks; the denominator, the gold in the country. The quotient is then a measure of the economy with which gold is used to support currency in the country considered. As examples, take first Egypt; there the factor is hardly greater than one, for sovereigns are the ordinary means of payment, paper being almost unknown. Next Japan, a country at about the same level of commercial development. Japan keeps nearly all its gold in the vaults of the central bank, and uses banknotes and to some extent cheques; the currency factor being about 4½. As a consequence of this difference in custom, Egypt uses 25 gm. of gold per head, while

[1] 1 metric ton of gold = £136,500.

Japan manages with 4 gm. For the most important countries the currency factor is estimated at

France	$2\frac{1}{4}$
Germany	$2\frac{3}{4}$
United Kingdom	5
United States	6

An extension of the cheque system would imply an increase in F: but it is probable that in the English-speaking countries economy has already been carried as far as is practicable. There are also countries, such as India, Spain, and Brazil, whose F is high—too high—not on account of the excellence of their banking system, but because they are too poor to buy as much gold as they need. In such cases F may be expected to decrease. Accordingly, taken over the whole world, the increase in F may be expected to be small.

(4) It is most difficult to get figures as to velocity of circulation. Certain American estimates make it appear that in the United States the velocity for coin and notes is stationary, but for bank deposits (which are far more important in that country), V has grown from 36, in 1896, to 54 in 1913: that is, that the average American depositor draws cheques, in a year, to the extent of 54 times his average bank balance. This shows great activity of business, or, in other words, business done on a very narrow margin of bank balance—probably more so than elsewhere, except in the leading commercial cities. Probably the same increase in activity is to be found all over the world: in which case V is increasing at a considerable rate.

On the whole, the most probable values for the equation of increments would seem to be, the one side:

Increase in gold	3%	Total 5 .
Increase in currency factor ...	2	
Increase in velocity		

On the other:—

Increase in trade ...	3% to $3\frac{1}{2}\%$	Total 5%
Rise in price ...	$1\frac{1}{2}\%$ to 2%	

In order that prices should not rise, the gold production ought to be reduced, according to the above figures, to 1 or $1\frac{1}{2}$ per cent. I prefer the latter figure, so far as the immediate future is concerned, because the commercial development of the world shows no sign of being checked, whilst improvements in the utilisation of gold in currency cannot be carried to an unlimited extent.

FORECAST.

We may now consider the likelihood of future production. The most important matter is that there are plain indications that the present spurt in gold production is weakening. The following figures are very instructive:—

RECORDED PRODUCTION IN TONS.

	Transvaal.	Rest of World.
1909	227	... 456
1910	234	... 454
1911	257	... 438
1912	283	... 418
1913	274	... 414

It is clear that, but for the Witwatersrand, there would have been a marked falling off already: how long will the Witwatersrand be able to keep its output up to the present level? Here the points to consider are: (1) The published estimates of the contents of the present mines; (2) the existence of low grade ore in them; (3) the undeveloped areas beyond; (4) the tendency to rising costs, owing to rising world prices, to greater demands on the part of workmen, and to the prevalence of miners' phthisis; and, in the opposite sense, efforts at greater efficiency.

My own conclusion on this frankly controversial matter is that the output of the Witwatersrand is not likely to increase. Of course, one must recognise that forecasts may be upset at any moment by some great new discovery of gold ore, or by a new process for treating ores of low grade. But, putting aside the unforeseeable, it seems as if in ten years a reduced output would have to meet larger demands for gold in the arts; and that the balance, having to be reckoned as percentage of a larger stock than now, is not likely to be more than enough to sustain the level of prices.

If this view is correct, the present period of rising prices is drawing towards its close, and will be succeeded by a fall. This is what happened with regard to the spurt given by the Californian and Australian discoveries. It would follow that remedies for the rise in prices are not so urgently needed as has been thought. Revolutionary suggestions, such as that of Prof. Irving Fisher's "standardised dollar", would not be justified unless the disturbance due to price changes turns out to be greater than is estimated in the present paper.

SUGGESTION.

It is, however, suggested that there

might be international regulation of output. This is an operation that has been attempted, with regard to other minerals, by financial groups, in their own interest; could it not be done for gold, by a combination of Governments, in the interest of the public? The details of such a scheme must be left for discussion on another occasion.

P.S.—The above was written before the outbreak of the European War, which is likely to modify some of the statements in the paper.

Mr. R. A. Barry *Member of Council*: I am sure we are very much obliged to Prof. Lebedeff for the very interesting paper he has given us to-night. It is very necessary while gold remains the basis of our monetary system that we should have papers which must have the effect of making everyone of us think, and many things are the very valuable thoughts and extracts to-night. I am sure the whole Society is much indebted to Prof. Lebedeff for putting such a valuable paper before us, and to-night's discussion will be of great interest to us. I always look upon it as a great privilege that I have to thank the author of a paper, and to-night I feel that it is a great privilege that I have to thank Prof. Lebedeff for the very interesting paper he has given us. I am not quite sure whether, try as we will, that is quite within the realms of possibility, as we already have, to take small items, some difficulty on the mines, at any rate with regard to certain supplies; not unsurmountable difficulties, but it just shows you that there are difficulties, such as the supplies of boxes, candlesticks and dynamite-boxes, or the timber for making the boxes. Trade in these directions must suffer, and, I take it, in a good many other directions, too. I understand from letters from England that there is practically no gold at all in circulation there at present, the whole English currency has suddenly been switched over to £1 notes, and for the time of the war; then, again, other people will undoubtedly hoard their gold, and the quantity of gold required per unit of population must alter. The American methods with regard to small bank balances are exceedingly interesting, and I hope that bank managers here will take serious notice of that factor.

With regard to the gold output of the world, especially that of the Witwatersrand, I am sure that there are a good many people who have given the matter grave thought, and I believe that the consensus of opinion with regard to the Witwatersrand gold output is that we have to-day reached our highest figures, unless there are fresh discoveries in front of us. In the light of present knowledge it would appear that we certainly have reached our high water mark; we may remain at that mark for some time, but afterwards our output must gradually decrease. Prof. Lebedeff has said nothing with regard to what effect population would have upon the factors he has put in front of us. I take it that as the world's population increases it must affect, at any rate the velocity of circulation, and perhaps have other effects upon its factors.

Whilst there are many other phases of Prof. Lebedeff's paper which I would like to touch upon, I feel unprepared to do so to-night, and must hope others will take it up. We are really very much indebted to Prof. Lebedeff for his most valuable contribution.

Prof. J. S. Cellier *Member of Council*: I have very much pleasure in seconding Mr. Barry's vote of thanks to the author for his valuable paper. As regards the Rand, I am afraid we have seen the zenith. We have been hearing that Sudwest has been discovered at Daggafontein; but meanwhile a lot of Sudwest is being worked out, and in depth we generally find reefs of lower grade.

ROCK TEMPERATURES.

(Read at May Meeting, 1914.)

By E. J. Moynihan (*Associate*).

DISCUSSION.

Mr. J. Gibson (*Associate*).—Mr. Moynihan's paper again draws attention to the question of rock temperatures, one which, without doubt, is interesting and important as well from the scientific point of view as from the more practical one of its relation to the actual conditions we find or are able to bring about in the mines. In this relation we are all deeply interested, but as we are prone to deal with the rock temperatures and those of the mines together instead of separately? It is the relative effect of the one on the other that is important to us. In regard to the methods hitherto used for ascertaining the rock temperatures, it has been customary to assume that comparatively shallow holes give results sufficient for the purpose, and before we commit ourselves to comparisons with other records it would seem that, at least, we should be sure the methods used are uniform. The author does not appear satisfied that the methods used are sufficiently accurate. Basing his argument on variations found in shallow holes on the earth's surface, he concludes that the figures obtained by holes in the strata at depth are erroneous. The question which seems pertinent is this: "Are these methods sufficiently reliable to give indications of rock temperatures which will be near enough to enable us to anticipate the conditions to be expected and combat with at any particular depth?" Viewing it in this way it is probable that these results are not so misleading as the author believes. Perhaps they do not reflect the actual rock temperatures of the rock masses *in situ*, but even so, it is not material to us.

It would be very useful however, if in making his reply Mr. Moynihan would suggest an alternative method whereby more reliable results might be obtained; also, if possible, some information were supplied as to the conductivity of rocks.

He states that the cooling effect of air currents in workings probably extends for 100 ft. or more from the surfaces exposed. Whether this be so is not easy to prove, but it is certainly reasonable to suppose that a steady diminution from within the rock masses to the surfaces exposed by mining operations does take place. If this be so, then there is a certain rate of transference of heat from the rock to the air currents. This rate must be some function of the conductivity of the rock. It will also depend upon the capacity of the medium (*i.e.*, the air in contact with the rocks) to absorb the heat radiated. So long, therefore, as there is somewhat more air in circulation than is sufficient to effect the removal of this heat radiated from the rock walls, mainly high working temperatures will not be experienced. We may therefore be fairly safe in concluding that it is not so much the actual temperature of the rock *in situ* with which we are concerned, as the temperatures near enough to the workings to affect the air currents. So far, the majority of observations made have been undertaken with this end in view, rather than to determine the true rock temperatures.

In discussing what has been termed the "temperature gradient," as deduced from observations recorded by others, the author has selected individual records to try and prove the futility of any attempt to establish a gradient. Naturally enough, in doing so he finds between the records from certain points a negative gradient, but surely this is no argument that a gradient cannot be found, whether its equation be that of a straight line or not. Might I suggest, rather, that such records be considered, not individually, but in groups, with proper weights applied before their utility is decried?

Contrary to the assumptions made by the author in his text, seasonal variations at the surface do not appear to affect the mine air temperatures very much, possibly not more than a few degrees. From the point of view of efficiency of labour, however, one very important aspect of this question is the percentage of humidity when temperatures up to 85° F. are experienced, and judging from the data available it is probable the estimate of Mr. Moynihan is nearly correct, that at about 5,000 ft. this limit will be reached on the Rand. Even before these high temperatures are attained, close attention to ventilation will be very necessary. It is also clear that consideration of the volume of air in circulation is most important, and that its velocity should not be low. In Prof. Cadman's paper this question is

GELDENHUIS DEEP, LTD.—Observations of Mine Air Temperatures.

	Date.	Number of Level.	Depth.	Temperatures. Dry Bulb	Temperatures. Wet Bulb	Remarks.
Geldenhuis Deep Section.	August 1913	2	495	66·5	66·4	No. 1 Shaft—Upcast with fan at surface
		8	1,012	68·4	68·3	
		12	1,521	70·1	69·9	
		15	2,059	71·9	71·1	
						No. 2 Shaft—Downcast
	June 1913	6	840	58·0	56·0	This is at the bottom of vertical
		8	1,035	62·6	61·3	
		12	1,576	66·0	63·3	
		15	2,088	67·5	66·0	
		17	2,534	72·0	70·7	The rock temp. on 17th Level is 75°.
	August 1914	8	1,035	57·6	57·0	No. 2 Shaft, taken on a very cold day last month.
		12	1,576	58·2	56·0	
		15	2,088	61·7	59·5	
		17	2,534	67·5	64·0	
	August 1913	1	782	67·5	67·2	No. 3 Shaft—Upcast, no fan.
		4	1,033	68·1	67·8	
		8	1,539	69·2	68·7	Air door closed.
		15	2,149	73·0	71·7	No connections yet made on this level.
		17	2,552	75·0	74·3	
Jumpers Deep Section.	November 1913	1	782	68·5	68·0	No. 3 Shaft—Upcast, no fan.
		4	1,033	68·6	68·1	
		8	1,539	69·0	68·3	
		15	2,149	72·0	71·1	Air door open.
		17	2,552	74·1	73·7	No connection yet made.
	August 1913	6	1,225	55·8	55·1	No. 4 Shaft—Downcast.
		7	1,350	56·8	56·0	
		8	1,506	55·9	55·0	
		10	1,790	58·4	57·7	
		11	1,942	59·3	58·5	
		15	2,113			
	November 1913	6	1,225	61·5	61·0	No. 4 Shaft—Downcast.
		7	1,350	61·9	61·3	
		8	1,506	62·2	61·8	
		10	1,790	63·3	62·8	
		11	1,942	64·3	63·6	
		15	2,113	64·8	64·0	

clearly discussed, and his conclusions will well repay studying.

While under the necessity of using so much water in the form of sprays, etc., for dust allaying purposes, it would appear impossible to endeavour to improve conditions where high temperatures are found, by lowering the percentage of humidity and supplying a much drier air current. This phase of the ventilation question will, before long, have to be considered and dealt with in the deeper mines, and so far refrigeration appears to be the only practical alternative suggested.

Above are a few short tables showing the results of observations of air temperatures actually obtained at the Geldenhuis Deep. These were taken at the shaft stations, and the adjacent workings do not show any great fluctuations where observa-

tions have been made. One thing evident from these tables is the high percentage of humidity throughout, also how the temperatures in No. 1 shaft are affected by the falling water in the vertical portion. Observations taken at No. 1 shaft on hot days in the Autumn showed that while the surface temperature was varying from 74° to 80° F., that at the 6th Level (1,225 ft. from the collar) was only 58° to 60° F., showing that the heat is rapidly extracted by evaporation of the water falling in the shaft. On cold days in winter, however, the cool air entering a shaft will not be warmed up to anything like a corresponding degree. A reference to the second table given for No. 2 shaft will make this clearer.

The Government Mining Engineer for the Union, in his annual report for 1913, refers to the reduction of rock temperatures at the St. John del Rey mine in Brazil,* where a reduction from 107½° to 98° F. was effected. A further reduction to 91° F. is anticipated, and this at a depth of 5,300 ft. Concluding, he states:—"These results indicate that increasing temperature should not be a serious bar to successful mining on the Rand at depths greatly in excess of that already reached. At a depth of 7,000 ft. on the Rand the rock temperature should be 97° F., and if the St. John del Rey experience counts for anything, this should be reducible to an air temperature of 87° F. and possibly less, owing to the difference in surface temperature and humidity being greatly in favour of the Rand."

THE WITWATERSRAND EARTH TREMORS.

(Read at April Meeting, 1914.)

By H. E. Wood, M.Sc., F.R.Met.S. (of the Union Observatory).

REPLY TO DISCUSSION.

Mr. H. E. Wood: The material on which I based my paper was entirely obtained from the records of a seismograph installed at the Observatory, and I was lead to the conclusion that the origin of these local tremors could not be entirely dissociated from the mining operations on the Witwatersrand. It is obvious, however, that I dealt only with one side of a large question, and that my paper is really not complete until I can answer such questions as those raised by Mr. Crosse and Mr. Hilditch Smith as to the synchronization of earth tremors and falls of rock in mines. I would emphasise the fact that it is not to be expected that every earth tremor will be accompanied by rock falls, and also that not every fall of rock must be the result of an earth movement. There may be falls of rock in a mine, even although the superincumbent mass may be perfectly stable, due to the disturbance of the balance of hydrostatic pressures.

Owing to the kindness of the Government Mining Engineer I have been placed in possession of a large mass of material dealing with the practical side of the question, from which I hope to supplement the paper I have read before you.

The Chairman: I would like to assure Mr. Wood that his supplementary remarks will be very welcome to the Society.

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

(Read at May Meeting, 1914.)

By G. H. Stanley, A.R.S.M., M.Inst.M.M. *(President).*

DISCUSSION.

Mr. G. A. Robertson *(Member):* I agree with the author that there is room for improvement in our present-day system of hydraulic classification on the Witwatersrand. That improvement is limited to (1) a saving in capital expenditure by the employment of shallow classifiers, shallow pits for tube-mills, and a low height of elevation of pulp; (2) lower working costs due to the handling of smaller quantities of sand in the tube-mill circuit; (3) to increased gold extraction. When a large number of mine reports show a gold extraction varying from 95 to 97 the improvement that can be made in classification in such cases is chiefly by reducing the tonnage of sand being fed to the tube-mills. That does not necessarily mean an increased gold extraction, but may mean a longer life for tube-mill liners and pump liners, and a resultant lowering of working costs consequent on the handling of a reduced tonnage of sand in the tube-mill circuit. In some cases this may mean a reduction from 300 to 400 tons per day down to 150 to 200 tons per day."

* See this issue of *Journal*, p. 65.

* See *Rand Metallurgical Practice*, Vol. II., p. 965.

The author's experiments are interesting, but he would have had better satisfaction had he applied to some of the mines near Johannesburg and been allowed to by-pass a stream of pulp from five stamps for classification purposes instead of contending with laboratory difficulties in the handling of streams of pulp.

I may here mention that experiments were carried out by me two years ago on the May Consolidated G.M. Co., through the kind permission of Mr. K. H. Sartorius, the Manager, with a new type of grinding mechine on a small scale. The pulp from five stamps was by-passed to the grinding mill so that the small experimental mill and classifier were installed between the stamp battery and the tailings pump. The pulp from five stamps passing a 600 mesh screen would be about 25 tons per day, so it was necessary to reduce this tonnage down to a stream containing about 1,000 lb. to 1,500 lb. of sand per 24 hours, which quantity was attended to and to the capacity of the machine. Sizing machine was, of course, quite unsatisfactory, but nevertheless it served its purpose in proving the principle of the mill, which has since been continued on a larger scale. The difficulty in classification was in getting the desired quantity of pulp, that is, in getting the quality by reducing it to the 25 tons. The following sketch shows the machine used, which worked very satisfactorily. The launder at the left end of the motor plate sloped towards the centre, where the pulp discharged through a 3" pipe, 3 feet long, with a valve to shut off to it to hold it during tests, connected up to the 3" pipe to fit to the end of the launder. Naturally, sand passed up the launder near this height. The 3" pipe could be shut down from the top of the 27" main pipe, and it was found at the bottom of the launder. The pulp from the five stamps flowing down each slope in the launder met over the lath in the front of the launder, where the coarse sand discharged through the lath. The fine sand discharged down the 27" pipe. The same principle carried out on a large scale would reveal some startling results, because the coarse sand could distinctly be felt falling down at slow velocity on the packed sand or soft lined launder, and the two currents meeting over the horizontal lath allowed a rapid separation of coarse and fine sand to take place. Hydraulic water was added to the pulp issuing from the hole in the front of the launder, and this flowed to a square box having a hole in one side for the classified pulp to flow to the grinding mill, and another hole on the opposite side of the box for the discharge of the overflow. Bearing in mind that the coarse sand flowing in the bottom of the launder takes the line of least resistance when falling, an iron plate was then placed inside the box across the falling stream. The coarse sand which fell on the side of the iron plate on the side of the falling stream, then flowed back into the grinding mill, while the upper portion of the stream passed over the plate and flowed to the tailing pump. By this method it was possible to get the desired quantity of sand for the mill from a 600 mesh aperture 0·039 in., giving in some cases only 2·9 of −90 grade. This classification, however, is only the sorting out of a small quantity of the coarse sand, and could not be considered representative of full classification results, but the points to be observed here are the low fall required for classification, which in the two stages could be carried out within a depth of 12 in. Naturally, for tube milling purposes further dewatering would have to take place, requiring further fall. On reviewing, then, the whole question of hydraulic classification, is it necessary to have classifiers varying from 1 ft. to 9 ft. deep in order to get a clean tube mill feed containing about 40 of moisture? Wide or narrow classifier overflows have no intrinsic effect on the results to be obtained, as illustrated in the launder classifier already referred to.

The experimental grinding mill, already referred to, developed into what is now known as the Krupp-Simon tube-mill, which was exhibited at the last mining exhibition. For its size it has shown some excellent results. The classification system had, however, to be altered, as it is now necessary in the second stage of classification to have two overflows giving equal gradings and no structure feeding both ends of the mill. Experiments along these lines are in progress, and I hope eventually to read a paper before the Society dealing with the subject.

All classifiers on the Rand properly adjusted to local conditions are doing excellent work, but local conditions do not allow the fall generally allotted for classification purposes, and result in encroaching on capital account, a good deal often to be learned by at least trying something new.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

CARBON MONOXIDE IN MINE AIR: METHODS OF ESTIMATION AND A SUGGESTED METHOD OF REMOVING IT FROM AIR.—"Most methods of estimation have the objection that they cannot be applied on the spot in mines, where the estimation would be of great value; the only test free from this objection being the use of small warm blooded animals (birds or mice). A mouse does not detect less than 0·1 of CO in the air through its symptoms, yet its blood may be used to detect as little as 0·01. It is unfortunate, however, that neither the blood nor the comparison carmine solution will keep even for a day, and that in artificial light the difference in tint on which the test depends is almost invisible. The writer thinks that mining men will agree that it can never be of any real use in mines from the practical standpoint. The author has however invented two processes—depending on the I_2O_5 method, whereby a determination of CO can be made on the spot in a mine.

Apparatus.—The apparatus used in this test consists of an instrument like a garden-syringe, which is used to collect and measure the sample. The rod of the piston is so graduated as to show the percentage of carbon monoxide in the air tested. The metal cylinder contains a definite quantity of air, and this must be dried, and any unsaturated hydrocarbons (preferably carbon dioxide also) removed. This is effected by passing it over bromine, water, and then solid caustic soda or potash contained in glass tubes; the air is then passed over the iodine pentoxide in a coil of glass or silica contained in a special safety-lamp above the outer gauze and inside the bonnet. It was found that the space above the outer gauze and inside the bonnet of a safety-lamp was just what was required, the temperature there being about 150 Cent. with the lamp full on and not "smoking." Here the carbon monoxide, if present, is converted into carbon dioxide and iodine is liberated. The air containing the iodine (if any carbon monoxide is present) is passed through a measured quantity of a standard solution of arsenious acid containing sodium bicarbonate and starch; and, as soon as the arsenious acid is used up, an intense blue colour due to the formation of iodide of starch is formed. The position of the piston in the syringe is noted, and the percentage of the carbon monoxide in the air read off therefrom.

The volume of the cylinder and the amount of standard arsenious-acid preparation are so arranged that if less than 0·01 of carbon monoxide is present, no colour is formed; but if a large quantity of the gas is present, the blue colour is shown very rapidly —0·4, for instance, is shown in a few seconds, 0·1 in about a minute, and 0·01 in about ten minutes. If a dangerous quantity of carbon monoxide is present, warning is given within a minute by this test; for less quantities a longer time is taken to obtain the result, which is just what is wanted in a test of this kind."

Another method, which is a modification of that just described, but better in some ways, is to make use of the fact that carbon dioxide is evolved by the reaction of carbon monoxide with iodine pentoxide—

$$I_2O_5 + 5CO = 5CO_2 + I_2.$$

In this process the iodine is kept back by passing the gas after leaving the iodine-pentoxide tube over filings or turnings of metallic copper, with which it combines. The gas containing the carbon dioxide is then passed through a spiral tube containing a definite quantity of standard solution of caustic potash (or soda), followed by a small vessel containing a trace of baryta water coloured by phenolphthalein. This colour is very intense, and so soon as the carbon dioxide (from the carbon monoxide, if present in the air) has used up the caustic potash (or soda) in the spiral, it comes forward to the vessel containing the baryta and phenolphthalein. When this happens the violet colour slowly disappears,* and the percentage of the carbon monoxide is read off on the rod of the piston as in the starch iodine reaction already described.

The yellow mercuric oxide process has only just been published in the scientific journals, and the writer has not yet had time to try it. It appears, however, likely to be better than the iodine pentoxide method, because it is stated to react with carbon monoxide at 100° C. (the boiling point of water).

Part II.—Removal of Carbon Monoxide.

A mixture of hydrogen with oxygen or air has an ignition point of about 1,200° F. (650° C.); but if the mixture be passed over palladium warmed to 140° F. (60° C.), combination takes place rapidly. It has, up to now, been generally supposed that carbon monoxide acts more or less as a poison to most catalysts, although the writer has not been able to find any experimental evidence in support of that supposition.

In the course of the investigation a method of estimating and recording oxygen in hydrogen was investigated. The best way of doing this appeared to be to use palladium as a catalyst to burn the oxygen to water, and to take the heat formed as a measure of the oxygen present. Palladium black was found to be the best catalyst, as it worked at the ordinary temperature and converted the oxygen into water. During these experiments the action of carbon monoxide on the palladium-black was tried, and, much to the writer's surprise, the supposed poison for catalysts was found to be so active in the presence of air that the palladium black was heated from the ordinary temperature to redness, and then the carbon monoxide ignited. It was found on further investigation that platinum black would act on carbon monoxide in air at ordinary temperatures, and convert the poisonous gas into carbon dioxide.

There was now no doubt as to the best method of estimating the carbon monoxide for the future in mining practice; but the question was: could the new discovery be made of real importance for the basis of a respirator, and perhaps be the means of saving many lives which are now lost through slow poisoning by carbon monoxide? There is no doubt that a respirator *can* be made which will convert the poisonous carbon monoxide into carbon dioxide at the ordinary temperature; but can it be done at a cost which is not prohibitive?

These metals (platinum and palladium) are about twice the price of gold; the supply is decreasing, and the demand increasing, despite the high prices. The demand is, to a moderate extent, due to the misuse of the metal in the making of jewellery.

* The piston is provided with a rack-and-pinion movement on carrying out a test. The piston is worked by a worm-and-wheel gear, so that the piston is moved along at a slow steady rate when the handle is turned sixty times per minute. A large nut screw connects the two gears, and when this screw is loose (as it is when taking the sample) the piston can be pushed in and out rapidly.

* The baryta must be quite pure and dissolved in distilled water, otherwise the colour does not disappear.

The writer has not made very much headway with the use of these active metals for the removal of carbon monoxide or for the estimation of it. Moreover, on account of these high prices a very extensive search was made to see whether some cheaper material could not be found. Up to the present nothing has been found that can take the place of these rare metals. It is quite possible, however, that some cheaper material may exist which will act in the same way, and it may be found sooner by publishing this work at a premature stage.

After an explosion of firedamp and coal-dust with insufficient air, carbon monoxide is always present, and is accompanied by hydrogen ($2CH_4 + O_4 = 4H_2 + 2CO$), for the oxygen always attaches itself first to the carbon part of a hydrocarbon during combustion. The hydrogen is also burnt by these rare metals at ordinary temperatures which would help in removing the carbon monoxide by giving heat. It has been found that the rate at which they act on carbon monoxide in air rapidly increases with only a moderate rise in temperature. It is possible that other gases which may be present after an explosion may poison the catalysts. All these points have to be carefully investigated before it can be decided that it is feasible to make a respirator weighing a pound or so which will be quite certain in its action and not be too costly.

During the experiments the writer has on several occasions after making the platinum and palladium-blacks found them to be inactive at the ordinary temperature, and the cause had to be traced in order that it could be avoided in future." DR. J. HABER. *Trans. Inst. Min. Eng.*, 1914, p. 533. (J. M.)

DETECTION OF TRACES OF ALUMINIUM. "A method is given by G. H. Pettit for the detection of traces of aluminium. The hydroxides of iron, chromium and aluminium precipitated in the usual way are washed thoroughly, transferred to a beaker by means of water, heated to boiling with solid barium hydroxide and filtered. The filtrate is acidified with hydrochloric acid, boiled and made alkaline with ammonia which has been kept over barium chloride. Aluminium when present is immediately precipitated. Where traces only of aluminium are present, the filtrate obtained after boiling the precipitate with barium hydroxide is made slightly acid with sulphuric acid, concentrated, filtered and ammonia added drop by drop (not more than 4 to 5 drops). It is then heated just to boiling and allowed to stand, when the aluminium hydroxide settles down if present. This method will detect 5 mg. of aluminium in the presence of 5 gm. of iron." G. H. PETIT, *Engineering and Mining Journal (Amer. Pharm. Chem.*, 1914, p. 66). April 25, 1914, p. 859. (G. H. S.)

PREPARATION AND PROPERTIES OF SODIUM ZINC CYANIDE. "This was prepared as follows: 'Zinc cyanide was dissolved in NaCN solution, free from potassium and carbonates, using an excess of zinc cyanide. The zinc cyanide was first prepared by dissolving zinc chloride in boiled water, acidified with a little hydrochloric acid. The free acid insured the presence of hydrocyanic acid in the solution after precipitation, lessening the tendency toward hydrolysis in washing. The precipitant was a solution of sodium cyanide, which had been treated with a slight excess of lime and filtered to remove carbonates. The solution was stirred vigorously while precipitating, and filtered with the aid of suction. The precipitate was washed, starting with warm and finishing with cold, recently boiled distilled water. The zinc cyanide was then digested with a solution of sodium cyanide, also treated with lime to remove carbonates. Then the solution was filtered, evaporated to a syrupy condition, and placed in a vacuum desiccator over concentrated sulphuric acid. After a few days a white layer was seen on the surface of the solution, which may possibly have been the compound $NaZnCy_3$ as described by previous investigators. It was not further examined, however, as upon attempting to remove this film for investigation it was found that below the surface there was a good crop of diamond-shaped crystals that were invisible when in the solution on account of having the same index of refraction as the strong solution. These crystals were removed, drained, dried on filter paper, and analysed with the following results:—

	H_2O	CN	Zn	Na (by diff.)
Per cent. ...	20·20	39·44	24·60	15·76
Molecular ratio	2·95	4·03	1·00	1·82

These results indicate a compound $Na_2ZnCy_4.3H_2O$, agreeing in composition with the solution.

A similar series of experiments on a larger scale gave products showing the following analyses:—

Na ...	2·2	2·06	1·98
Zn ...	1·0	1·00	1·00
CN ...	3·85	3·88	4·02
H_2O ...	2·96	2·91	3·07

The solubility was determined by determining the specific gravity of a saturated solution at 15° C = 1·37, weighing a portion of this solution and determining the Zn content. The results were: 1 part anhydrous salt is soluble in 0·83 part water, and 1 part $Na_2ZnCy_4.3H_2O$ in 0·47 part water.

The crystals were measured and the measurements are given in the original paper." E. HERZ. *Journal Amer. Chemical Society*, xxxvi., No. 5, May, 1914, p. 942, 946. (J. A. W.)

THE DETERMINATION OF SMALL QUANTITIES OF METHANE IN MINE GAS. —"Very small quantities of CH_4 are determined directly by passing the gas free of CO_2 over hot CuO, and determining the CO_2 which is formed by titration. It is difficult to get complete absorption of the CO_2. When CH_4 is present in quantities of less than 0·05 vol. %, this may result in serious errors. A better result can be obtained by passing the CO_2 through a glass tube with a capillary end directly into the $Ca(OH)_2$ solution." E. MUCHANN, *Oesterr Chem. Ztg.*, 17, 69. *Chemical Abstracts*, June 20, 1914, p. 2135. (J. G.)

COLORIMETRIC ESTIMATION OF SMALL AMOUNTS OF HYDROGEN SULPHIDE. —"By the action of $FeCl_3$ and dimethyl *p*-phenylenediamine sulphate upon aqueous solution of H_2S containing free HCl, methylene blue is produced." Under given conditions the intensity of the colour developed is proportional to the amount of H_2S present, and the colour is stable for several weeks. Variations in the amounts or order of addition of the reagents alter the intensity of the colour. The standards to be used in the colorimeter must be prepared from known amounts of H_2S, determined iodometrically, since the colours produced are not exactly the same as those formed by the dilution of solutions of methylene blue. Conditions recommended for determining 1,000 to 3000 μg.* of H_2S per litre are, 500 cc. solution, 10 cc. concentrated HCl, 25 mgm. 'diamine sulphate,' and 2·5 cc. 0·1 *M* $FeCl_3$ (in 1 : 1 HCl). For less than 1,000 μg. H_2S per litre 10 cc. HCl, 10 mgm. 'diamine

* 1,000,000 μg. = 1 gm.

sulphate and 1 cc. 0·1 *M* FeCl$_3$ are added to 500 cc. of the solution. The solutions are allowed to stand several hours before being compared with the standards. Results with as little as 10μg. H_2S per litre were very satisfactory." W. Mecklenburg and F. Rosenkranzer, *Z. anorg. Chem.*, *86*, 143-53. —*Chemical Abstracts*, June 10, 1914, p. 1938. (J. G.)

Determination of Cyanide in Electroplating Solutions.—"A measured quantity of the cyanide solution, preferably containing from 0·2 gm. to 0·5 gm. of cyanide, is diluted to 100 cc., treated with 1 cc. of ammonia solution, and 1 cc. of a solution of 8·9 gm. of dimethylglyoxime in 1 litre of 95 alcohol, and titrated with a standardised nickel ammonium sulphate solution (containing 15·3 gm., with 5 cc. of concentrated sulphuric acid, diluted to 1 litre) until a permanent red precipitate is formed. In the reaction one molecule of the nickel sulphate combines with four molecules of potassium cyanide."—G. E. F. Lundell, *Met. and Chem. Eng.*, 1914, *12*, 340. —*Journal of the Society of Chemical Industry*, June 15, 1914, p. 596. (J. G.)

Volumetric Determination of Free Sulphurous Acid.—"The determination of free sulphurous acid in the presence of other mineral acids could not be satisfactorily effected by titration with *N*/10 caustic alkali, using methyl orange (to bisulphite), and phenolphthalein (to neutral salt), as indicators, as the latter developed colour but slowly and appeared to be 'bleached.' When excess of hydrogen peroxide was added after the first titration a sharp colour change was obtained in the second stage with phenolphthalein." —(See *J.S.C.I.*, 1914, 442).—E. Kedesdy, *Chem. Zeit.*, 1914, *38*, 601-2. —*Journal of the Society of Chemical Industry*, June 15, 1914, p. 615. (J. G.)

A New Indicator is afforded by the new potato, Solanum Commersoni, which is being grown in France, which contains a violet colouring matter, according to the *Chemical Engineer*. If the potatoes are boiled in water, the ferments are destroyed and the resultant blue solution keeps well. As an indicator it is very sensitive, turning green with alkalies and bright red with acids."—*Engineering and Mining Journal (Chemical Engineer)*, June 6, 1914, p. 1153. (G. H. S.)

Detection of Cyanides in Presence of Ferro- and Ferricyanides and Thiocyanates.—"The test is based upon the solubility of copper sulphide in alkali cyanides. A suspension of copper sulphide is made by passing a few bubbles of hydrogen sulphide into a weak ammoniacal solution of copper sulphate, and if the colour is discharged on the addition of an ammoniacal solution of the substance, the presence of a cyanide is indicated. By using a sufficiently dilute copper solution, the test has a delicacy of 1 part of hydrocyanic acid in 10,000, and it is not seriously interfered with by the presence of ferro- or ferricyanides or thiocyanates. In presence of 2 gm. of these substances, 0·5, 1, and 0·5 mgm. respectively of hydrocyanic acid can be detected. By using a standard copper solution the test can be utilised for a rough quantitative determination if only cyanides are present." —O. L. Barnebey, *Jour. Amer. Chem. Soc.*, 1914, *36*, 1092-1093.—*Journal of the Society of Chemical Industry*, xxxiii., July 15, 1914, p. 690. (J. A. W.)

METALLURGY.

Leaching at Chuquicamata, Chile.—"The Chuquicamata deposit is a brochantite mineral contained in cleavages of granodiorite, mixed in part with chalcanthite and atacamite, and with a deposit of salt. Some *caliche* containing nitrates is present in the upper part of the ore body.

The so-called Llampera ore body is about 8,000 ft. long by 500 ft. wide. Numerous tunnels have been run through the ore body, and some open cuts made. Raises have been made from the tunnels almost to the surface, only a shell 8 to 12 in. thick being left in places. Seven shafts have also been sunk in the deposits, the deepest, 110 m. deep.

It has been explored by churn drilling and an ore body of over 200,000,000 tons developed, but most of the drill holes were stopped in ore. Some drill holes have been sunk at a considerable distance west of the Llampera, which encountered chalcocite and chalcopyrite after penetrating 300 or 400 ft. of capping. The indications are that Chuquicamata is the largest known copper deposit.

It was formerly assumed that the deposit was atacamite, which could not be profitably treated by established methods because of the silicious nature of the ore and the absence of sulphides and water, and on account of the volatilization of the cuprous chloride. However, it was eventually shown the ore was brochantite and salt, and not atacamite (oxychloride). Brochantite is insoluble in water, but readily soluble in dilute sulphuric acid.

About 600 tons of the ore, an average sample, so far as could be taken, were shipped to New York for experiment, first on a laboratory scale, then in a 2-ton per day plant. The problems to be solved were: Solubility of the copper in dilute sulphuric acid; the chemicals required; the impurities dissolved; the electrolytic precipitation.

It was quickly found that the copper was readily soluble in cold dilute sulphuric acid, and that only coarse crushing was required. The acid from the ore (oxysulphate) more than made up the losses of the acid used up in dissolving other materials than copper (mainly CaO), and the waste of solution in the discarded residue.

The 2-ton experimental plant was run as a self-contained unit, leaching the ore, washing, purifying and electrolyzing the solutions, and leaching the following lots of ore with the previously electrolyzed solution.

It was shown in running 131 charges, that no harmful impurities except chlorine had a tendency to accumulate in solution. The average extraction was 90·98 %.

Following the experiments in the 2 ton plant, a leaching tank was built of concrete, 4 6 15 ft. deep, holding 15 tons of ore, allowing the use of the full height of ore column since adopted for Chuquicamata. This gave equally good results, showing this height of column was feasible.

The gain in acid in the experimental work amounts to about 9 lb. H_2SO_4 per ton of ore treated. Since this was made treating ore which averages higher in chlorine than even the average of the upper layers of the ore, and the greatest loss of acid is in forming Na_2SO_4, in final practice the acid gain should exceed this.

The main impurity in the solutions is, as said, chlorine.

Experiments showed the feasibility of eliminating the chlorine from solution as cuprous chloride, by treatment with shot copper. Cuprous chloride formed on metallic copper sticks tight to copper shot

and it is necessary to present constantly clean surfaces to maintain effective precipitation. After trying several methods it was decided that the best apparatus for this purpose would be revolving drums similar to tube mills, containing the shot copper. Cuprous oxide also works well as a precipitant, but at a lower efficiency, even after allowing for the theoretical difference.

It is the intention to filter press the cuprous chloride precipitate, mix the cake with limestone and coke, and smelt. A liquid sodium chloride slag is obtained practically free from copper, and there is no loss of cuprous chloride by volatilization.

The solution, freed from chlorine, is electrolyzed, using copper starting sheet cathodes and fused magnetite anodes. The cathodes are practically free from chlorine, and are free from arsenic and antimony, as these are not in the electrolyte. The solution will enter the head tank containing about 5 Cu, and 2½ to 3 free acid, and will leave the last tank of the cascade at about 1·5 Cu and 8 to 9 free acid. Within these limits it is always possible to produce a good, hard cathode, equal to or better than the usual standard.

What to Treat. 10,000 *Tons of Ore per Day.* The first unit of the Chuquicamata plant is designed to treat 10,000 tons of ore per day. The electrolytic refinery will have a capacity of 325,000 lb. per day. The ore will be mined by steam shovels. The ore will be taken to the mill, about 2 miles away, in standard gauge 60 ton cars. At the plant it will first pass through gyratory crushers, thence through Symons disk crushers, and finally through Garfield rolls, until a 10 mesh product is obtained. After sampling, the ore will go by conveyor belt to the leaching tanks, of which there will be six, end to end, 100 ft. wide, 100 ft. long and 15 ft. high.

The heat will be discharged into the leaching tanks by an electric traveling bridge. The leached and washed residue will be removed by a 15 ton skip bucket traveling on an unloading bridge, and after sampling, will be delivered to a conveyor belt for disposal to the tailings dump.

The tanks are being built of heavily reinforced concrete, and will be lined with waste asphalt 3 in. thick. This lining is made of one part bituminal asphalt, to which is added three parts screened quartz or granite. The lining has been in use in the experimental plant over a year, both in the leaching and electrolytic tanks, and not a single leak has developed. It does not soften at 50° C., but probably softens between 50° and 70° C. In one case a 250 lb. cathode was dropped 5 ft., so that one of its sharp corners struck the tank bottom. In this case, a piece of lining only about ½ in. thick and 2 in. in diameter was broken out, so the lining also seems to stand up to physical tests. It is unaffected either by strong or weak acid solutions.

From the bottom of the leaching tanks the solution will be removed through eight 6 in. openings evenly spaced over the tanks. A filter bottom, consisting of 2 in. planks with a layer of excelsior between, will be employed. This will be raised 1 in. above the tank bottom.

For the main solution circuits, 9 in. and 6 in. lead lined iron pipe will be used. Open head horizontal centrifugal pumps, made of type metal, will be used for pumping the solutions. The pumps will have a capacity of 550 gal. per min., elevating to a height of 60 ft.

The leaching cycle will be approximately six days, of which one day will be occupied in filling the tank, two days in leaching, two days in washing and draining and one day in discharging. The solutions will be conducted to the leaching tanks by gravity from nine solution storage tanks: Two tanks, 12 130 150 ft.; seven tanks, 12 70 150 ft.

From the strong solution tank the solution will flow by gravity through the dechlorinating plant, consisting of twenty-one 30 × 4 ft. inside dia. revolving steel drums lined with duriron. These drums will be half full of shot copper. From these drums the solution, containing the cuprous chloride in suspension, will flow to seven Dorr thickeners made of concrete and lined with mastic asphalt. The clear solution overflowing will travel by gravity to the electrolytic refinery, while the thickened cuprous chloride, together with approximately 1% of the original solution, will go to filter presses.

The electrolytic refinery will consist of 510 electrolytic tanks 19 ft. long, 3 ft. 6 in. wide, 4 ft. 10 in. deep. They will be made of concrete, lined with mastic asphalt, and will be set through the floor, having inspection aisles underneath and a concrete working platform between the rows of tanks. The tank house will be of concrete and steel construction.

Of the 510 tanks, 30 will be used for making cathode starting sheets, and the balance, 480 tanks, will be used to electrolyze deposition of the copper from solutions. The tanks will be arranged in five electrical circuits, with 96 tanks to a circuit, and will be divided into 30 solution circuits, the solution in each circuit flowing through 16 tanks placed in cascade. The anodes, made of magnetite, will be 5 in. wide, 2 in. thick and 4 ft. long, five to a cross rod. The cathodes will be 3 ft. wide by 4 ft. deep.

The spent electrolyte, containing about 1·5% copper, will be pumped back to the storage tanks to be used for leaching subsequent batches of ore. The cuprous chloride from the dechlorination plant will be smelted and cast into shot copper for use in the dechlorinating drums as outlined above.

The power plant will consist of a primary power station at the coast, using oil fired Babcock and Wilcox boilers driving four steam turbines connected to four 10,000 kw. alternating current generators. The current will be transmitted 85 miles at 100,000 volts to a substation located at the mill site. The substation for the electrolytic refinery will consist of seven motor-generator sets, each of 2500 kw capacity." E. A. C. SMITH, *Mining and Water Journal*, April 25, 1914, p. 865 (U. H. S.)

AMALGAMATING IN CYANIDE SOLUTION.—"In case coarse gold is encountered in cyaniding ore, the question of amalgamation is not so important as formerly. When separate treatment of sand and slime was the general practice the question was quite important, but the problem was never satisfactorily solved. With the prevalence of all-slimed ore the case is quite different. The ore travels in a closed circuit through the fine grinder and classifier until reduced to slime, and any coarse gold which may happen to be in the pulp is thus 'worn out' and gives no trouble in the extraction department. Therefore, in all slimed mills it is considered good practice to simplify the process by cutting out amalgamation, even when the ore contains coarse gold.

However, there are cases in which it is best to treat a part of the ore as sand, and then if it contains coarse gold the question of amalgamation comes up. To crush in water, amalgamate, dewater and then treat with solution occasions a loss of

cyanide besides complicating the process; while amalgamating in cyanide solution has its drawbacks. In the latter case we have the gold cleaned and brightened by the cyanide solution and in the very best possible condition for amalgamation, but we are confronted by new difficulties. The solution attacks the plates, rapidly eating them away, while the amalgam is hardened by the dissolution of the mercury until it becomes unmanageable and the plates inefficient. The difficulties encountered have generally been sufficient to result in the abandonment of amalgamation in cyanide solution after many careful and persistent efforts.

It is the object of this article to bring into notice a phase of the subject which seems to have escaped general attention.

It is well known that if metallic sodium be mixed with mercury and immersed in a watery solution the action of the sodium in going into solution as NaOH at the surface of the mercury will produce practically the same result as that occurring at the cathode of an electrolytic cell. If there is gold or other metals in the solution they will be deposited upon the surface of the mercury and amalgamated. But in plate-amalgamation the mercury and sodium is spread out into such an exceedingly thin film that the sodium is rapidly oxidized and carried off, the good effect of it being noticeable for only a very short time after dressing the plates. This is the main reason why amalgamators generally regard sodium as of little or no value.

Riffles Substituted for Plates.—However, if riffles containing mercury be substituted for plates the last named difficulty is overcome, and it is possible to keep the quantity of sodium in the mercury fairly constant. Some time ago the writer was in charge of a mill, treating sand and slime separately, where gold nuggets the size of a pea were sometimes found, although the ore averaged a little below $10 per ton gold. The ore was a weathered dump and contained some copper as azurite, but no native copper. The crushing was done in solution in a Chilean mill through 16-mesh screens. The mill was provided with plates, but they were taken out and riffles were substituted.

In cleaning-up it was found that the greater part of the amalgam was so fine that it could be squeezed through chamois leather, and the bullion from the riffles was found to contain about two pounds of copper for each ounce of gold and silver. It is evident that the copper was deposited from the solution, and that in addition to recovering free gold that passed through the screens, the riffles had been of service in mitigating troubles arising from copper in the zinc boxes.

It was aimed to keep the amount of sodium just below the point where the mercury would amalgamate a freshly brightened iron nail. To accomplish this some mercury containing a large amount of sodium amalgam was kept on hand in a dry, glass-stoppered bottle and a determined amount of this put into the riffles every four hours. No mercury was detected in the solution, and as none was floured, a weighing at the end of the run showed no loss except what might be expected in retorting.

Returning Part of Riffles Discharge by Tailing Wheel.—In operating riffles it is found that those of the best design will choke when the feed is temporarily diminished below full capacity, and some trouble is experienced in getting them started again. This trouble may be obviated by placing a small tailing wheel in such a manner that it will return a variable amount of the overflow back to the head of the riffles. By this means riffles of comparatively large capacity may be operated on a small amount of pulp. A section of the feed spout may be made to slide in telescope style, or a movable partition may be placed in the spout to cut out a variable amount of pulp.

I think it very doubtful if it would pay anybody to make riffles should they desire to use them with mercury, for this or any other mill purpose. The Pierce riffles, called the Pierce amalgamator, can be depended upon if given the proper amount of feed, and the sodium will protect the metal parts. They look somewhat expensive in first cost, but any riffles that will hold mercury and not pack with sand are open to the same objection. In operating riffles it is quite necessary to have a screen above them to intercept gravel that may occasionally come from a broken mill screen." JOHN RANDALL, *Metallurgical and Chemical Engineering*, April, 1914, p. 248. (G. H. S.)

DURIRON.—"Duriron is a non-corrosive iron alloy which is resistant to both alkalis and acids. Its exact composition and method of manufacture are not known, but it is supposed to be a modification of ferrosilicon, containing about 10 Si. In the *Journal* of May 2, 1914, p. 917, it was said that this material had failed to stand up as a tube-mill lining in the Chile Exploration Co.'s experiments in Chuquicamata ores, where it was exposed to solutions carrying $CuSO_4$, $CuCl_2$ and H_2SO_4. As a matter of fact, it did prove resistant to these solutions, and earthenware linings were chosen solely because they were thought to be cheaper."—*Engineering and Mining Journal*, May 16, 1914, p. 1014. (G. H. S.)

CONTINUOUS CYANIDE TREATMENT.—"A system of continuous cyanide treatment has recently been devised by Messrs. Menell & Grothe in which 'Pachuca' vats are arranged to operate in 'series.' The first vat receives the pulp from the mill after classification and dewatering, and the last after the passage of the pulp through each vat discharges it continually and by gravity to the filter plant. For practical purposes a 'battery' of four vats gives sufficiently good results.

The obvious advantages are the avoidance of loss of time in filling and emptying vats and the immense saving in labour and supervision. With a series of four vats, reckoning 48 hours as the time required to give the best economic extraction, it is calculated that only 6·25 % of the ore will receive less than this time of treatment, while 6·25 % will receive more.

The process cannot strictly be called continuous till the filtering arrangements are made continuous also, but the author anticipates future improvements in the cyanide process will be in this direction, and that costs will be still further cut down."—G. BASIL BASHAM, M.E., A.M.I.E.E., *Mining Journal*, July 25, 1914, p. 690. (H. R. A.)

INFLUENCE OF MOLYBDENUM ON CORRODIBILITY OF STEEL.—"In a paper by Messrs. J. Newton Friend and C. W. Marshall, read before the Iron and Steel Institute, it was shown that the presence of molybdenum in steel to a greater extent than 1 %, largely increases its corrodibility, at any rate when submitted to alternate wet and dry action."—*London Mining Journal*, May 9, 1914, p. 446. (H. R. A.)

MINING.

HYDRAULIC MINE FILLING THROUGH BOREHOLES.—"The most common method in use for getting underground the material used in hydraulic filling of the anthracite mines in Pennsylvania, is to shoot it down a borehole. These holes are put in with churn drills of a diameter to suit the quantity of material to be handled, usually 12 in. Through the surface material down to the bedrock, the drive pipe is protected by means of a lining pipe of metal or of specially vitrified terra cotta, as shown in Figs. 1 and 7. Metal lining pipe should be non malleable and capabled of being broken into small pieces and removed when worn out. It has special lock-flange connections, or bell-and spigot joints; standard flanges require too large a hole to be practicable. Bell-and-spigot joints are packed with oakum and cement mortar as the pipe is lowered in the hole. Terra cotta lining may have plain butt joints, or bell-and-spigot. Butt joints are reinforced with a triple thickness of burlap wrapping, extending for 9 in. on each side of the joint, and stiffened by 1 2 18 in. wooden strips held firmly in place with No. 10 galvanised wire, as shown in Fig. 3. Bell-and spigot joints are packed with hemp and mortar, as shown in Fig. 2. The joints between these linings and the rock hole require special nozzle pieces or shoes, as shown in Figs. 4, 5 and 6.

To insert a lining in the hole, a hemp rope of strength to support the full length of pipe is threaded through a single pulley supported above the hole, and the lower end of the rope is knotted to support a clamp under a circular plate of diameter 1 in. less than the hole in the rock. The cast-iron rock nozzle or shoe rests on the plate, and in turn supports the lining pipe proper. Two lengths of lining pipe are threaded on the rope, the rope and pulley hung from the support, and the loose end of the rope tightened with the drilling engine until the pipe hangs in a vertical position over the hole. The joints between the sections are then made and the pipe lowered till its top end is brought flush with the top of the drive pipe. The lower part of the rope is clamped in this position and two more sections are threaded on the upper part while the rope and pulley lie on the ground. Rope and pulley are again hung from supports, the two new lengths lifted by means of the drilling engine into a vertical position, and the two new joints made; the pipe is then lowered into the hole as before and the operation is repeated until the entire lining has been inserted. The space between the outside of the lining pipe and the inside of the drive pipe is filled with cement grouting.

Ordinary screw-sleeve gas pipe lining may be used, and it so used, is lowered into the rock for several feet to act as a rock nozzle as in the other cases.

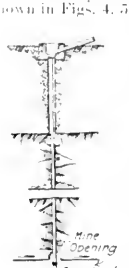


FIG 1 TYPICAL GENERAL ARRANGEMENT

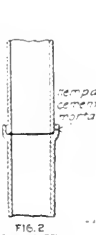


FIG 2. PACKING BELL AND SPIGOT JOINT

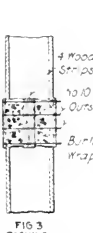


FIG 3. PACKING BUTT JOINT

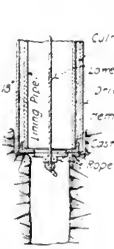


FIG 4 LOWERING DEVICE AND ROCK JOINT

FIG 5. ROCK NOZZLE

FIG 6. IRON SHOE ROCK JOINT

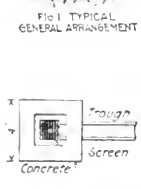


FIG 7 DETAIL OF SURFACE END

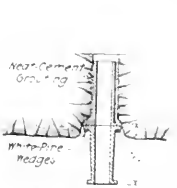


FIG 8. CASTING ON BOTTOM END

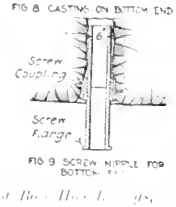


FIG 9. SCREW NIPPLE FOR BOTTOM END

Arrangements and Details of Bore Hole Linings for Small Flows.

Lining pipe is usually suspended by clamps over the receiving basin, which is generally made of concrete, and is protected by means of a screen, as shown in Figs. 1 and 7.

Where the borehole opens into the mine workings, there is used either a specially cast connecting piece, as shown in Figs. 1 and 8, or a short piece of wrought iron pipe is fitted with a screw sleeve and inserted up the hole for 2 ft. or 3 ft., as in Fig. 9. The space between this finishing piece and the sides of the hole, after packing with hemp, wedges or burlap, is filled with a grouting mixture of proper consistency."—*Engineering and Mining Journal* (*Bulletin 60, U.S. Bureau of Mines*), June 13, 1914, p. 1196. (G. H. S.)

WIRELESS MINE TELEPHONE.—"The unique system of telephoning here described was invented by Herr Reineke, in Germany, in the attempt to overcome the difficulties resulting from broken wires in ordinary systems of telephony. His system has now been in regular use for two years in the Carolinengluck Pit, Bochum, Westphalia, where no other method of signalling is used. The Wireless Telephone

Co., Ltd., London, is the owner of the British and Colonial patents.

The invention seems to be entirely original; in fact, the German Patent Office did not at first believe it to be practicable, but, on its being demonstrated, granted a patent without restrictions or anticipations being cited, which is most unusual.

While called 'wireless,' the system is entirely distinct from that commonly used in wireless telegraphy.

In this system the transmission and reception of speech occur in the same manner as in ordinary telephoning, but there the resemblance ends. Instead of the current from the transmitter being conducted along the wire to the receiver, it is converted to high pressure by means of a transformer contained in the telephone case. The high pressure or secondary winding of this transformer is connected to the ordinary rails or pipes in the mine and communicates to them a charge of electricity, which spreads over the whole system of rails, and varies with the fluctuations of the primary current in the transmitter circuit of the telephone. It is unnecessary to make any special provision to insulate the rails or pipes, or to connect the adjacent lengths electrically.

It is interesting to note that the invention was led up to by the discovery that the conditions of distribution of electric charges underground are entirely distinct from the conditions that hold on the surface of the earth, and the working of the system largely depends on this fact, which was demonstrated for the first time by the inventor.

Speech can be received by connecting telephones to the rails or pipes at any point of the system, by bridging a sufficient length so as to obtain a suitable difference of pressure; by connecting between the rails and pipes, if these exist; or, as is sometimes convenient in the case of portable instruments, by connecting to an antenna consisting of a loose coil of wire, which may be laid on the floor or hung on the timbers.

As in the ordinary telephone, a bell is used for calling attention, the bell being operated by a specially designed relay. On the button being pressed at the transmitting stations, all the telephone bells will ring, and all the telephones can receive the message by taking the receiver off the hook, which operation cuts off the bell and connects the receiver in the usual manner. It is possible to tune the relays so that each will respond only to its own particular signal, and thus any one station can be called, but in mining work this is usually not necessary or desirable, and the different stations are called by giving one, two or three rings, as prearranged, while it is always possible, by a special signal, to call all the stations and speak to all simultaneously. The current required is furnished by an ordinary battery; in fact, in outward appearance the telephones are exactly similar to an ordinary mining telephone.

The portable instrument, which can be carried about and used in any part of the mine to which the rails extend, is contained in a strong metal case, the total weight being about 20 lb. This telephone can either ring up the other stations or be rung up by them. A smaller or pocket instrument is also made, which can ring up other stations, but cannot be called by them. With this instrument messages can be sent to or received from any of the fixed telephones, the only difference between it and the heavier portable type being that the other telephones cannot call it by ringing its bell. Thus each official provided with an instrument can give notice of any accident and ask for assistance from any point in the mine.

A telephone can be fixed at any convenient position on the surface, and connected to the headframe, if of iron, or to the hoisting-rope or pipes which go down the shaft. This telephone can speak to any of the telephones underground if necessary, but it is usually found more convenient to arrange for it to speak to the station at the pit-bottom only, and for this station in turn to speak to the underground telephones. In metalliferous mines, where a number of different levels run from the shaft, communication can be established from one level to another as required, the presence of the ore bodies not affecting the speaking in any way.

A further most important application of the system is that of speaking from the moving cage to the hoist house, and for the latter position the inventor has designed an ingenious method of operation. The telephone is mounted on a standard fixed close to the hoisting engineer's ear. On receiving a call, he has only to put his ear to the receiver, and a slight pressure is sufficient to switch in the telephone and enable him to communicate with the cage, without removing his hands from the levers. Various methods of connection are possible for the telephone in the cage, one of the most satisfactory being connection into a loop formed by the hoisting ropes, and by the balance rope (or, if this is not used, by a wire suspended below the cage in the same way). It is then possible to speak from either cage to the surface or to the pit bottom, without any alteration in the ordinary hoisting arrangements. The surface telephone is preferably connected in circuit with a loop of wire fixed to the headgear, or in the shaft in any convenient place."—*Engineering and Mining Journal*, June 13, 1914, p. 1192. (G. H. S.)

SAN JUAN DEL REY MINE TEMPERATURES.—"Temperatures of air and rock in the St. John del Rey mine, Brazil, during the four hot months of 1913-14, namely, December, January, February and March, are as follows, according to the annual report of the superintendent, George Chalmers:—

Horizon number.	Depth on incline, ft.	Air, deg. F.	Rock, deg. F.
Adit	300	77	78
8	2,300	84	82
10	2,800	85	85
11	3,100	85	87
12	3,400	86	88
13	3,700	87	89
14	4,000	89	92
15	4,300	90	94
16	4,600	91	97
17	4,900	96	101
18	5,226	97	104

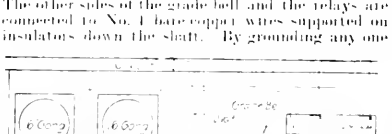
All temperatures were taken in the downcast. A large Sirocco fan is in continuous operation, and another is being installed. When the workings are all connected with the main ventilating system, the following air temperature is expected: horizon 18, 92°; 17, 91°; 16, 90°; 15, 89°; 14, 88°; 13, 87°; 12, 86°; 11, 85°; 10, 84°; and 8, 82°. Rock temperature at the time of opening horizons is as follows: 8, 85°; 10, 89°; 11, 91°; 12, 94°; 13, 96°; 14, 99°; 15, 101°; 16, 103°; 17, 106°; and 18, 108°. The past year shows a slight reduction compared with the previous year."—*Mining and Scientific Press*, July 11, 1914, p. 67. (G. H. S.)

MADAGASCAR GRAPHITE.—"The expectations expressed in the beginning of 1913 with respect to the development of the graphite industry in Madagascar have been fully realised. Although in the

last quarter of that year there were no new discoveries, yet the number of new mining claims declared in the districts yielding graphite amounted to 1,690. At the end of 1913 there were in existence 2,427 claims, as compared with 942 at the termination of the year 1912. The total number of metric tons raised during the past year was not less than 8,000, of which 6,314 tons were exported, which is more than double the production of 1912. The exportation of graphite commenced with 19 tons in 1909, which was increased to 545 in 1910 and to 1,247 in 1911. The best qualities contain from 90 to 97% of carbon." — *Mining Journal*, May 9, 1914, p. 447. (H. B. A.)

BARE-WIRE ELECTRIC SIGNAL SYSTEM. — "The Penn Iron Mining Co., in Michigan uses a rather unusual and highly interesting system of electric signalling. An alternating current of 110 volts taken from the lighting system is reduced to 30 volts through a small transformer. This 30 volt circuit is used to ring what is called a grade bell, *viz.*, a signal to indicate the class of ore hoisted, and also to operate two relays, one of which rings the skip bell in the hoist house by closing a 110 volt circuit, while the other similarly rings the cage bell.

One side of the 30 volt circuit is grounded, the other connected to the grade bell and the relays. The other sides of the grade bell and the relays are connected to No. 4 bare copper wires supported on insulators down the shaft. By grounding any one



Circuit-wires for Ringing and Dust-Signal System.

of these copper wires, a current will flow through the grade bell or a relay and give a signal.

The bell wire for the cage is near the centre of one side of the compartment so that it is almost impossible to reach and ring it unless one stands on the cage. This prevents ringing the cage bell when the cage is not in sight, a frequent source of accidents. A short piece of flexible wire with a pierced plate No. 4 wire on its end is fastened to the iron work of the cage so that the shaft signal wire can be grounded from any point in the shaft, whether the cage be moving or at rest.

The skip and cage bells are 16 in. in diameter and are set in a local 110 volt circuit, which is closed and opened by the relays mentioned. The blow on the gong is struck by a bar in a heavy solenoid. An indicator which registers the number of bells rung is also actuated and a lamp is lighted." — *Trans. N. E. M. E., Feb., 1914. Abstract in Mining Journal*, June 6, 1914, p. 1146. (H. B. S.)

IMPREGNATION OF MINE TIMBER. — "Pine wood logs were treated by both the vacuum and dipping processes. In the former, the timber was subjected to 650 mm. of vacuum for an hour and then the impregnating agents were added under 4 to 7 atmospheres pressure for 1 to 2 hours. The materials tried were basilate (NaF and dinitrophenolamines), Bayer salt (Na_2SO_4 and Na oxy-mercuric chlorophenol), Hg silicate ($HgCl_2$, Na silicate, NH_4OH), Glueksauf salt (nitrated phenols), Viesal salt (Cu and Zn salts, NH_4OH, phenol), Ruping tar oil. The dipping process is far less expensive. The materials used in this were hot tar oil, creosophenol, Na silicate with and without $Ca(OH)_2$, $HgCl_2$. Periodic observations are to be made, extending into several years." — O. DUBELSTEIN and WEBER, *Engineering*, 97, 692. *Chem. Abs.*, July 10, 1914, p. 2470. (J. G.)

MISCELLANEOUS.

COALING AT DELAGOA BAY. — "The new McMyler coaling installation at Delagoa Bay, South Africa, will have a normal loading capacity of 400 tons per hour, but this can be increased if necessary to 600 tons. The structure, which is some 66 ft. high, raises the coal truck by means of a cradle up to the height required by the state of the tide, turns it on its side, and throws the coal into an apron ending in a telescopic tube, through which the coal descends steadily into the ship's hold so as to avoid breakage. This tube, which is always kept full, has a movable end worked by an electrical winch fixed to the apron, which distributes the coal evenly in the hold, so that there is a considerable reduction in the trimble attendant on hand labour in trimming the coal. For the purpose of weighing, two scales are employed. The whole installation is worked exclusively by electricity." — *Iron and Coal Trades' Review*, May 29, 1914. (H. M.)

USE OF GOLD IN CERAMICS. — "Gold is used on pottery either for gilding or as a colouring agent in glazes. Its value for gilding lies in its resistance to oxidation and the fineness of layer which can be produced, for which reason it is not an expensive stain. The aid of a flux is employed in application, and mercurous oxide is found to reduce the amount of gold required. A mixing of the composition: 30 parts by weight of precipitated gold, 15 of black mercurous oxide, 2.5 of basic bismuth nitrate, and 0.3 of melted borax, is recommended; the addition of 3 parts of silver carbonate to this produces a greenish tint. After firing in the muffle the gold has a matt surface, and must be polished with blood stone or agate. A cheaper process, which does not require polishing after the burn, is the use of 'Glanzgold' or 'Meissen gold,' a preparation of gold solution with an organic medium such as turpentine, lavender oil and a balsam or resin. As a colouring agent in glazes, gold produces shades from violet to purple and red, of which the best known is purple of Cassius, a lake of precipitated gold with metallic hydroxide, preferably that of tin, which was formerly regarded as a definite compound. The presence of ammonia or alum in the stannous chloride solution produces shades from dark to rose purple, and the tint may be further modified by the substitution of antimony or magnesia for tin. Choice of flux materially influences the colour; various recipes are given containing lead oxide, borax or felspar; and kaolin may be used as a base for the lake. In lead glazes 1 part of gold in 1,000 parts of glaze powder will produce a deep red glaze; and in leadless glazes, 1 in 10,000. An underglaze stain can be produced from 50 parts of precipitate, 50 of alumina and 1 of gold chloride. Gold may be applied under a glaze on places previously moistened with gum tragacanth solution." — A. GRANGER, *La Ceramique*, 1912, 262 and 1913, 3; *Sc. Abstr.*, 1914, *17*, 359-360, 373-375. — *Journal of the Society of Chemical Industry*, xxxiii., 13, July 15, 1914, p. 693. (J. A. W.)

RAINFALL AT JOHANNESBURG (JOUBERT PARK), TRANSVAAL PROVINCE.
Longitude 28° 3′ E. Latitude 26° 11′ S.

Year.	Jan.	Feb.	March.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Total for Year.
1888		–	–	–	–	—	1·45	0·89	0·34	3·59	2·23	4·42	12·98 inches
1889	5·13	3·15	2·00	1·30	0·00	0·00	0·00	0·00	1·00	0·25	5·02	2·00	19·85 ,,
1890	6·14	3·47	1·61	2·17	0·00	0·00	0·75	0·00	0·10	0·18	3·89	7·63	25·94 ,,
1891	10·21	8·09	5·55	5·16	1·12	0·71	0·00	0·56	0·05	1·44	2·32	5·64	40·85 ,,
1892	6·61	5·11	3·83	1·41	1·09	0·09	0·00	0·00	2·03	3·68	2·20	1·49	27·54 ,,
1893	7·77	5·97	3·49	1·19	0·00	0·18	0·28	0·00	2·76	2·45	8·09	5·51	37·69 ,,
1894	4·59	7·16	4·89	1·50	2·71	0·14	0·00	0·85	2·32	0·45	4·92	5·80	35·43 ,,
1895	3·82	4·57	7·02	2·01	0·23	0·12	0·30	0·00	0·33	0·66	3·99	6·99	29·14 ,,
1896	1·65	3·74	1·95	1·70	1·26	0·64	0·00	0·79	0·45	2·40	2·46	6·22	23·23 ,,
1897	9·71	4·99	3·94	0·56	0·53	0·00	0·00	0·19	0·10	3·27	2·19	3·50	28·98 ,,
1898	8·93	5·95	2·82	0·44	1·50	0·00	0·00	0·20	1·01	0·64	2·60	4·90	28·99 ,,
1899	7·12	2·91	2·08	2·97	0·85	0·12	0·34	0·04	2·02	2·68	2·87	4·61	28·57 ,,
1900	9·28	4·17	1·10	1·87	0·00	0·14	0·66	1·52	0·00	2·39	7·94	3·73	32·80 ,,
1901	5·20	2·72	6·96	4·28	0·26	0·22	0·00	0·20	1·68	5·43	4·12	4·41	35·48 ,,
1902	7·64	4·63	5·06	2·83	0·17	0·05	0·17	0·17	0·85	2·91	4·24	4·79	33·51 ,,
1903	4·61	3·42	2·31	4·98	1·21	0·00	0·00	0·00	0·56	1·59	4·23	4·32	27·23 ,,
1904	4·20	7·84	10·91	0·50	0·38	0·45	0·00	0·05	0·28	1·40	6·51	4·08	36·00 ,,
1905	4·00	4·07	3·82	1·67	0·26	0·00	0·00	0·10	0·36	0·78	7·47	5·04	27·57 ,,
1906	2·37	8·10	3·83	1·35	0·13	0·00	0·00	0·00	0·63	4·53	6·08	5·20	32·22 ,,
1907	7·88	7·81	2·84	3·39	0·27	0·00	0·00	0·00	2·79	2·56	5·75	4·06	37·35 ,,
1908	3·27	2·36	6·50	0·19	0·00	0·02	1·03	0·23	0·80	4·20	3·83	4·23	26·06 ,,
1909	19·98	8·68	5·76	0·33	0·93	0·00	0·60	2·09	0·55	1·81	4·17	7·67	52·57 ,,
1910	4·93	4·73	7·84	0·52	0·04	0·34	0·00	0·00	2·50	5·60	4·32	5·98	33·20 ,,
1911	5·19	3·84	2·94	3·30	4·09	0·00	0·58	0·07	0·15	2·63	3·34	3·38	29·51 ,,
1912	3·40	6·55	2·19	3·54	0·79	0·03	0·01	0·00	0·38	1·25	2·30	5·94	26·38 ,,
1913	2·81	3·71	4·56	2·29	0·40	0·00	0·17	1·15	0·47	4·55	6·09	4·38	30·18 ,,
Means	6·26	5·11	4·23	2·06	0·72	0·13	0·23	0·35	0·94	2·39	4·24	4·84	31·50 inches

From Report from the Select Committee of the South African Senate on Droughts, Rainfall and Soil Erosion, June, 1914, p. 20. (W. A. C.)

EVAPORATION AT JOHANNESBURG OBSERVATORY.
Longitude 28° 3′ E. Latitude 26° 11′ S.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Total for Year.
1904			—				6·89	7·79	8·96	10·25	10·68	9·42	53·99 inches
1905	7·94	6·10	6·10	6·05	6·42	5·71	5·47	6·75	8·82	11·46	7·38	7·86	86·06 ,,
1906	8·23	6·63	5·24	6·20	6·01	4·67	5·96	7·16	8·57	6·83	6·73	6·02	78·25 ,,
1907	4·78	4·60	5·76	3·44	5·56	4·69	4·84	6·70	7·00	6·96	7·28	6·54	68·15 ,,
1908	7·90	7·18	5·44	6·79	5·38	4·68	3·89	6·15	8·40	8·76	6·74	7·30	78·61 ,,
1909	3·95	4·54	3·89	5·30	3·98	3·69	4·28	6·51	6·43	8·58	8·04	6·14	65·33 ,,
1910	6·09	4·74	4·72	5·30	5·25	3·53	5·01	6·19	6·87	5·58	6·75	7·61	67·04 ,,
1911	5·70	6·02	4·27	3·93	3·04	3·50	4·16	5·35	8·27	7·94	7·42	8·20	68·00 ,,
1912	7·93	5·46	5·08	3·79	4·05	4·06	3·75	5·54	8·12	7·99	9·09	6·14	71·00 ,,
1913	6·59	5·11	5·32	3·00	4·43	3·80	4·72	5·62	6·54	7·18	6·76	8·37	67·44 ,,
Means	6·57	5·60	5·09	4·87	4·90	4·26	4·89	6·40	7·80	8·15	7·69	7·37	73·59 ,,

From Report from the Select Committee of the South African Senate on Droughts, Rainfall and Soil Erosion, Appendix B, June, 1914, p. 23. (W. A. C.)

Abstracts of Patent Applications.

(C.) 341·13. Urlyn Clifton Tainton (1), Malcolm Foerster Lambe Amye Aymard (2). Improvements in the electrolytic recovery of metals from their solution and in apparatus therefor. 16.7.13.

This is an electrolytic process for the precipitation of metals from solution in which the solution is passed under pressure through a porous cathode, the interstices of which are very minute.

(C.) 293·14. Aubrey George Eneas. Improvements in or relating to spray nozzle and method of distributing liquids. 2.7.14.

This application relates to a method of distributing liquid over an extended area which consists in simultaneously producing both rotary and non-rotary jets, causing the mixture of the said jets within a confined space, the mixed jets to issue from the said confined space in a spray of uniform homogeneity. Other claims include apparatus for producing the above result.

(C.) 302/14. Herman Alexander Wagner. Improvements in and relating to process of separating metals from ores. 9.7.14.

This application claims a process of extracting metals from their ores by electrolysis, which comprises passing a current from an anode to cathode, both being immersed in a suitable electrolyte, holding the ore in the anode compartment, the liquid in the anode compartment being kept separate from the liquid in the cathode compartment, and maintaining a depolarising material including an oxidising agent, in the path of the current, between said anode and said cathode.

(C.) 312/14. William Barnard Easton. Improvements in tube mills. 15.7.14.

This application relates to an improved method of discharging pulp from tube mills. By means of the combination of a perforated plate and adjustable vanes, the quantity of material discharged is controlled independently of the inflow.

The apparatus is so designed that it can be adjusted to accelerate or retard the discharge of the material being ground.

(C.) 324/14. Francis J. Hobson (1), Charles Edward Meyer (2). Improvements in ore separators. 16.7.14.

This application is for a rotary pulp classifier, the rotor or drum of which consists of end discs, a hub and a series of transverse vanes dividing the drum into radial pockets, the whole mounted in an open bottom hopper supplied with adjustable means for dividing the material discharged.

In operation the pulp is delivered at the top of the slowly revolving drum into the radial pockets, and the vanes on arriving at a position low enough discharge the water and fine material on to one side of the adjustable divider while the coarse and heavier product is carried over to the other side.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

compiled by C. H. M. Kisch, F.M.C. (Chart Inst. P.A. London), Johannesburg (Member).

(N.B. In the list (P.) means provisional specification, and (C.) complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)

(P.) 290/14. Charles Hansen. Improvements in attachments for rock drills for supplying water thereto. 7.7.14.

(P.) 300/14. John Mills Haxtable. An appliance to facilitate the lighting of fuses used in blasting operations and to increase safety. 8.7.14.

(C.) 301/14. Edvard Bjeggen. Automatic lubricating device for compressors. 9.7.14.

(C.) 302/14. Herman Alexander Wagner. Improvements in and relating to process of separating metals from ores. 9.7.14.

(P.) 305/14. Seguin Fisher. Automatic water supply to hollow rock drills. 10.7.14.

(C.) 306/14. Thomas Henry Bradbury. Improvements in and relating to rock drills. 10.7.14.

(P.) 307/14. William Bruce Paterson. Improvements in jockeys or rope grips for mechanical haulages and the like. 11.7.14.

(P.) 309/14. James Hamilton Anstruther Macadam (1), John Thirwell (2). Improvements in rock drilling machines. 13.7.14.

(P.) 310/14. James Hamilton Anstruther Macadam (1), John Thirwell (2). Improvements in rock drill cradles. 13.7.14.

(P.) 311/14. Harry August Stockman. Improvements in concentrators or machines applicable for separating particles of metal or precious stones from the matter associated therewith. 15.7.14.

(C.) 312/14. William Barnard Easton. Improvements in tube mills. 15.7.14.

(P.) 313/14. Thomas Harrison (1), Daniel Corlett (2). An improved type of self lubricating roller. 16.7.14.

(P.) 314/14. Cecil Doveton Dunlop Helps. Improvements in soil working implements. 16.7.14.

(C.) 315/14. Charles Benant. Improvements relating to the manufacture of drill steels and similar tools. 16.7.14.

(C.) 316/14. Charles Benant. Improvements relating to the manufacture of drill steels and similar tools. 16.7.14.

(C.) 317/14. Charles William Fozer. Improved construction of retort for the distillation, carbonisation, and the like, of coal and other carbonaceous materials. 16.7.14.

(C.) 318/14. Orville John Parker. Improved process for treating carbonaceous material. 16.7.14.

(C.) 319/14. Jean Charles Grieve. A process and apparatus for the manufacture of dry sugar in grains, moulded sugar, sugared products or chemical products without centrifuging and without drains. 16.7.14.

(C.) 320/14. John van Nostrand Dorr. Apparatus for the separation of finely divided solids from liquids. 16.7.14.

(C.) 324/14. Francis J. Hobson (1), Charles Edward Meyer (2). Improvements in ore separators. 16.7.14.

(C.) 322/14. Henry Moore Sutton (1), Walter Livingston Steele (2), and Edwin Goodwin Steele (3). Improvements in processes of and apparatus for sizing and classifying comminuted materials. 16.7.14.

(C.) Henry Rawes Whittell. Improvements in and relating to earth excavating machines. 18.7.14.

(P.) 325/14. Charles Jack David Symons. Improvements in electric fuse holders. 18.7.14.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

CHEW, R., *c/o* Johannesburg; New Fortuna G. M. Co., P. O. Box 124, Barberton.

ELLIS, L. L., *c/o* Langbaagte; Government Gold Mining Areas, P. O. Box 28, Brakpan.

HERBERT, CHAS., *to* Shamva Mines, Ltd., Shamva, Southern Rhodesia.

MCLENNAN, A., *c/o* Johannesburg; Eileen Alannah Mining Co., Gatooma, Rhodesia.

ROBERTSON, J. B. H., *to* Scotland; Mysore G. M. Co., Marikuppam, Mysore State, South India.

TAYLER, W. V. C., *to* Germiston; Sub Nigel, Ltd., P. O. Box 50, Nigel.

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Vol. XV. OCTOBER, 1914. No. 4.

Proceedings
AT
Ordinary General Meeting, October 17th, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building on Saturday, 17th October, Mr. J. E. Thomas (Vice-President) in the chair. There were also present:—

19 Members: Prof. J. A. Wilkinson, Messrs. E. Pam, W. R. Dowling, Prof. J. S. Cellier, H. Meyer, C. Toombs, John Watson, H. A. White (Members of Council), W. Beaver, B. A. Cooper, E. H. Croghan, T. N. Dewar, J. M. Dixon, H. W. Gill, P. T. Morrisby, E. A. Osterloh, W. S. V. Price, A. Thomas, and E. M. Weston.

5 Associates and Students: Messrs. O. A. Gerber, W. Human, A. King, G. F. Nisbeth, and F. J. Pooler.

3 Visitors, and Fred. Rowland, Secretary.

MINUTES.

The Minutes of the last Ordinary General Meeting were confirmed.

VOTE OF CONDOLENCE.

The Chairman: It is with deep regret I have to announce the death of Mrs. Stanley, the wife of our President, Prof. G. H. Stanley. I am sure you will all join with me in expressing to Prof. Stanley and his family our heartfelt sympathy on the occasion of his great loss. I will ask all present to signify their sympathy.

Those present signified their sympathy by rising.

NEW MEMBERS.

The Chairman: I regret to have to state that this is perhaps the first meeting on record at which there are no new members standing for election. This is a very bad state of affairs, and I hope it will not occur again.

GENERAL BUSINESS.

CHEMICALS AND STORES FROM GERMANY, ETC.

Mr. John Watson (*Member of Council*): There is a matter to which I would like to call the attention of members, that is the fact that the President of the Board of Trade has appointed an influential and representative Committee to consider and advise as to the best means for obtaining, for the use of British industry, sufficient supplies of chemicals, etc., hitherto produced by Germany and Austro-Hungary. The last number of the Journal of the Society of Chemical Industry calls our attention to this Committee, of which Viscount Haldane is Chairman, and Mr. F. Gossling, Secretary. The latter's address is:—

Commercial Intelligence Branch,
Board of Trade,
73, Basinghall Street,
London, E.C.

Certain articles have been going up very much in price out here. Of course, the question of cyanide has been dealt with by the Chamber of Mines, which is satisfied that we can get plenty of cyanide to supply all the requirements of this gold industry. I will mention three or four other things:

Mercury, some three months ago, could be bought for something like £7 10s. per bottle, but recently the price went up as high as £22 10s. Apparently, a "ring" was attempted. Mercury was obtained from Idria, in Austria; but it also occurs in Almaden, Spain, in New Almaden, California, and in China.

Porcelain basins and crucibles are largely used in assay offices. I understand that when the present stock on hand is used up it is doubtful where further supplies are to come from. I gave the Derby Porcelain Co. a hint regarding this matter last July, when visiting their works.

Salicylic acid is a household medicine, as soda salicylate, or "Aspirin," for rheuma-

tism, etc. Sodium salicylate has gone up in price from about 1s. 6d. per bottle, wholesale rate, to 10s. in the last few months.

Potassium permanganate and *pyrogallic acid* are also reagents which have considerably increased in price.

In addition to the Committee mentioned above, the Society of Chemical Industry and the London Chamber of Commerce have also each appointed a Committee, and they are now seriously doing their best to help to establish the manufacture of such products in Britain and her Colonies. I am calling attention to this matter because, I think, buyers on the mines throughout South Africa should know to whom to apply in case they are being hampered by high prices. The Committee has got to work, and published lists of:—

a. Articles which inquirers wish to buy, and

b. Articles which inquirers desire to sell. (See Journal Society Chemical Industry, 15th Sept., 1914.)

They are filing catalogues and have a large sample room for goods, which have hitherto been made in the countries with which we are now at war, but which it is hoped, ere long, to obtain from Great Britain, her Dominions, or friendly countries.

The Chairman: I think our thanks are due to Mr. Watson for drawing attention to this source of supplies. At the same time, I should like to point out that this is a most excellent opportunity for people in South Africa to see if they cannot themselves manufacture many of the items mentioned.

NOTES ON THE USE OF POTASSIUM PERMANGANATE AS CYANICIDE IN SAND-FILLING SOLUTIONS

By R. A. Cooper, Member.

During the routine analyses to determine the degree of decomposition of cyanide in the operation of the sand-filling plant, it has appeared on several occasions, where from some cause the permanganate had been omitted, that the cyanide contents were not materially affected. To verify this a series of special samples was taken. These were as follows:—

1. Circulating solution before any additions of permanganate were made, though sand-filling had been rapid and continuous for three hours.

2. Overflow at dewatering cones, rate of sand discharge having been lessened and permanganate added until all solutions were strongly coloured.

3. Returned circuit water as used to flush sand from tip. Strong permanganate coloured solution.

4. Overflow of dewatering cones when sand filling had been resumed at full capacity. The permanganate was immediately destroyed on contact with sand.

In the above four samples the cyanide, as determined by evaporation with ammonium sulphide, was practically constant at 0·0015 KCN—only showing a slight increase to 0·0017 on the fourth sample.

The strong excess of permanganate in circuit, which existed when samples 2 and 3 were taken, was maintained for a considerable time before sampling to allow a maximum effect, if any, to be observed. Sample 3 is exactly the same solution as 2 but has retained permanganate during the time necessary to complete circulation. That additional length of time in contact did not result in any alteration of cyanide percentage, and it was still unaffected after one hour circulation.

When the present East Rand Proprietary Mines plant is working at a capacity of about 900 tons of sand per day, with a *circulating* solution of 20,000 gallons, it is not economically possible to maintain coloration by permanganate in the solution: it is, in fact, doubtful whether it is practically possible to colour it without extremely large additions of permanganate.

During the present tests a strong stream of fresh water was run through a box containing a large quantity of potassium permanganate crystals and the resulting solution went direct to the circulating water immediately after that water had overflowed from the cones. This gave a very strongly coloured circulating solution, but it was not even then possible to maintain the colour in full circuit, as it was immediately destroyed on contact with the sand at the tip. In this connection I might refer to Mr. A. F. Crosse's remark that "It is a very different problem, using an oxidising agent on a very dilute cyanide solution by itself to using the same agent in presence of sand containing pyrite, and in practice this would be the case." (*Journal* of this Society, Vol. XI., No. 1, 1910.) Even were it economically possible to maintain the permanganate excess in entire circulating solutions it

appears doubtful whether any real benefit would be obtained.

In a plant where the sand tip is from 50 to 100 yards from the dewatering cones, the sand is in the solution for only a very few minutes and very short contact results; the solution itself is constantly changing owing to the amount which is discharged with the sand (especially great when breakaways occur in the cones, owing to irregular sand feed)—and consequent make-up with fresh water to the circuit.

If the action of the cyanicide were immediate, the brief contact would be adequate, but the action rather appears to be comparatively slow. Mr. White, in his comprehensive paper in June, 1910 (Vol. X., No. 2), and further remarks in November, 1913, does not appear to be perfectly clear on the point of time required for the various reactions—as witness the following quotations:—

> "In practice, if permanganate be used, four hours contact must be given and the solutions leaving the vats must be distinctly red";

also:

> "That completion of the reaction may take as long as an hour in unfavourable cases":

and:

> "That sulpho-cyanide destruction is complete at 15° C. *in a few hours.*"

In present practice the leached sand residue is tipped on to a grizzly and immediately washed away by the alkaline permanganate solution, which is decolorised, and the resulting pulp flows to dewatering cones, and thence direct to a borehole where a small amount of fresh water is added to facilitate flow in underground launders. The contact is very brief and dilution very great.

Sand residues carry about 12% moisture—cone underflow 28%, and the addition at borehole brings *total* water used to about 37%—that is 60,000 gallons of water to 900 tons sand. The water carried in the cone underflow contains not more than 0·002% KCN, and as this is further diluted, the actual amount of cyanide in drained sand in stopes is infinitely small. In no case does the sand receive permanganate treatment on the surface for more than a very small fraction of even the shortest time considered necessary by Mr. White.

To determine definitely, if possible, what time is necessary for decomposition of various cyanide compounds by permanganate, the following solutions were allowed to stand with permanganate and were tested at intervals:—

1. Normal solution from sand-filling plant containing 0·0013% KCN and 0·0045% KCNS.

2. Pure sodium cyanide—0·0062% KCN.

3. Pure sodium-sulpho-cyanide—0·0062% KCN.

4. Sodium-zinc-cyanide—0·002% KCN.

The strength of these solutions was determined by the usual evaporation with ammonium sulphide and subsequent colorimetric test, and is expressed in terms of potassium cyanide.

Tests of the solutions with permanganate were made at 15 minute intervals for three hours—then at hour intervals, and later at longer intervals, for four days. The test used was by the hydrocuprulignone solution on filter paper which was placed inside the neck of a flask containing an acidified portion of the solution to be tested. The working solution, pure cyanide and sulpho-cyanide solutions were decomposed regularly and were obviously considerably affected by the permanganate, but nevertheless gave a test after four days, sufficiently positive to demonstrate that cyanide decomposition was incomplete and that the solution was still capable of evolving minute quantities of hydrocyanic acid in the presence of sulphuric acid; the working solution gave a distinctly stronger reaction than either of the other two. The sodium-zinc-cyanide reacted vigorously at all stages of the experiment, and appeared to be slightly—if at all—affected by the permanganate even after four days' contact. As a very large proportion of the cyanogen in working solutions appears to be in combination as the sodium zinc cyanide compound, the fact that this compound is extremely slowly decomposed by the permanganate is very important.

In the comprehensive paper on cyanicides, written by Dr. Moir and Mr. Jas. Gray, permanganate was apparently not deemed worthy of consideration, and though Mr. White discussed it at length, and appears to have been satisfied that ultimate decomposition of cyanides by permanganate is possible, he admitted failures in treating working solutions, and attributed them to the fact that the tests were made on a cold day. Undoubtedly a certain amount of cyanide decomposition does take place, because generation of ammonia underground is obvious, and by gradual

collection of the gas in poorly ventilated areas it becomes at times a nuisance; thus the delayed action of the cyanicide may be a double source of danger: firstly, because it may allow of evolution of hydrocyanic acid from the most recently filled areas, and, later, when a slight action has taken place, the formation of ammonia is troublesome.

In view of the high cost of most cyanicides and the difficulties of their practical economical application and chemical control, it is of interest to note that their object appears to be efficiently attained by purely natural means.

Recently occasion arose to drive a tunnel through a sand dump, and the opportunity was taken to test the air very carefully for hydrocyanic acid, as the ventilation was poor and the unwashed sand, of original high cyanide contents, had become very acid. It was not possible at any time to detect hydrocyanic acid in the air, but "Prussian blue" precipitate was abundant in the sand.

In sand-filling work the sand is sent underground in a strongly alkaline solution which, on contact with mine water containing ferrous and ferric sulphates, precipitates ferrous and ferric hydroxides. The former is a very powerful cyanide absorbent —being used medicinally as cyanide antidote—and is quite capable of combining very rapidly with any free cyanide present. Samples of sand from stopes that have been filled for some time, and through which mine water has since percolated, show a considerably higher percentage content of iron than any original sand. A sample recently tested contained 7% Fe, and thereby gave evidence of considerable iron precipitation in situ.

It appears that a high alkalinity in sand sent underground would solve the problem of safely disposing of cyanide in extremely dilute solutions. To determine exactly the condition of solutions actually delivered to, and drained from, an area being filled, two samples were taken underground and tested. The solution as delivered to the stope contained 0·0011% KCNS and 0·0004% KCN (cyanogen compounds other than sulpho-cyanides expressed as KCN) and was strongly alkaline,—the drainings from the same area at the same time contained traces of sulpho-cyanide, but *no other* cyanogen compound; they were neutral to methyl-orange indicator, and contained a considerable amount of ferrous-sulphate.

If the above remarks throw sufficient doubt on the claims of permanganate as an efficient cyanicide to stimulate further research on the subject their object will have been fully met.

In conclusion, I have to acknowledge the help of Mr. E. H. Johnson, who made many suggestions for experiments during the preparation of these notes.

Mr. H. A. White *(Member of Council):* The Society is to be congratulated upon this very useful paper, and the author upon the endeavours he has made to throw further light upon this subject.

The paper is concerned with both the practical aspect of the question and the theoretical consideration of the speed and limit of reaction with dilute solutions. And the practical result of the operations is to deliver sand in the stopes with an infinitely small amount of cyanide remaining therein. The author quotes 0·0004% KCN as a normal figure, and if it be noted (1) that the solution is still alkaline and that the effective dilution of the solution associated with the residue from the vats is 38 to 12, or, roughly, 3 to 1, it will be obvious that the method is practically perfect, and if permanganate is not really the responsible agent, it is difficult to suggest what is. The author hints that ferrous sulphate may be concerned. This seems impossible, as the solution entering the stopes is said to be strongly alkaline, but he might throw a further light by determining the ferrocyanide present as compared with that in the residue and the cyanogen to be accounted for. The observation that the solution value leaving the cones is considerably reduced (from 0·0015% KCN to 0·0004%) at delivery to stope, and further reduced in the stope drainings (elsewhere even while alkaline) confirms experience on other mines and suggests the possibility that the hydrated manganese dioxide in fine colloid suspension which is formed, has the power under the special circumstances of enabling the atmospheric oxygen to complete the work of cyanide destruction. I have not been able to verify this, but have had no time to execute any experiments which could be called crucial, and am uncertain if experiment No. 1 of the author's supports the idea. Before dismissing the practical side of the question I would suggest that the author might give us the alkalinity of the make-up water used for the circulating solution, the "organic matter to permanganate" and the effect, if any, it has upon cyanide. (See the *Journal*, Feb., 1906, where Heymann found a con-

sumption of 0·08 lb. of KCN per ton of water.)

Dealing with the purely experimental side of this useful paper, I feel bound to enter a protest against the light-hearted manner in which neglect of addition of permanganate is referred to, and the omission of the qualifying conditions under which it was deemed safe to carry out experiment No. 1. No doubt the author will explain these matters, but as they stand now they offer a highly dangerous precedent, and I consider that permission in writing of the responsible authority should be obtained before they are repeated elsewhere. I am, of course, perfectly certain that in this case there were substantial grounds to warrant the test, and that the author will state them in his reply.

The four first experiments described are certainly suggestive, but require for their elucidation a determination of the contents of the solution adhering to the sand residue and of the water used. Note that the permanganate was instantly destroyed, and this is certainly not due to pyrite present, as a simple experiment on clean sand in alkaline solution will show practically no effect upon permanganate, as I pointed out in June, 1910, in reply to Mr. Crosse's opinion, which the author quotes.

I quite agree with the author's opinion that it is not necessary to use excess of permanganate, and the results of practical operations at the East Rand and everywhere else show this to be the case, though it was not foreseen.

With reference to the time of contact necessary I would call the author's attention to the remarks made (Vol. XI., page 16), and to the fact that in all cases the great bulk of the cyanide is destroyed very quickly. The complete elimination of HCN (say below 0·001) was found to require in no case more than 24 hours, which was the ultimate time limit (including stope delay) which I then had in mind. This was with artificial solutions, but the presence of coagulants (as in practice) makes a difference. The author's remark that the dilution is very great is not borne out by his figures, which only show an effective dilution of 3:1.

Referring to the second lot of experiments 1 to 4 it must be remarked that hydrocoerulignone is sensitive to one in five millions, and that its use without properly arranged blank experiments may at times prove delusive. It is recommended by Dr. Moir that tolidine should be substituted, and I have found phenolphthaline to be more reliable, especially if a simple blank is compared with the solution to be tested. I have found that the action on a solution of sodium zinc cyanide shows practically no difference from one of a similar strength of KCN, provided it is not acid; and a synthetic solution of 0·05 % or even 0·005 % KCN will show the action practically at once if a little less than the required amount (say, 40 c.c. and 4 c.c. respectively of N/10 permanganate per 100 c.c.) be added. Very rapidly the permanganate is destroyed, as is obvious to the naked eye. If sufficient $AgNO_3$ be first added to combine with all the KCN, the remaining $Zn(CN)_2$ will be found to be very slowly attacked, as is also the case if an excess of zinc chloride be added, though in this case a speedy action is produced by excess of alkali. In any case, however, it would be difficult to explain the author's own stope results if the double cyanide were unacted upon in the short time given.

The author is in error in stating the opinion of Dr. Moir and Mr. Gray upon the permanganate process, which as given on page 152, Vol. XI. of our *Journal* is absolutely favourable, and if he will again refer to page 445, Vol. X., he will withdraw the statement that I have admitted failures in testing working solutions. The evolution of ammonia referred to by the author is curious, and from the standpoint he has set up requires explanation. He takes up the attitude that the 0·0004 KCN in solution entering the stope is all that the permanganate (which anyhow was all destroyed at the sand tip) has to work upon; but the ammonia derivable from this insignificant amount would certainly not be detectable in the air of a stope if we bear in mind the solubility of ammonia in water and its immediate fixation by any acid formed on the stope walls.

The paragraph with reference to the high cost of cyanicides may be dismissed with the remark that the process now in use is not likely to be changed for any cheaper one unless some other advantage is secured. Hypochlorites are possible at half the cost, but the slight disadvantages more than outweigh the saving. But surely there is nothing unnatural or supernatural about $KMnO_4$. The reference to the old sand dump (fairly *dry*, I presume) of original high cyanide content reminds one of the prophecy that the introduction of sand-filling would connote some reduction in cyanide thrown away with residues. The author, however, attaches more belief to the action of ferrous and ferric sulphates in *complete* elimination

of HCN than a study of the paper by Dr. Moir and Mr. Gray (quoted) warrants (Vol. X., page 133). It is to be remarked that a high alkalinity alone as recommended by the author might yield disastrous results when the slope drainage met acid mine water as 1:3 of resume solution strength must be reckoned with, and even then the moderate alkalinity required with permanganate might on the whole show the lesser cost.

The fact must also not be overlooked that it reliance is placed upon the "natural" conversion to ferrocyanide, it is impossible to regulate the proportion between ferrosocteric sulphates and acid in the mine waters or the relative volumes of reacting solutions, though Dr. Moir and Mr. Gray have pointed out that the conditions for a high percentage of conversion are by no means elastic.

In conclusion, though I have traversed the whole of the author's conclusions, I am bound to admit that this is a very skilful attempt to substantiate a certain standpoint, and that even if it fails therein it illustrates the point that on every new mine where it is proposed to use current sand for stope filling the whole of the relevant conditions will repay proper investigation.

On the proposal of Mr. **John Watson**, a hearty vote of thanks was accorded Mr. Cooper for his paper.

IMPROMPTU DISCUSSION ON WASTE ROCK VALUES.

Mr. E. Pam *(Vice-President)*: Mr. Thomas has had to leave the meeting, but will probably return shortly. The real agenda of the meeting is finished, but several societies have recently been very successful in having impromptu discussions on various subjects, and I have been asked by several members to start a discussion on the sorting of rock on the mines.

A very interesting discussion then took place on this subject. After Mr. Pam had given his views, discussion for and against was given by Messrs. White, Toombs, Meyer, Thomas, Dowling, Cellier, and Weston.

The Chairman: Before drawing this meeting to a close, I would like to know your wishes with regard to the publication of this discussion in the *Journal*, or whether you merely wish it to be informal?

Mr. J. E. Thomas *(Vice-President)*: I beg to move that it be not published. This was seconded, and declared carried.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ASSAY OF CRUDE PLATINUM.—"Crude platinum is on the market in the shape of small nuggets, ranging in weight from 100 mgm. down. It may be divided roughly into two portions, the insoluble and the soluble in *aqua regia*. The insoluble includes osmiridium as its only valuable constituent, while the soluble includes platinum, iridium, palladium, rhodium and gold. It is desired to determine each of the above metals with accuracy

This original method is now in use by a large buyer of crude platinum and has been checked up by results obtained in refining the purchased lots of crude.

The Method.—Weigh 2.5 gm. of the sample into a 300 cc. beaker. Add 200 cc. of *aqua regia* (1 part nitric to 3 parts hydrochloric acid, by volume) and place on the water bath for about one hour, then place on a thin piece of asbestos on the hot plate and boil gently for two hours. Remove from the hot plate and allow to settle for 15 minutes. Decant the solution into another 300 cc. beaker and again boil for about an hour to dissolve any finely divided metal, which may have decanted over. Remove from the hot plate and fill up the beaker with water. Stir well, then settle over night. Immediately after decanting the *aqua regia* from the original beaker, add another 200 cc. of *aqua regia* to the residue therein and repeat the warming and boiling of the acid. Finally remove from the plate, fill with water, stir, and let settle over night. Unless the ore is in unusually large pieces, the solution of the soluble minerals will be complete with this treatment, which is planned to take one day.

Decant the clear solutions from both the beakers into a 1,000 cc. volumetric flask, being careful not to allow any of the insoluble to be carried over. Wash the contents of one of the beakers into the other so as to have everything in one beaker and to dilute the acid before filtering. Filter through a 9 cm. paper into the 1,000 cc. flask. Wash the insoluble from the beaker on to the paper and then wash on the paper till the solution which runs through is perfectly colourless.

Determination of Osmiridium. Transfer the filter paper and its contents to a 2½ in. scorifier, add 10 gm. to 15 gm. of test lead, fill the scorifier level full with litharge and fuse in the muffle for half an hour. Allow the scorifier to cool, break it, and remove the lead button. Clean the lead button thoroughly with as little hammering as possible, place it in a 150 cc. beaker and dissolve out the lead with dilute nitric acid (1 volume nitric acid to 3 volumes water). Place on the water bath to assist solution. When the lead has been all dissolved, filter through an ashless filter and wash the residue of osmiridium on the filter till free from lead. Dry the filter, separate the osmiridium from it as completely as possible, and burn the filter paper. Add the osmiridium to the residue from the paper, dry thoroughly on the hot plate and weigh as osmiridium.

The scorification with lead is for the purpose of cleaning the osmiridium from sand, heavy oxides, etc. The osmiridium is collected by the lead and the impurities are slagged off. Some care is required in igniting the residue, as osmiridium will lose osmium if heated to redness in the air. For this reason, the paper is burnt after separating from the osmiridium.

Determination of Platinum.—Fill the 1,000 cc. flask containing the solution of the ore to the mark. Mix thoroughly by pouring into a dry beaker and back into the flask four times. Taking a 25 cc. pipette, which has been carefully standardised against the 1,000 cc. flask, rinse it out with a portion of the solution and then pipette two portions to 75 cc. beakers. Evaporate these portions just to dryness on the water bath. Add to each 5 cc. of dilute hydrochloric acid (1 acid : 9 water) and warm for five or ten minutes. Allow to cool and unless perfectly clear filter through 5½ cm. papers into similar beakers. Wash with water till the filter papers are perfectly colourless. This can be done easily with less than 20 cc. of water.

Make up each portion so that it contains about 25 cc. Add to each 8 gm. of c.p. ammonium chloride and heat on the water bath for one hour with occasional stirring. Remove from the bath and let stand at least three hours, or over night if convenient. Decant the clear solutions through a 9 cm. ashless filter into 150 cc. beakers. Allow all the solution to run through and then wash the precipitates of ammonium chloplatinate, $(NH_4)_2PtCl_6$, onto the filters with 20% solution of ammonium chloride. Allow all the solution to run through and then wash repeatedly around the edge of the papers with small quantities of the ammonium chloride solution, till the filtrate is colourless and the papers are white. Finally rinse around the edge of the papers with alcohol. After all the solution has run through place the filters with their points down into size B Battersea annealing cups. Carefully fold over the edges of the papers so as completely to inclose the precipitate in an envelope of paper. Cover the crucibles and bring to a good red heat by very slowly increasing the temperature. Finally remove the covers from the crucibles till the papers are burnt off. After cooling, transfer the platinum to the pan of the assay balance and weigh (The duplicate should not vary more than 2 mgm. The average is taken as correct.)

The precipitation of $(NH_4)_2Pt(Cl_6$ is usually made with the addition of alcohol to reduce the supposed solubility of the precipitate. I am sure that by the above method I can take 0·0004 gm. of platinum in the given size solution, 25 cc., precipitate it, wash with the usual quantity of wash solution, and get back more than half of the original platinum. The solubility under the conditions given is negligible for this assay.

The solutions are heated on the water bath for two reasons: First, to get all the ammonium chloride rapidly into solution and second, to change the precipitate from an amorphous to a sandy, crystalline precipitate, which is easily filtered and washed.

The precipitate of platinum usually contains iridium as $(NH_4)_2IrCl_6$. Iridium alloyed with platinum in small quantities, less than 10 %, dissolves with the platinum in *aqua regia*. It precipitates with the platinum, but not quite so completely. The traces which pass the precipitation are afterward estimated with rhodium.

Pure $(NH_4)_2PtCl_6$ is a pure yellow. The iridium compound has an intense red colour. If ½ % of iridium is present it gives a decided red tint to the platinum precipitate. The quantity of iridium present, if not more than 5 %, can be judged by the colour of the precipitate quite accurately enough for setting a value on the ore.

There is a slight loss in igniting the platinum precipitate. The loss is lessened by keeping inclosed in the filter paper and by slow heating.

Determination of Gold.—Combine the filtrate from the platinum precipitation, and add 5 gm. of ferrous sulphate in water solution plus HCl. Allow to stand till the gold has settled. Filter and wash on the filter, transfer to an annealing cup and burn off the paper in front of the muffle. When cool, wrap in lead foil and cupel as usual for gold.

Determination of Palladium.—Add to the filtrate from the gold about one-fifth its volume of hydrochloric acid. Add a 10 % solution of potassium iodide five drops at a time till no further precipitation of black palladous iodide (PdI_2) takes place. Heat on the hot plate nearly to boiling. Cool, filter and wash with dilute hydrochloric acid (1 acid : 4 water). Wash free from iron. Transfer to a porcelain crucible and ignite in the muffle at a good red heat. Cool, moisten with a few drops of dilute formic acid to reduce oxide and dry thoroughly on the hot plate. Transfer to the pan of the assay balance and weigh the palladium.

Textbooks usually describe palladous iodide as easily soluble in excess of KI. This is true, but the presence of a large amount of HCl probably prevents any excess of KI being present, hydriodic acid and potassium chloride being formed. This precipitation is complete. The strong hydrochloric acid solution also prevents the precipitation of lead and copper iodides if only a slight excess of KI is used.

Dilute formic acid reduces palladium oxides to metal.

Determination of Rhodium. To the filtrate from PdI_2 add c.p. zinc till the solution is colourless, heating the solution while doing so. Allow all zinc to dissolve and the precipitate to settle. Decant the solution and wash the precipitate three times by decantation with 100 cc. portions of hot water. Add 25 cc. of dilute nitric acid (1 acid : 3 water) and place on top of the water bath for 10 minutes. Filter through an ashless filter, wash two or three times with water, ignite, reduce in hydrogen, and weigh as rhodium plus traces of iridium.

Zinc reduces all the platinum metals from solution. In this case only rhodium and traces of iridium are present. The nitric acid dissolves any base metals which may be present, but as freshly precipitated rhodium is slowly soluble in nitric acid, the treatment must not be too much prolonged.

A representative assay of crude metal would be the following: Osmiridium, 20 fine; Pt (containing 3% Ir), 830; Pd, 5; Au, 30; Rh, 20 fine.

The strongly coloured solutions furnished by all of the platinum metals, platinum giving the least colour, form a good guide for completeness of washing, zinc precipitation, etc.

The following outline of the method used in the laboratory of Ledoux & Co., for the separation of gold, platinum and palladium was dictated by A. M. Smoot, and should be considered as supplementary to his 'Suggestions on the Platinum Palladium Assay,' in the journal of Dec. 20, 1913. The subject is of great present interest, as practically every electrolytic copper refiner is now turning out crude palladium, to be sold on assay.—*Editor.*

In a solution containing gold, platinum and palladium, determine the gold by precipitation with oxalic acid. For this precipitation the solution should have been evaporated with hydrochloric acid to get rid of nitric acid, should be only slightly acid with HCl, and should be hot. After adding the oxalic acid, the solution should be allowed to simmer for about one-half hour, then allowed to stand over night. Oxalic acid should be present in excess.

Filter off the gold. It will be necessary to dissolve the gold and reprecipitate if any great amount of palladium is present, adding the second filtrate to the first.

Throw down the metals in the filtrate with sulphuretted hydrogen and ignite the sulphides cautiously. Dissolve the ignited sulphides in *aqua regia* and evaporate three times with hydrochloric acid. This not only gets rid of the nitric acid, but reduces all the Pd to $PdCl_2$. If a little black residue remains, filter it off, ignite, treat with a little strong formic acid to reduce any palladium oxides to palladium, dissolve in *aqua regia* and treat to remove nitric acid, then add to main solution.

Add saturated solution of ammonium chloride. Pt comes down as $(NH_4)_2PtCl_6$. Allow to stand over night and filter off. If much palladium is present, redissolve and reprecipitate. If only a little palladium is present, dissolve the precipitate in hot water, add a little sulphuric acid, and pass in sulphuretted hydrogen. Filter off, ignite and weigh as platinum.

The palladium in the filtrate from the platinum may be determined directly by the method of Wunder and Thuringer (*Zeit. für anorg. Chem.*, Vol. 52, p. 101). To the solution made up to about 150 cc., add 1 gm. dimethylglyoxime in 5 cc. hydrochloric acid. Dilute to 300 cc., allow to stand one-half hour on water bath, and then settle over night. Filter on a tared porcelain gooch, wash with hot water, then with alcohol and weigh.

The precipitate is $(C_4H_7N_2O_2)_2Pd$, containing 31·686 palladium. The separation seems to be quantitative from all metals except nickel, and this will have been eliminated, either in a sulphuretted-hydrogen precipitation, or in cupellation. If a large excess of silver be present, there is little loss in the cupellation of any of the platinum metals." M. SCHWEITZER. *Engineering and Mining Journal*, June 20, 1914, p. 1249. (G. H. S.)

PLATINUM ASSAY. "I would direct attention to certain difficulties in the three methods published by you in tabular form on p. 81 of the May 16 issue." The first method requires the use of a reagent, HIO_3, which is not ordinarily carried by assay offices. I may also add that perhaps the limits of accuracy of this separation when considerable relative quantities of Au are to be separated from small amounts of Pt have not yet been sufficiently worked out. In the second method there are two points where great care and attention to details are required to secure anything like a clean separation. To dissolve the Ag in HNO_3 and to leave the Pt undissolved requires extremely close work, which cannot ordinarily be given in a commercial assay office, while it is practically impossible so carefully to adjust the strength of acid and temperature as completely to dissolve the Au in *aqua regia* and leave the Pt unattacked. As to the third method, some tests made in the New York assay office have shown that Cd does not always prevent the solution of some of the Pt in assay buttons and this separation cannot be relied on." F. P. DEWEY. *Mining and Scientific Press*, July 4, 1914, p. 20. (G. H. S.)

SLAETA'S LAKE POTASH. — "The deposit consists of a body of crystals, covering approximately 20 square miles, and averaging 70 ft. in thickness. These crystals are permeated by saturated brine, the average level of which is 1 in. below the surface of the crystals. The brine is substantially uniform in composition throughout, while the crystals vary considerably in different portions of the deposit and at different depths of the same well. More than 200 wells have been drilled and the deposit has been thoroughly explored. The raw material which will be used is the brine. The mining costs represent the cost of pumping the brine. This cost is ridiculously low, as the elevation of the works above the surface of the crystal body is only about 75 ft.; and to turn out the capacity contemplated in the large-scale works, it will be necessary to pump only 2,000,000 gal. per day.

The principal salts in the brine are sodium carbonate, potassium chloride, sodium borate, sodium chloride and sodium sulphate. The salts of value are potash, borax and soda ash, the potash being by far the most valuable. At present no value is attached to the salt or sulphate, but it is hoped some day to find a market for them, or to convert them into other and more valuable products." L. H. EDDY, *Engineering and Mining Journal*, July 11, 1914. (G. H. S.)

CHEMICAL COMPOSITION OF RAIN IN THE UNION OF SOUTH AFRICA. "Monthly analyses of rain water collected at (1) Grahamstown, (2) Kokstad, (3) Bloemfontein, (4) Durban, and (5) Cedara, Natal, 1910-12, and for shorter periods at Retreat, Douglas, Cradock, Johannesburg, Potchefstroom, Modderpoort and Wremen.

The yearly averages for the five first-mentioned places are as follows:—

		N per million.		N per acre.			Chlorine	
		Rain-fall. inches.	As Am. inches.	As Nit. inches.	As Ammonia lbs.	As Nitrates lbs.	Total lbs.	Per million. lbs.	Per acre lbs.
1.	1911-12	24·37	0·171	0·152	0·944	0·731	1·675	4·26	23·5
2.	1912	26·54	0·156	0·112	1·118	0·670	1·788	—	—
3.	1910-11	27·87	0·382	0·255	3·658	1·620	5·278	1·06	6·7
	1911-12	15·49	1·389	0·388	4·870	1·363	6·233	0·66	2·3
4.	1911	42·34	0·381	0·129	3·651	1·234	4·885	0·39	6·2
	1912	31·07	0·353	0·178	3·906	1·249	5·155	10·91	70·4
5.	1912	26·68	0·780	0·143	4·710	0·865	5·575	2·68	16·2

Comparing summer and winter rain, it is shown that the former brings down more nitrogen in both forms than the latter.

Sulphates were estimated in a few samples of rain at Bloemfontein (SO_3 = 2·10–6·49) and at Durban (SO_3 = 0·50—5·80) per million. At the latter place the sulphates brought down during the eight months in which most of the rain fell amounted to 18·9 lb. per acre (as SO_3)." C. F. JURITZ, *S. A. Journ. Science*, 1914, *10*, p. 170. —*Journ. Chem. Society Abstracts*, July, 1914, *1*, p. 916. (J. A. W.)

TITANITE A NEW EXPLOSIVE. "A new explosive has been invented and patented by Norbert Ceipek, of Vienna (Brit. pat. 9713 and 13,549 of 1911). It is essentially an ammonium-nitrate explosive to which turmeric powder is added, for which many advantages are claimed. The turmeric or curcuma root is nitrated by boiling in water to extract colouring substances, drying, grinding, treating with a mixture of sulphuric and nitric acids, washing and drying the product of the reaction. Carbonised turmeric powder may be similarly nitrated. Ammonium-nitrate explosives usually contain carbonaceous material, and carbonised turmeric powder and carbonised sandalwood are recommended as excellent for this purpose, the sandalwood hastening the speed of combustion. As a formula for an explosive com-

* See also this *Journal* Vol. XIV., Feb. 1914, p. 7.

pounded along these lines, the following is suggested: Ammonium nitrate, 88 % ; carbonised turmeric powder, 1·05 % ; carbonised sandalwood, 0·95 % ; nitrated turmeric powder, 10 % .

It has been found, however, that the carbonised material is unnecessary when the nitrated turmeric powder is used and a simpler formula offered is : 80 to 90 parts of ammonium nitrate and 20 to 10 parts of the nitrated turmeric powder, carefully dried, intimately mixed and finely ground.

The addition of trinitrotoluene and gelatine, the latter consisting of collodion cotton and dinitrotoluene, results in an explosive of greater strength. A formula for such a compound would be : Ammonium nitrate, 82 parts ; nitrated turmeric powder, 4 parts; trinitrotoluene, 10 parts ; gelatine, 4 parts ; (the latter to consist of 0·17 part of collodion cotton and 3·83 parts of dinitrotoluene.)

It is claimed that these explosives are stable when stored, the hygroscopic quality of the ammonium nitrate being overcome ; that they are entirely safe, being insensitive to blows, shocks and fire, and exploding only from an initial explosion as by a powerful detonator ; that they are powerful in their effect, and that they yield only a small amount of noxious gas. As a matter of fact, these claims have been pretty well borne out by trial at one of the largest mines in this country. The difficulty in the way of the general introduction of the explosive, which is called titanite, seems to be that ammonium nitrate cannot now be obtained quite cheap enough to enable the titanite to compete in price with the other high explosives in common use.—*Engineering and Mining Journal*, July 18, 1914, p. 116. (G. H. S.)

INTERNATIONAL ATOMIC WEIGHT TABLE, 1915.—

Aluminium	Al	27·1
Antimony	Sb	120·2
Argon	A	39·88
Arsenic	As	74·96
Barium	Ba	137·37
Bismuth	Bi	208·0
Boron	B	11·0
Bromine	Br	79·92
Cadmium	Cd	112·40
Caesium	Cs	132·81
Calcium	Ca	40·07
Carbon	C	12·00
Cerium	Ce	140·25
Chlorine	Cl	35·46
Chromium	Cr	52·0
Cobalt	Co	58·97
Columbium	Cb	93·5
Copper	Cu	63·57
Dysprosium	Dy	162·5
Erbium	Er	167·7
Europium	Eu	152·0
Fluorine	F	19·0
Gadolinium	Gd	157·3
Gallium	Ga	69·9
Germanium	Ge	72·5
Glucinum	Gl	9·1
Gold	Au	197·2
Helium	He	3·99
Holmium	Ho	163·5
Hydrogen	H	1·008
Indium	In	114·8
Iodine	I	126·92
Iridium	Ir	193·1
Iron	Fe	55·84
Krypton	Kr	82·92
Lanthanum	La	139·0
Lead	Pb	207·10
Lithium	Li	6·94
Lutecium	Lu	174·0
Magnesium	Mg	24·32
Manganese	Mn	54·93
Mercury	Hg	200·6
Molybdenum	Mo	96·0
Neodymium	Nd	144·3
Neon	Ne	20·2
Nickel	Ni	58·68
Niton (radium emanation)	Nt	222·4
Nitrogen	N	14·01
Osmium	Os	190·9
Oxygen	O	16·00
Palladium	Pd	106·7
Phosphorus	P	31·04
Platinum	Pt	195·2
Potassium	K	39·10
Praseodymium	Pr	140·6
Radium	Ra	226·4
Rhodium	Rh	102·9
Rubidium	Rb	85·45
Ruthenium	Ru	101·7
Samarium	Sa	150·4
Scandium	Sc	44·1
Selenium	Se	79·2
Silicon	Si	28·3
Silver	Ag	107·88
Sodium	Na	23·00
Strontium	Sr	87·63
Sulphur	S	32·07
Tantalum	Ta	181·5
Tellurium	Te	127·5
Terbium	Tb	159·2
Thallium	Tl	204·0
Thorium	Th	232·4
Thulium	Tm	168·5
Tin	Sn	119·0
Titanium	Ti	48·1
Tungsten	W	184·0
Uranium	U	238·5
Vanadium	V	51·0
Xenon	Xe	130·2
Ytterbium (Neoytterbium)	Yb	172·0
Yttrium	Yt	89·0
Zinc	Zn	65·37
Zirconium	Zr	90·6

—*Journal of the American Chemical Society*, xxxv., 8, August, 1914, p. 1587. (J. A. W.)

A GENERAL METHOD FOR ESTIMATING ARSENIC IN ORES.—"After disintegrating 1—5 gm. sample by employing tartaric acid, HNO_3 and H_2SO_4, reducing with Na_2SO_3 and precipitating with H_2S in presence of HCl, dissolve As_2S_3 in NH_4OH and H_2O_2, precipitate with magnesia mixture, filter through a weighed alundum filtering crucible and weigh as $Mg_2As_2O_7$."—B. G. DUNSTAN, *Chem. Analyst*, 2, 7-8.—*Chemical Abstracts*, July 10, 1914, p. 2325. (J. C.)

PURIFICATION OF COAL GAS BY HEAT.—"Plant for passing purified coal-gas over finely divided nickel at 430° C. so as to convert carbon bisulphide into hydrogen sulphide, which is then removed, has been installed capable of treating 15,000,000 cub. ft. of gas per day. The plant consists of 5 units, each comprising a gas-producer and combustion chamber, two heating chambers containing vertical reaction tubes packed with the catalytic material, and two heat-interchangers for preheating the gas to 400° C. The weight of nickel used is 1,324 lb., and the plant consumes 5½ tons of coke per day. The average quantity of sulphur in the gas was reduced from 35·07 gr. to 7·82 gr. per 100 cub. ft. or by 77·7%. The

greatest reduction was from 63·78 to 10·34 (83·8 %), and the least, from 19·21 to 5·34 (72·2 %). The quality of the gas was as follows:

	Before Treatment	After Treatment
Illuminating power (candles)	14·7	14·7
Calorific power (gross B. Th. U.)	590	594·4
Naphthalene (grains per 100 cb. ft.	5·54	4·71
Hydrocyanic acid ,, ,,	22·49	18·75
Analysis (per cent. by volume):		
Carbon dioxide	1·40	1·45
Hydrocarbons	3·55	3·63
Oxygen	0·33	0·06
Carbon monoxide	8·16	8·03
Methane	26·85	27·26
Hydrogen	54·19	54·25
Nitrogen	5·48	5·32

One of the retort houses was worked at low temperature so as to produce gas of average candle-power 20·54; after treatment the candle power of this gas was 20·46. The capital cost of the plant was about £1,500 per million cub. ft. per day and the working costs were 0·293d. per 1,000 cub. ft., including repairs, maintenance, interest, and depreciation. Plant for treating 1½ million cub. ft. of gas per day has been worked for two years. The wasting of the reaction tubes is very slight and the efficiency of the catalyser is unimpaired." C. C. CARPENTER, *J. Gas Lighting*, 1914, *126*, 928. — *Journal of the Society of Chemical Industry*, xxxiii., 13, July 15, 1914, p. 682. (J. A. W.)

SOME CAUSES OF ERROR IN IODOMETRY.—"In titrating I solutions with $Na_2S_2O_3$ solution the presence of alkalies seriously interferes. To 10 cc. of 0·1 *N* I solution 10 cc. of a saturated solution of an alkaline salt were added and the solution titrated with 0·1 *N* $Na_2S_2O_3$. With KHCO$_3$ solution but 6·5 cc. of the $Na_2S_2O_3$ solution were required, or an error of 35 %; with NaHCO$_3$ 8·2 cc. of $Na_2S_2O_3$ were required, or an error of 18 %; $Na_2B_4O_7$ solution 6·4 cc. of $Na_2S_2O_3$ solution, or an error of 36 %; Na HPO$_4$ 9·3 cc. of $Na_2S_2O_3$, or an error of 7 %; $(NH_4)_2CO_3$ (not saturated) 4·5 cc. of $Na_2S_2O_3$, or an error of 55 %. The inaccuracy holds true if the 0·1 *N* I solution be added to the mixture of $Na_2S_2O_3$ and alkali, although the error is not so great. Temperature is an important factor. In an experiment with $Na_2B_4O_7$ at 13·4°, 8·7 cc. of 0·1 *N* $Na_2S_2O_3$ were used while at 38·40° but 5·9 cc. were required. The rapidity of the titration influences the accuracy; in general, the slower the titration the greater the accuracy. The dilution of the solution does not seem to exert much influence. In titrating alkaline I solutions the alkalinity should first be determined by adding to a measured volume just sufficient Na$_2$S$_2$O$_3$ solution to remove the colour and titrating with 0·1 *N* H_2SO_4, using methyl orange. In a second measured volume an equal number of cc. of 0·1 *N* H_2SO_4 are added and the titration completed with 0·1 *N* $Na_2S_2O_3$." A. LECLERC, *J. Pharm. Chim.*, 9, 341-5. *Chemical Abstracts*, July 20, 1914, p. 2539. (J. G.)

VOLUMETRIC DETERMINATION OF FERRICYANIDES IN PRESENCE OF FERRIC IRON AND CYANIDES. "To suppress the liberation of iodine by ferric iron, potassium fluoride is added to the feebly acid solution of the ferricyanide in the proportion of 6 gm. atoms of fluorine to 1 of iron. Potassium iodide and zinc sulphate are then added, and the liberated iodine titrated with thiosulphate. Any hydrocyanic acid present is first removed by passing a current of carbon dioxide free from reducing substances through the liquid warmed to 50° C. and made acid with sulphuric acid." E. MULLER and F. SEIDEL, *Z. anal. Chem.*, 1914, *53*, 416-420.—*Journal of the Society of Chemical Industry*, xxxiii., 13, July 15, 1914, p. 680. (J. A. W.)

METALLURGY.

PROGRESS IN THE METALLURGY OF ZINC, COPPER, LEAD, GOLD AND SILVER IN 1913.—"Under this title Dr. Nagel, of Berlin, publishes in *Metall und Erz* of April 8th last a most comprehensive compendium—compiled partly from private information supplied to him by prominent metallurgical works, and partly from articles and notes published in the world's mining and metallurgical press—of the innovations and improvements patented and experimentally or definitely introduced in metallurgical practice during 1913. Of the more important ones the main features are more or less fully described, while of others merely the numbers of the patents and the names of the patentees are mentioned. The following abstract contains a summary of some of the more important features:

Zinc.—Considerable activity has prevailed in all branches of zinc reduction, and although no epoch-making changes have been introduced, great improvements have been accomplished in a number of old processes. As regards ore roasting, special mention is due to the Spirlet furnace, introduced by the Erzrostgesellschaft of Cologne, and definitely adopted with satisfactory results by the Aktiengesellschaft fur Blei-und-Zinkfabrikation of Stolberg and Aix-la Chapelle. It consists of several circular roasting chambers placed one above another, and mutually connected by alternating openings in the centre and along the circumference. Being of moderate dimensions, they treat from 4 to 6 tons per twenty-four hours according to the quality of the crude ore. The desulphurisation is highly satisfactory, and the coal consumption is low, averaging about 6 % of the crude ore. Attendance is simple and cheap, two men per day and two per night being sufficient for a three-furnace installation.

Another mechanical roasting furnace which marks a substantial improvement on the old hand-worked types was designed by the Metallbank and Metallurgische Gesellschaft of Frankfurt on Maine, and erected at the Bensberg-Gladbach works. The ore is fed into the roasting chambers by automatically acting mechanical appliances, and also drops automatically from the furnace. The roasting hearths, similar to the well-known Hasenclever type, are arranged in four tiers, one above another. The roasting blast is introduced at the lowest tier, encountering and passing through the ores as they automatically move from one tier to the next. The furnace is heated by gas, the flues being so arranged that the fire gases cannot come into contact with the roasting gases. The capacity of the plant averages about 10 tons roasted blende per day, the desulphurisation being satisfactory. Among the minor advantages claimed for this furnace is that the roasted material leaves it of a uniform size, so that any further grinding, screening and repeated roasting of lumps is avoided.

In the muffle process of zinc reduction progress has been confined to innovations in the introduction of temperature, partly by improved generators and recuperators, and partly by a better adjustment of the heating and combustion gases; but successful efforts of rather considerable importance have been made to improve the treatment of zinciferous residues. For instance, at the Hans-Heinrich works at Langelsheim, the erection of which was completed in the course of the year, a waterjacket shaft furnace of ordinary type has been adopted for smelting slag averaging about 14% zinc, 1–2% Cu, and 1·6 to 3·5% Pb, with coke as flux. The resulting copper-lead regulus, which would also hold any precious metal contents of the slag, is treated in the usual way, while all the slag containing zinc oxide which overflows from the forehearth is conveyed direct into reverberatory furnaces, where the zinc is eliminated by lime and carbon at a temperature of about 1,500 deg. C. The results are stated to be satisfactory.

As regards electro-thermic zinc recovery, it cannot yet be said to have reached a stage where it can be regarded as a serious competitor of the old muffle process. Before that could come to pass further great improvements to prevent loss of metal and excess of zinc dust generation, as well as in condensation and power consumption, would have to be made. Among the various electro-thermic processes adopted in Europe the Côte-Pierron method is stated to give satisfactory results at the Ugine works in Savoy, where its employment is greatly facilitated by exceptionally favourable local conditions. At Trollhattan, in Sweden, and Sarpsborg, in Norway, several thousand tons of spelter are yearly produced by the electro-thermic process, and although the work cannot yet be considered economically satisfactory, it is stated that the results obtained in August, 1913, and later have been so much better than in previous times that the outlook for the future is regarded as most satisfactory. In America particular attention has been devoted to direct smelting in the electric furnace of simple as well as of complex zinc ores without preliminary or preparatory reduction processes; but, so far, all attempts in this direction have had more or less disappointing results. On the whole, however, it is considered that the definite solution of the problem is now well in sight. The same remark applies to the efforts made in America to find a suitable treatment for the concentrates obtained by flotation processes, which, owing to their fine granulation, oppose great difficulties to present roasting methods. Electrolytic zinc recovery, too, has been the subject of very thorough investigations and experiments in America, notably at the Bully Hill works, California. The problem of obtaining from zinc sulphate solutions a dense, adhesive zinc cathode has apparently been overcome, and, altogether, the process seems to be technically practicable, but economic obstacles prevent its adoption, owing mainly to the initial cost of an electrolytic plant, which, compared with a muffle furnace installation, is simply enormous.

Of hydro-metallurgical methods the bi-sulphite and the ammonia processes have attracted the greatest interest during 1913.

Copper.—Copper recovery by dry reduction has now been so highly developed that further fundamental changes can scarcely be expected, and improvements are confined to working details. Nearly all that has been effected in that respect during the year under review has to be credited to America. For ore-crushing the Hardinge mill has come to the fore, while slime treatment has been improved by the Dorr thickening apparatus and a new type of round tables. The old types usually had a diameter of from 20 to 30 ft. and a capacity of about 6 tons per twenty-four hours. The latter has now been increased to as much as 120 tons by the construction of round tables in tiers one over another. Tables with twenty tiers have been in common use, and the construction of even more efficient ones with thirty tiers is in contemplation. Several minor improvements have been effected in roasting processes, and the Dwight-Lloyd sinter apparatus has gained in favour. The erection of a rotary kiln for sintering flue-dust by the United States Metal Refining Company, of Chrome, N.Y., is also worth mention. Its size is 60 by 6 ft., and it is stated to give an excellent product at a cost of 80 cents. per ton, the daily capacity being 60 tons.

As regards reverberatory furnaces, it has now been accepted as an axiom that economically the most favourable size is 19 by 100 ft., or at the most 110 ft. At the Copper Cliff works of the Canadian Copper Company the use of coal-dust fuel has proved a success.

The main interest in the metallurgy of copper has centred during 1913 on the recovery of the metal from poor ores by leaching. Most American copper producers are busily engaged in the solution of this question, and the results so far obtained point to the probability that we are on the eve of most remarkable developments in this direction. Extensive trial operations have shown the possibility of reducing to a minimum the loss of metal by waste and of profitably treating cupriferous material which hitherto has been rejected as unpayable. Sulphuric acid is generally regarded as the best leaching agent, and the process of getting the copper into solution does not appear to present any difficulties. Porphyritic ores especially require but little sulphuric acid, because, apart from their copper contents and a little iron and aluminium, none of their other component parts go into solution. In most cases the copper extraction exceeds 90%. For precipitating the copper either electrolysis or iron is used. Cappelin, Smith, van Arsdale, Canby, Channing, and Fields recommend the former. Improved anodes of fused magnetite, and the introduction of dioxide of sulphur into the electrolyte, greatly reduce the electric energy requirements without causing any extra cost worth mentioning. Thus a new and highly promising sphere of utility has been opened for this method of precipitation. Crossdale, Laish, and Mathewson adhere to the old method of precipitating by means of iron, the efficiency and general usefulness of which have been greatly heightened by the adaptation of spongy iron to the purpose. This precipitating agent is produced by roasting iron pyrite and then recovering the spongy iron by using coal as reducing agent. Crossdale asserts that the simple process can be completed at a cost of no more than 8*d* per ton of iron sponge. Besides removing the difficulty of securing a sufficiency of iron scrap, this spongy iron is an ideal precipitating agent, offering a large superficies and acting very quickly. If very poor iron pyrite is used for its production, it has, of course, previously to roasting, to be separated from the gangue by the magnetic or some similar process. In addition to these improvements, certain mechanical appliances, the usefulness of which has been demonstrated in the cyaniding of gold and silver, have been introduced in the hydro-

metallurgy of copper. In Chuquicamata the leaching of copper is carried out in tanks holding 10,000 tons. Their size is 16 ft. by 100 ft. by 150 ft.

Lead. So far as European, and more especially German, lead works are concerned, the year under review has been noticeable for the quasi-general introduction of the Dwight Lloyd roasting and sintering process, in some cases with certain modifications by Schlippenbach. At the Binsfeldhammer works the crude material is subjected to a preliminary roasting, after the Huntingdon-Heberlein method, and then conveyed by an aerial tram to the automatic plant for further treatment. In addition to treating pyritic ores the Dwight-Lloyd process has also been adopted for the agglomeration of oxidised material (mullle residues, fine iron ores, &c.).

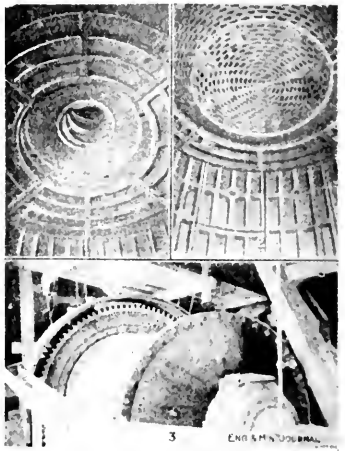
Particular attention has been paid to the problem of the recovery of fluedust, &c. Some slight improvements have been introduced in the old dry methods of precipitation in channel or chamber systems, and a new wet method has been adopted with good results. Catching the dust in sack filters has also proved efficient, but the process is rather expensive, owing to the necessity of providing artificial draught and excessive wear and tear of the sacks.

For the purpose of desilvering lead the Consolidated Mining & Smelting Company, of Trail, B.C., is now erecting an electrolytic refining plant after the Betts method. Formerly all the antimony was practically lost in the anode slime, but by the new method the slime is boiled in sulphide of sodium, when from 80 to 90 of the antimony and 50 of the arsenic go into solution. This solution is filtered off and electrolysed in iron tanks with anodes of lead-sheet and cathodes of steel-sheet, the density of the current being 8 amp. per sq. ft. of cathode surface. The antimony, which has to be removed from time to time, contains 2 arsenic, which is eliminated by smelting with alkaline fluxes when the cathode precipitate is formed into blocks.

Gold and Silver. Though some substantial improvements have been introduced during last year they are not of epoch making importance, but rather apply to details of working methods. The cyanide process still occupies the leading position among reduction methods. The most important improvement added to it last year was the use of aluminium dust as precipitating agent. The innovation acquires additional importance owing to the general applicability of its underlying principle *i.e.*, the fact that aluminium does not combine with cyanide, and that after the elimination of the metals from the cyanide solution the cyanide can be used for further solutions. This points the way to a practical process by which ores containing soluble base metals might be profitably leached with cyanide. There are large quantities of complex ores containing so much copper that their gold and silver contents cannot under existing circumstances be treated with cyanide owing to excessive consumption of that expensive chemical. But if the cyanide used for dissolving these base metals can be reintroduced without appreciable loss into the solution the whole aspect of the matter is changed, and in order to extract the precious metals on a profitable basis it would only be required to find a method by which the several metals might be separately precipitated from the solution."—Dr. Nagel, *Metall und Erz* (original) from *Mining Journal*, June 13th, 20th, 27th and July 3rd, 1914, p. 553, 587, 601. (H. R. A.

A Short Tube Mill of Large Diameter.—"In accordance with the present-day tendency to modify the tube mills in use, both in manufacture and in practice, it is interesting to note a mill of large diameter and short length which has been installed by the Butte & Superior Copper Co., Butte, Mont. The mill is 7 ft. in diameter by 10 ft. long overall. The gear is 14 in. wide on the face.

The feed scoop is of the three-cup spiral type, so that with the mill operating at 22 r.p.m., it is fed oftener than once each second. The mill is lined with a Forbes type liner redesigned to suit local conditions



Short Tube Mill.

1. Inside of feed end. 2. Inside of discharge end. 3 Outside of mill.

and will use No. 5 Danish pebbles. The mill has developed an average capacity of approximately 300 dry tons of ore per day, of which quantity approximately 75 tons are returned to the mill as a circulating load.

A second mill, almost identical with this one, is now being installed. The type has proved economical and satisfactory. The accompanying photographs show a view of the outside of the mill, and one each of the inside at the feed end and the discharge end."

Engineering and Mining Journal, June 6, 1914, p. 1149. (G. H. S.)

Saving Centrifugal Pump Shafts.—"When centrifugal pumps are operated on sandy pulp, the usual result is the cutting out of the impeller shafts at the point where the packing bears upon them. One of the most popular methods of reducing this wear factor and, in fact, about the only one which has so far proved effective, is to feed water or solution constantly into the gland, thus washing back the sand into the pump.

At the plant of the Deloro Mining & Reduction Co., at Deloro, Ont., this procedure is impossible, as the extra liquid added would soon run the vat over. The difficulty is surmounted by using a spiral metallic

packing in the gland and feeding hard grease into it by constant pressure. The feeding arrangement is so adjusted that the pressure is just sufficient to keep any solids from entering the gland. Centrifugal pumps so equipped use one shaft for about nine months, whereas before the idea was conceived they would cut out in a few days."—*Engineering and Mining Journal*, June 6, 1914, p. 1150. (G. H. S.)

PRODUCTION OF STEEL DIRECT FROM THE ORE.—"The steel produced in primitive furnaces for Damascus and Toledo blades, practically direct from the ores, although of uneven composition, possessed remarkable qualities of strength, endurance and toughness difficult even nowadays to equal.

The authors have made tests of direct reduction by the electric furnace, and have come to the conclusion that under certain satisfactory conditions the economic production of steel from the ore in one operation is a practical possibility. Among other advantages it is stated that titaniferous and other refractory iron ores can be treated and that the steel produced is freer from impurities."—ERNEST HUMBERT and AXEL HETHEY, B.Sc., from paper before Iron and Steel Institute.—*Mining Journal*, June 13, 1914, pp. 553-554. (H. R. A.)

THE DESULPHURISING PROCESS AT THE NIPISSING MILL, ONTARIO.—"To James Johnson and his coworkers must be given due credit for working out a successful cyanide treatment for the cobalt ores, in which there is such a varying amount of complex minerals. This led to the discovery during the period of experimental work, of what is now known as a 'Wet Desulphurising' process. Briefly explained the preliminary desulphurising treatment breaks up the refractory silver minerals when the slime pulp is brought into contact with aluminium in a caustic soda solution, the silver being left in a spongy, metallic state readily amenable to cyanide treatment. The desulphurising is accomplished by passing the slime pulp through a revolving tube mill, in which there are a number of aluminium ingots. A further treatment is given in a vat lined with aluminium plates, in which vat the pulp is slowly agitated.

The practical effects of this preliminary treatment on the ores has resulted in an equal, and frequently a better, extraction being obtained by 48 hours cyanide treatment, than was otherwise obtained in 120 hours with no desulphurising. In some of the ores which contained a greater proportion of refractory silver in minerals, improved extraction of from 1 oz. to 4 oz. per ton is obtained by desulphurisation and cyaniding, than with a cyanide treatment and no desulphurising.

A tube mill, 25 ft. long and 4 ft. in diameter, and lined with 2 silex blocks 2 in. thick, is used for the first stage of the desulphurising treatment. This mill makes 10 r.p.m. and carries a load of about 4,000 lb. of aluminium ingots, cut into cubes of about 1½ in. to 2 in. The slime pulp is fed through this mill at the rate of 14 tons of dry slimes per hour, diluted with 21 tons of caustic soda solution, making the ratio 1·5 caustic soda solution to 1 dry slime. The pulp then passes into the alkali stock pulp vat, 34 ft. diameter by 13 ft. deep, which is arranged with mechanical agitation and lined around the side with aluminium plates. The pulp is agitated in this vat from 24 to 36 hours, or until it is gradually drawn off in about 40 ton charges to the dewatering filter box. On account of keeping the mill cyanide solutions in balance, it is necessary to eliminate as much as possible of the crushing caustic soda solution from the slimes before they are transferred into the cyanide vats for treatment, and this is done by a Butters' filter plant equipped with 60 leaves, which are capable of dewatering 270 tons of slime per day. The cake when discharged carries 26 alkali solution as moisture. The slime after dewatering flow into a pulping vat, 16 ft. 5 ft. deep, where they first come into contact with cyanide solution—being diluted with about 2 parts of cyanide solution to 1 of slime. A centrifugal pump of 4 in. diameter is used for transferring the pulp into the cyanide treatment vats."—*Mining and Engineering World*, June 6, 1914, p. 1070. (J. C.)

MAGNETITE ANODES.—"We have several times mentioned the new development of magnetite electrodes, unaffected by oxygen, chlorine, or SO_4 ions.

The process of manufacture of these anodes from fused magnetite is kept secret, but the following is taken from H. Specketer's U. S. pat. 931,513. This patent is assigned to the Griesheim works:

I have found that when Fe_3O_4 is melted, oxygen escapes and Fe_2O_3, together with a small proportion of FeO are formed. The presence of this FeO is objectionable for two reasons—first, it causes the electrodes to break on the setting of the melt for the reason that the protoxide does not crystallize uniformly with the magnetic oxide, second, the product will not resist chlorine or anodic actions during electrolysis, and is dissolved by them so that after a certain time the electrodes become ineffective. To obviate these defects and to convert the FeO, formed during the melting of the Fe_3O_4 as aforesaid into Fe_2O_3 I add to the mixture, while in a molten state, a small quantity of unmelted, finely pulverized Fe_2O_3, which unites with the FeO to form Fe_3O_4, so that the entire mixture will be of the same composition. The proportion of pulverized iron oxide must be at least sufficient to convert the whole of the existing protoxide into magnetic iron oxide, but an excess of the Fe_2O_3 is not objectionable because it dissolves in the melt in a uniform manner and crystallizes homogeneously. The oxide of iron (Fe_2O_3) added to the molten mixture being in an unmelted condition, and such addition preferably occurring when the mixture is removed from the furnace, is not heated sufficiently to form the objectionable FeO. In lieu of iron oxide other oxides which combine with ferrous oxide (FeO) to form a compound which dissolves in magnetic oxide (Fe_3O_4) to form a homogeneous crystallizing melt, such, for instance, as sesquioxide of chromium (Cr_2O_3), which combines with FeO to form iron chromite ($FeO.Cr_2O_3$), or sesquioxide of manganese (Mn_2O_3), which combines to form iron manganite ($FeO.Mn_2O_3$), may be used. The homogeneous crystallizing melt thus produced is poured into moulds, closed at the bottom.

The mass solidifies gradually from the outside so that a rigid layer is first formed in the exterior, which layer gradually increases in thickness toward the centre of the mass. When such layer has reached the desired thickness, the remainder of the fluid contents of the mould is returned to the melting vessel to be used for a new charge. The mould is then opened and the electrode, which is closed at its lower end, is removed and allowed to cool slowly. The cooled electrode may for the purpose of increasing its conductivity be provided on its inside with a coating of metal applied by electro deposition or otherwise, or a bar or rod of good conducting metal, such, for instance, as copper, nickel, or the

like, may be introduced into the central part of the mass, while it is still liquid, so that the bar or rod will form an integral part of the electrode when the mass is completely solidified.

In practice, however, the weight of a large anode made with a solid bar forming the core would be too great, so the anodes are cast hollow, as described in the patent, and the interior of the anode is electroplated heavily, and copper contact strips cemented on. The anode walls are about ¼ in. thick. These contact strips may be seen coming from the top of the anodes in the illustration. This plating must be heavy and uniform throughout the length of the anode, otherwise too great a proportion of the current gets out through the top of the anode, giving excessive current density at the top of the cathode. The anode weighs 35 lb., the joint will stand a pull of about 350 pounds.

It is understood that a small amount of silica is beneficial in making these electrodes, so that in practice calcined Rio Tinto cinder is said to be used as the basis of them. Possibly this silica is effective in combining with the FeO. Contrary to popular belief, magnetite is said to melt at easily attained temperatures, about 1350° C. Although highly resistant to chemical corrosion, the anodes are extremely subject to damage from sudden alterations in temperature."—*Eng. and Mining Journal*, April 25, 1914, p. 870. (G. H. S.)

ORE SAMPLING AND WEIGHING AT THE NIPISSING MILL. — "Sampling of mill feed continues to be a vexed question among metallurgists; the merits of the bullion plus tailings method for arriving at the ore content, and the direct-sampling method, are still argued. An example of one mill where actual weighing and sampling are insisted upon will be of interest.

The new low grade mill of the Nipissing Mining Co., at Cobalt, Ontario, has facilities for accurately weighing and sampling every pound of ore that comes into it. Ore from the various mines of the company comes into a bin at the mill. From this bin it is drawn into cars which hold about 20 cu. ft. each. These cars are taken to a platform scale, where an operator balances the load and turns a screw which punches in a card both the gross weight and the tare, or car weight. The operator has nothing whatever to do with the actual figures of the weighing. He simply balances the scale and punches the card. The figures are recorded automatically. Moisture samples are taken from time to time, so that its percentage is known. Thus the weight of ore going into the mill is accurately known and recorded.

At this mill the ore is crushed in a caustic soda solution, then automatically sampled before going to the desulphurizing process described by James Johnston. Pulp is sampled by cutting the stream every six minutes as it comes into the collecting tanks. Since the solids in this pulp are all finer than 200 mesh, and there is no cyanide or other solvent in the solution, an accurate sample is assured. With the weights and samples thus secured, an exceedingly close estimation of the actual ore content is made, which is charged against the mill.

Weights secured by the scale method are checked against those made by specific-gravity estimations on the plant, but by reason of the varying specific gravity of the ores, which differ widely, this latter method is not entirely reliable. In one month the specific gravity may be 2·6, while in the next it may rise to 2·8. This shows the danger of using the specific-gravity method constantly, with an average calculated specific gravity of solids. Serious error might result under such circumstances.

Residue samples are taken from the filter tank at the time of its discharge. A number of cuts are made, and these are all mixed to make the total sample. It will be seen that through these careful weighings and samplings, an accurate knowledge of what is going in and out of the mill is obtained. The bullion production must check with the amount indicated by weighing and sampling, and there is no chance for an error of any kind to pass undetected.

At the high grade mill of the company even more stringent methods are followed. The entire ore tonnage of this mill, averaging about 2500 oz. silver per ton, is sent to an outside sampling plant, where it is crushed, weighed and sampled by machinery especially adapted for such work. Ore, then, delivered to the mill, must be accounted for in its entire silver content, either in bullion produced or in the tailing. In neither of these plants are estimations countenanced. Tonnages are actual weights by scale and contents are by actual assay, corrected by most careful methods, on actual samples, correctly taken."—*Engineering and Mining Journal*, June 27, 1914, p. 1293. (G. H. S.)

MINING.

RIVETLESS CONNECTION FOR STEEL SETS. — "A socket connection for steel I-beams has been devised by Wesley Wait, Newburgh, N.Y. The horizontal member or cap has its flanges slotted to fit a tapered notch so cut in the top of the post and in the plane of the web as to remove a length of the web; the joint is made by slipping the cap into the notch in the post. Loads up to 45,000 lb. were applied by

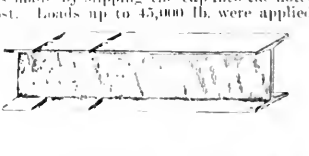


Fig. 1. Manner of Making Wait Joint.

loading blocks at the quarter-points of the cap, and no weakness of the connection was evident, although the yield-point of the caps was passed.

The joint is thought to have possibilities for use in steel mine sets, particularly on account of the quickness and ease of connecting up." —*Engineering News*, May 28, 1914 —*Engineering and Mining Journal*, June 20, 1914, p. 1246. (G. H. S.)

CHINA'S NEW MINING REGULATIONS. — "The government of the new Chinese republic is starting out to build a firm foundation for future national prosperity by adopting new and liberal regulations governing mining. Under the empire, mining like other industries, was hampered by excessive taxation and prohibitive regulations. The mining industry could not be developed without the aid of

foreign capital and foreign enterprise, but conditions have been such in the past that neither were attracted. The new regulations, it is stated, will give full protection to investors and will permit of free and untrammelled entry of foreign capital for the development of the country's mineral resources.

An important change has been made in the law regulating the acquirement of mining lands. According to the laws of the empire, the owner of the surface owned the mineral beneath, which made the acquirement of mining privileges uncertain and expensive. Under the new law, the mineral is the property of the state, and the owner of land and surface improvements cannot hold up mining operations by demanding exorbitant prices for his land, but is required to sell at current market rates.

The new area of a mining property is 10 sq. li (about 1 sq. mile) for coal lands and 5 sq. li for other mineral lands. Still larger areas may be obtained by mining companies by applying to the Minister of Commerce and Industry provided he considers it advisable. Under the old law the area was smaller. The tax on mineral output has been reduced from 5% and 10% to from 1% to 1·5%. The country has been divided into eight mining districts to which will be attached Chinese and foreign technical experts.

The development of mining in China from its present state of insignificance will open a large market for mining and metallurgical machinery and equipment. A new field will also be presented for legitimate investment in mining enterprises."—*Mining and Engineering World*, June 13, 1914, p. 1101. (J. C.)

To Lessen Mining Perils.—"Guaranteed to discover the presence of fire-damp in mines, a remarkable invention was recently introduced to a gathering of colliery and other engineering experts. The inventor is Mr. Alfred Williams, an Oswestry scientist, but the instrument has been perfected in a little laboratory in London. Prof. Silvanus P. Thompson, F.R.S., and Dr. J. Erskine-Murray, F.R.S.E., have tested the invention and pronounced unreservedly in its favour. Messrs. Smillie and Brace declared it to be the kind of thing miners have been wanting for many years. Mr. Brace went so far as to appeal to colliery representatives present to give Mr. Williams their support, and Mr. Smillie hinted that it might prove advisable for the Government to compel the use of the fire-damp detector in every fiery mine. Like most of the inventions which have revolutionised industries the Williams "methanometer" is remarkably simple in construction and application. On a table beside Lord Ninian Crichton Stuart, M.P., who presided, stood a box less than a foot square. From it came a few feet of wire encased in maize-like material, which covers the wire of a telephone. At the other end of the wire was an electric torch. A few inches away lay a cylinder containing some deadly fire-damp from a Monmouthshire colliery. The inventor touched something. The fire-damp was turned on. In the same instant a bell rang to indicate the presence of the gas, and a hand on the dial showed the quantity. Not only will the methanometer disclose fire-damp, but it will indicate the presence of the vapours of petrol, ether, benzol, and methylated spirit. Dr. Erskine-Murray declared it will even tell whether a man has been spending his time in the bar. Mr. Williams has utilised the well-known virtue of platinum black, which exhibits a rise in temperature in the presence of fire-damp. He has overcome the difficulties formerly found attending its use by an ingenious application of the principle of the thermopile, so that directly the platinum black detector responds to the fire-damp a galvanometer actuates a pointer that, traversing a scale, minutely calibrated by means of mine gases, shows at a glance to a fraction the percentage of fire-damp in the pit atmosphere. The apparatus can be made so small that it can be held in one hand, or the scale and pointer may be fixed at any distance from the detector. Thus the dial may be in the mine manager's office and the detector down in the mine, or it may be carried entirely by the miner. Prof. Thompson told the audience that the instrument would register fire-damp down to as small a quantity as 1%. 'When we have its records compared for some time with the barometric records,' he said, 'it may be even possible to know the probabilities of explosions, which at present we are as little able to predict as we are earthquakes.'"—*The Science and Art of Mining*, July 18, 1914, p. 583. (J. C.)

Concrete Head Frame.—"The No. 1 vertical shaft at the Perseverance mine of the Alaska Gastineau Mining Co. has a total depth of 1,544 ft. The collar of the shaft has an altitude of 2,300 ft. above sea level. The shaft is intersected by the Alexander cross-cut at a depth of 905 ft., and the main hoist is built on this level in a station cut in the foot-wall. The ropes from the drums pass up a raise and under deflecting sheaves, then up the ladder-way compartment of the shaft and over the main sheave wheels near the top of the reinforced concrete head-frame, and then return to the cages.

Owing to the collar of the shaft being on a hill-side and also at this elevation a very heavy snowfall being experienced during the winter months, a reinforced steel and concrete head-frame was designed, in which the sheave wheels are placed at a sufficient height to provide a safe margin for over-winding.

In building the concrete head-frame all lumber for forms, cement, and sand was hoisted up the shaft and the rock was taken from the hill-side nearby and washed before using. A small gasoline mixer was used for mixing the concrete. The mixture was 1 of cement, 3 of sand, and 5 of rock. The mixer was placed on the hill-side near the rock pile at a sufficient elevation so that the concrete from the mixer was wheeled along a scaffold and dumped into the forms and tamped. The reinforcement used consisted of old 12 lb. rails and worn hoisting cable. The walls of the head-frame have a thickness of 2 ft. 6 in. at the base, and taper to 1 ft. at the top.

After the concrete had set a few days the forms were removed at the back, and a dry wall consisting of large rocks was built up to a height of about 25 ft. Loose gravel and rock was then sluiced down and placed at the back of the head-frame to a depth of about 25 ft., giving it a batter from the head frame to the hill-side, so as to take up the heavy snow pressure."—C. T. Jackson.—*Mining and Scientific Press*, July 18, 1914, p. 99. (G. H. S.)

Dip Chart.—"The writer has observed that some confusion is experienced by many mining engineers in making vertical sections of ore deposits in mining properties which they have examined, hence it occurred to the writer that the accompanying diagram may be of use to others who at times may be called upon to prepare numerous sections. For example, a great mass of data on strike and dip of ore bodies is collected underground in separate shafts, on different levels, drifts, cross-cuts, raises, winzes, etc. In assembling these data

and preparing vertical sections the various dips of the deposit are frequently plotted just as recorded underground. The result is that a deposit, which may be continuous from the surface to the lowest level, appears on the vertical section as a series of disconnected fissures having dips ranging from 0 to 90. This is due to the fact that observations are taken in various parts of the mine and the plane of the vertical section is rarely, if ever, at right angles to the strike of the vein throughout the course of its dip plane.

Fig. 1 represents the correct and incorrect way of plotting a vertical section of the same vein where observations have been taken at different levels on a vein having a uniform dip. In this ideal figure it is presumed that each level intersects the vein at the same angle. *A* and *A'* represent the plotting of a vein having a uniform dip of 70° on a section the plane of which is at right angles to the strike of the vein. *B* represents the same vein incorrectly plotted on a section the plane of which intersects the strike of the vein at an angle of 30°. Observations of the dip and strike of the vein were made on each level and the vein plotted with its actual dip. *B'* represents the same vein correctly plotted on a section the plane of which intersects the strike of the vein at an angle of 30°.

While the formula for determining the apparent dip of a vein to be plotted on a vertical section may be known to many, this method of determining apparent dips involves a very tedious series of computations if the sections are numerous. The accompanying dip chart, Fig 2, is submitted with the hope that it may save much laborious calculation in preparing vertical sections. This diagram is not original. D. W. Brunton devised a somewhat similar method for his own use at Butte, about ten years ago, and he now has in the process of manufacture "Brunton's Slope Chart," a 9 by 10 in. cardboard diagram with a swinging arm which can be read to a quarter of a degree and is complete for dips ranging from 0° to 90°. This will be entirely satisfactory for office use.

In 1912, D. F. Hewett published a brief description of 'A Graphic Method for Dips on Geologic

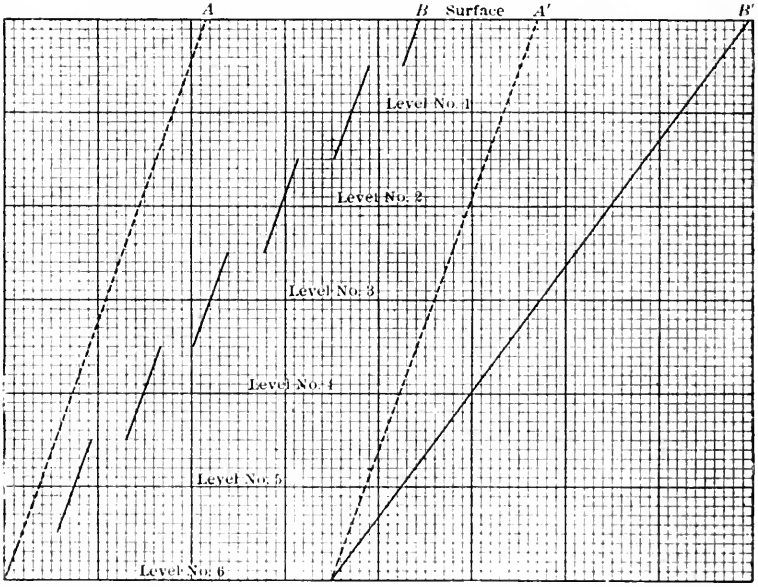


FIG. 1. CORRECT AND INCORRECT WAY OF PLOTTING A VERTICAL SECTION OF A VEIN.

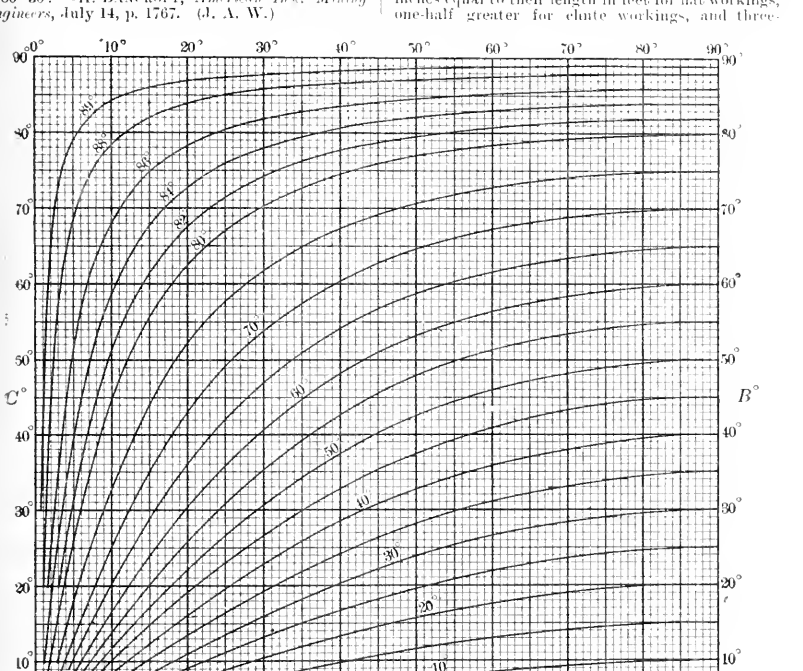
Sections,' but it is too large for field use, and the dip angles have 5° intervals.

The accompanying dip chart, prepared by A. B. Crosley, is of such size that it can conveniently be placed in the back of the ordinary field notebook, and the intervals between angles of dip of veins from 80° to 90° are only 2°. Hence, intersections with the vertical section for veins dipping over 80° can be more accurately plotted with this diagram than by Mr. Hewett's.

Mr. Brunton's slope chart is more accurate than either Mr. Hewett's or the diagram here presented, but it is not practicable to carry Mr. Brunton's chart in the field, its use being confined primarily to the office.

In using this diagram it is to be remembered that the angles indicated at the bottom (*A*) represent the

difference in strike of the vein it is desired to plot and the plane of the vertical section. The curved lines represent dips of veins (*B*). The points of intersection of the vertical lines (above *A*) with the curved lines (true dip lines *B*) at the horizontal lines (*C*) are the apparent dips at which the veins should be plotted on the vertical section. For example, the strike of a vein dipping 86° intersects the plane of a vertical section at an angle of 25°. It should be plotted on the vertical section with an apparent dip of 80° 30'."—H. Bancroft, *American Inst. Mining Engineers*, July 14, p. 1767. (J. A. W.)



A Vein with Dip B, Intersecting a Line of Section at an Acute Angle A°, is Plotted on the Section with an Apparent Dip C°. Formula Tan C° = Sin A° Tan B°.

Fig. 2.—Dip Chart.

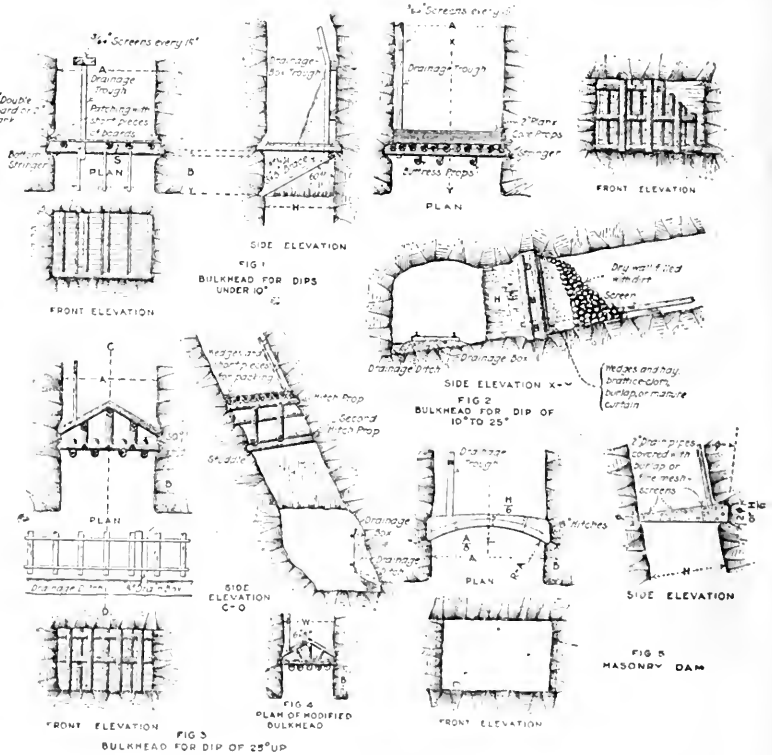
Bulkheads for Hydraulic Filling.—"In the anthracite regions of Pennsylvania, where hydraulic filling for the underground workings is extensively practiced, certain types of bulkheads have become standard for confining the filling to its proper resting place. The design of these will vary chiefly with the steepness of the working; that is, the dip of the coal bed. The particular functions of such a bulkhead are to retain the solid material and permit the passage of water. Certain rules for proportioning the various dimensions are as follows: The headed distance *B* should be one-half the width of the opening for flat workings or workings dipping up to 10°; two-thirds the width of the opening for chute workings, those dipping between 10° and 25°; and the same for pitch workings, those dipping more than 25°. The width of opening is designated by *A* in the illustrations. The props should have a diameter in inches equal to their length in feet for flat workings, one-half greater for chute workings, and three-quarters greater for pitch workings. The spacing of the props across the opening is determined as follows:

For flat workings, $S=\frac{A}{H}$; for chute workings, $S=\frac{2A}{3H}$; for pitch workings, $S=\frac{4A}{7H}$, it being understood that A = the width of the opening in feet, H = the

height of the opening in feet, S the distance from centre to centre of props in feet.

For flat workings, the method of construction is shown in Fig. 1. The props are securely wedged at the top, and manure or dirt is firmly packed around the bottom after the bottom plank has been attached, to prevent leakage. The inside of the line of props is lined with 1½ in. planks or double 1 in. planks, the bottom, sides and tops being carefully joined or patched by short pieces of board nailed to the main boarding and the solid coal or pillar. For the steeper of the flat workings, a hay, straw, burlap or shortest possible period of time. Where the inclination of the work is 18° or more, manure, straw or hay is placed between the two courses of boards as a screener, and in many places a dry wall is constructed to act somewhat as a filter.

A typical bulkhead for a pitch working is shown in Fig. 3. The spacing rule in such cases may call for more props than can be inserted in one row, and therefore a second row becomes necessary. The V-form shown in Fig 3 is one intended to resist the highest pressures. The construction shown in Fig. 2 is also applicable, using horizontal timbers hitched



Timber and Masonry Dam Bulkheads for Use in Working Various Dips to Catch Flushed Filling.

hessian cloth packing is used, or a drainage trough is inserted, as shown, the object of either method being to avoid excessive hydraulic head.

The method of constructing a bulkhead for a chute working is shown in Fig. 2. In some cases buttress props are advantageous. Drainage should be effected by the use of a trough, as illustrated. The chief advantage of this trough method is the fact that the water is rendered available for use again in the into the side of the working for reinforcing purposes. Where the V form is used, a layer of fine manure and dirt is placed as a bedding for the first layer of timbers in the bottom hitches. The planking is placed vertical, with as few nails as possible. Fig. 4 shows an alternative method of arranging the timbers. The use of a screener and a dry-wall filter is important with pitch workings.

Masonry is frequently desired instead of timber,

the most common form of bulkhead being that of a full-struck arch, with radius equal to the width of the opening, the thickness of the bulkhead at the haunches being equal to one-third of the height, and the crown equal to one-sixth of the height. These conditions are illustrated in Fig. 5. Dry walls are also used for bulkheads, and offer the advantage of being both a filter and a retaining wall at the same time. Concrete bulkheads have been tried and found advantageous where great pressures are encountered and timber is expensive."—*Engineering and Mining Journal (Bulletin 69, U.S. Bureau of Mines)*, July 18, 1914, p. 115. (G. H. S.)

MISCELLANEOUS.

A NEW POROUS MINERAL FILTER.—"The new porous mineral septum termed 'Filtros' has such a combination of remarkable properties that, although but recently developed and marketed, it has aroused a widespread interest among mining, metallurgical and chemical engineers. 'Filtros' possesses many of the properties which are required of a filtering medium to make it adaptable to a great variety of commercial filtrations of gases and liquids. It is a white, rigid ware composed principally of silica, and is unaffected by acids, weaker alkalies and chemical solutions. The pores are so minute, in the finer grained ware especially, that they do not permit the finest particles to penetrate, but are so numerous as to allow a free passage of the liquid or gas to be filtered. The porosity is exceedingly uniform. Temperatures up to 1800° F. do not affect it. This is a valuable asset in the filtration of many gases and smelter fumes.

Although 'Filtros' has been manufactured on a commercial basis for but a few months, a number of installations of from 6 to 8 months' standing have been made possible by the accumulation of small numbers of perfect tile resulting from the extensive research work necessary for its commercial production. These installations are giving excellent results in the filtration of acid and of chemical solutions, reducing the time of the operations in some cases by half and producing a filtrate of greatly increased clarity.

The method of producing 'Filtros' was discovered by J. E. Porter, of Syracuse, N.Y., after considerable research work. Broad patent claims have been allowed which have been assigned to the General Filtration Co., Inc., which company has also acquired a number of other patents on filtering devices and the use of a porous mineral medium in some of the industrial arts. Under arrangements with the Harbison-Walker Refractories Co., this concern has developed the commercial production of 'Filtros' and manufactures it on the order of the General Filtration Co., Inc.

'Filtros' is made in a standard tile 12 by 12 by 1½ in., which is a convenient shape, admirably applicable to the majority of industrial operations. These tiles are of excellent hardness and physical strength, capable of withstanding the weight of several hundred pounds when supported on ½ in. bearings. A variety of shapes also are possible for devices of various designs. Tile of practically any grade of porosity can be made and all grades are of exceptional uniformity.

The uniformity of the tile is illustrated by the following air rating tests. These show the number of cubic feet of air per square foot of area per minute that pass through the standard tile, 1½ in. thick, at the given pressure:—

Specimen.	Cu. ft. of air at 2 in. water pressure	Cu. ft. at 3 in. water pressure
1	12·2	24·3
2	12·9	25·0
3	12·0	23·9
4	11·9	24·0
5	12·1	24·1
6	12·3	24·5

An indication of the free passage of air or gas at low pressure is also made evident by these tests. These qualities of uniformity and free porosity will bear emphasis, particularly important as they are in problems involving aeration and agitation. Coupled with the property to resist high temperatures, this remarkable uniformity suggests the application of 'Filtros' ware in the broad field of bag house practice.

Of the many immersion tests made, the few given here will be of interest.

Solution.	Strength.	Time immersed, days	Loss.
Sulphuric acid	96	8	0·013
Nitric acid	38	8	0·016
Hydrochloric acid	70	8	0·026
Water	Distilled	8	0·000
Sulphuric acid	20	35	0·110
Hydrochloric acid	20	35	0·140
Nitric acid	20	35	0·080
Potassium cyanide	50	20	0·09
Potassium cyanide	25	7	0·13
Calcium hydrate	Saturated		
Sodium chloride	13	7	0·03
Ammonium chloride	15		
Ammonium hydroxide	1		

Other tests prove the resistance of 'Filtros' to organic acids and solvents as well.

Engineers and metallurgists confronted by difficulties in problems of filtering liquids and gases, as well as in the problems of cyanide and flotation processes requiring agitation and aeration, will in many cases find the solution in a porous medium having a combination of the many valuable properties of 'Filtros.'"—*Mining and Engineering World*, Aug. 15, 1914, p. 292. (J. C.)

Abstracts of Patent Applications.

(C.) 549/13. Alexander Fraser. Improvements in the extraction of precious metals from ores. 17.11.13.

This specification describes and claims a cyanide treatment of finely crushed ore which comprises the circulation and agitation of the pulp either in the presence or absence of mercury and lead salts. Zinc plates, either plain or with an amorphous coating of lead, mercury or copper, for the precipitation of the precious metals are immersed in the pulp.

(C.) 352/14. F. Picone. Ore concentrators. 17.8.14.

This patent application is for an ore concentrating table having a number of parallel riffles running at about 30 degrees with the length of the tables. These riffles are crossed by other riffles at right angles.

These two sets of riffles are so arranged that one takes the heavy product across the table and discharges it on the side, whereas the lighter product is carried down to the foot and to the other side of the table.

Underneath this table is a device for pulsating air through holes in the bottom of the table, thus pulsating the body of ore on the table. The whole table is reciprocated lengthwise.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B. In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 326/14. Kenneth Somerset Bell (1), George Gilbert Carter (2). An improved shovel for manual use. 18.7.14.

(P.) 327/14. Sydney Edward Thacker Ewing. Improvements in jockeys or rope grips for use in mechanical haulages and the like. 20.7.14.

(P.) 328/14. Walter James Creeser. An improved valve motion for steam locomotives, reciprocating steam engines and the like. 20.7.14.

(C.) 329/14. Jean Harle. Detonating fuse. 20.7.14.

(P.) 330/14. The Skefko (South Africa) Ball Bearing Company, Limited (1), Percy Atkinson (2). Improvements in and relating to rope idlers and the like. 21.7.14.

(P.) 331/14. Ludwig Knipp. Rail fastener. 21.7.14.

(P.) 332/14. Joseph Gouldie. Improvements in either side brakes, combining hand lever action with continuous brake mechanism for railway wagons and other vehicles. 22.7.14.

(P.) 333/14. James Edward Chard. An improvement in spray nozzles. 23.7.14.

(P.) 335/14. William Charles Stephens. Improvements in or connected with rock drilling machines. 23.7.14.

(P.) 336/14. William Charles Stephens. Improvements in valve gear for rock drilling and like machines. 23.7.14.

(P.) 337/14. Charles William Dowsett. Improvements in automatic regulator for connecting rods. 27.7.14.

(P.) 338/14. Evelyn Clifton Eamton. Improvements in the electrolytic recovery of metals from their solutions, and in apparatus therefor. 27.7.14.

(C.) 340/14. Sidney Thomas Hollingham (1), Harry Percy Budd (2). Automatic self coupling buffer for use on railway rolling stock and the like. 29.7.14.

(P.) 341/14. Orenstein Koppel, Ltd. (1), Martin Christensen (2). Improvements in and relating to axle bearings for trucks and the like. 30.7.14.

(P.) 342/14. Charles Bennett. Improvements in the manufacture of bricks, tiles, slabs, blocks and the like. 30.7.14.

(C.) 343/14. Kenneth Bertram Lamont. Improvements in securing collars or like parts to shafts. 30.7.14.

(C.) 344/14. The Vulcan Foundry, Ltd. (1), William Collingwood (2). Improvements in or relating to riveting machines. 30.7.14.

(P.) 345/14. Jessie Clark. An improved dust collector for machine drills in use in mines. 31.7.14.

(C.) 346/14. Amos Ivan Addison. Process of hydrocarbons. 31.7.14.

(P.) 347/14. Moses Mittledorfer Marcuse. Sanitary closets and the method of treating the deposits usual to the proper use thereof. 1.8.14.

(P.) 348/14. Arthur Eli Abrahams (1), George Klahn (2). A self opening gate. 5.8.14.

(P.) 349/14. Alwyn Jacobus Francis. A new and improved method of automatically locking balcony gates of railway vehicles. 5.8.14.

(P.) 355/14. Frederick Victor William Swanton. Improvements in power producing apparatus. 20.8.14.

(C.) 359/14. John Wilkinson Kirkland. Improvements in controlling means for electric motors. 21.8.14.

(C.) 360/14. John Wilkinson Kirkland. Improvements in metallic resistances for use in the speed controlling means of electric motors. 21.8.14.

(P.) 361/14. James Hardgrave Shekleton. Improved appliances for locking nuts or bolts. 22.8.14.

(P.) 365/14. Frederick William Lace (1), William Patrick Cowan (2). Improvements in water supplying means for rock drilling machines to prevent the formation of dust when drilling. 26.8.14.

(P.) 368/14. Edward Dosser. Warning signal device for mine hoisting machinery. 29.8.14.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non receipt of Journals and Notices.

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Vol. XV NOVEMBER, 1914. No. 5.

Proceedings
AT
Ordinary General Meeting, November 21st, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building on Saturday, 21st November, Prof. G. H. Stanley (President) in the chair. There were also present:—

13 Members: Prof. J. S. Cellier, Messrs. C. Toombs, J. Watson, H. A. White (Members of Council), W. Beaver, T. N. Dewar, J. M. Dixon, H. W. Gill, S. Newton, J. A. Taylor, A. Thomas, W. E. Thorpe, and E. M. Weston.

7 Associates and Students: Messrs. W. Human, H. G. Kay, A. King, A. Milliken, G. F. Nisbeth, P. Scatterty, and W. J. Sutherland.

5 Visitors, and Fred Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBER.

Messrs. J. Watson and H. A. White were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidate for membership had been elected as follows:—

AGNEW, JOHN ALEXANDER, 1, London Wall Buildings, London, E.C. Mining Engineer.

The Secretary notified that the following had been admitted an Associate since the last meeting.

HADDEN, EDWARD LINDSAY, Sisanna Mine, *per* Private Bag, Bulawayo, Rhodesia. Mine Manager.

GENERAL BUSINESS.

NEW PROCEDURE RELATING TO MEETINGS AND PUBLICATIONS.

The President: Under General Business, I have rather an important announcement to make. The Council have felt that the attendance at recent meetings has been very far from satisfactory and, in addition, it has been very difficult indeed to get sufficient papers and business for the meetings. They have felt that it was largely due to the fact that we have too many meetings per year; one a month—12 in all, being more than most Societies, at least oversea, hold. It was therefore proposed and adopted to hold only eight meetings a year. We shall have a break in the summer and in the winter; the December and January meetings will be dropped, and also the June and July meetings; but it does not follow that no *Journal* will be issued. As a matter of fact, the Council have arranged to issue one *Journal* in each of these breaks.

Another modification in procedure is that original papers in future, instead of being printed in galley form and distributed at the meeting as they are to-day, will be printed in the *Journal* and circulated before the meeting; so that at the next meeting, at which the paper is nominally presented, members who care to discuss it will, it is hoped, be ready at the meeting with their discussion. So that at that one meeting the paper will be dealt with very fully, and members who are interested in that particular subject will feel that it is worth while attending the meeting. We hope, therefore, in that way to have a much better attendance and a more interesting meeting. It follows, too, that it will not be necessary to read the paper in full, as it will have been circulated beforehand. All that will be necessary will be for the author to read a summary and elucidate such points as he he may have brought forward with the may care to, and explain such diagrams as paper. Also, the Secretary will be instructed to get all the discussion possible from likely contributors to be presented at that particular meeting. The Council hopes that this will result in improving attendance and interest in meetings and, there-

fore, will be in the interests of the Society. In regard to this new procedure, of course the Council have power to act in the matter; but, in communicating it to the members at this meeting, I should just like to hear whether they have any observations to make.

THE SOCIETY'S ROLL OF HONOUR.

Well, if there are no observations in that connection, the next point is that several of our members—a good many, in fact—are on active service, and we desire to have a list of all those who are so acting. We have a Roll of Honour of names that are already known, it is as follows:—

Member of Council.

Hilditch Smith, Lieut G., 1st Transvaal Scottish.

Corresponding Member of Council.

Stokes, Ralph, Royal Engineers.

Members and Associates:

Brise, H. R. Ruggles-, Natal Light Horse (now Lieut. and A.D.C. to the Governor-General, Lord Buxton).

Gardner B. L., Public Schools Battalion (Middlesex).

Lucke, P. K., Legion of Frontiersmen.

Rabone, P., Rhodesian Volunteers.

Robinson, R. W., 5th Inniskilling Fusiliers.

Simpson, G., junr., Public School Corps.

Swinney, L. A. E., 28th Middlesex (Artists).

I am sure that we all feel a keen sense of pride that our members have come forward in this way. Besides the names I have already read out, there must be many more members, and we wish them the best time it is possible for them to have in such an occupation as they will have to go through for the next few months, and a safe return.

NOTES ON TREATMENT OF ZINC-GOLD SLIME.

Mr. Andrew King (*Associate*): The modification of clean-up practice recommended by Mr. H. V. White in the September *Journal* was tried at two successive clean-ups at the cyanide works with which the writer is at present connected.

The clean-up plant on this mine is such that it is possible to treat the various products carrying white precipitate quite separately and with excess of acid without interfering to any great extent with the dissolution of the rest of the zinc. The material so treated came from the top compartments of the weak boxes, together with the white product from the filter press, short zinc with a large quantity of white precipitate resulting from the trommelling of zinc still good for further precipitation, and a certain amount of zinc from strong boxes to neutralise the excess acid with which the other products had been left in contact overnight, the whole making up about 30 of the total zinc dissolved. The resulting gold-slime was washed, filter-pressed, calcined and smelted separately. The amount of calcined slime from the white precipitate products amounted to 27·8 of the total. The fine gold yielded by the calcined slime was as follows:—

(1) From separate treatment of the above white precipitate products ... ... 18·1%

(2) From remainder of zinc dissolved during the clean-ups 24·0%

(3) From total calcined slime ... 22·3%

Average fine gold in calcined slime from six clean-ups during July, August and September, 1911 23·17%

From these figures it would appear that no particular advantage is to be gained on this mine by separate treatment of the white precipitate products.

This was further shown during the fusion of the calcined gold slime. A small reduction in fluxes was made (of course, before the above figures were available) and it was found that complete fusion was much slower, and from the result of the subsequent slag smelt it would appear that more gold than usual was retained in the slag.

The writer regrets that his results do not bear out those obtained by Mr. White, and concludes that there must be a considerable difference in the composition of zinc-gold slime on different plants.

Mr. A. Thomas (*Member*): I am not prepared to say anything about the acid treatment of the slime, but will discuss some of the details of smelting. The author says that by using powdered graphite between the crucible and fireclay liner he is able to get 50 more fusions than he usually obtained. We would be able to understand the value of this arrangement better if he would tell us the average number of fusions obtained. Also, does he experience any difficulty in removing the liner from the crucible after the liner is worn through? And does the graphite adhere to the liner? This is important. Many mines use the crushed liners which are gold-bearing, as a source of silica, whereas the sand which the author mentions using would be comparatively barren. I take it, too, that the crushed liners are quite as suitable as the

sand in counteracting the effects of corrosion on the liner. But if the powdered graphite adheres to the liner in any large quantity, I do not think it could be satisfactorily used for fluxing on account of the graphite acting as a reducer and causing the base metals which are usually present in the slime to be brought down with the gold bullion.

The author says in order to avoid the formation of cracks in the liner it should be evenly supported and heated. I might also say that a good deal of the cracking of liners can be minimised by placing all new liners in the furnace with the first charge of the smelt, when the temperature of the furnace has not been brought up to the smelting point, thus allowing the liner to be properly annealed.

The author advises preliminary experiments with two pots to determine the necessary amounts of flux required for the remainder of the smelt, and I should like to know in what part of the furnace those two pots should be placed on account of the varying temperature in different parts of the furnace. With regard to the amounts of sand and borax required for economical smelting, I quite agree with the author that elasticity for conditions must be allowed. My own conclusions are that it is not economical to cut down the borax too much, but rather to have a slight excess. There are a good many mines on these fields where the slag from the gold slimes smelt is re-treated, together with other by-products which are collected from the amalgamating room, extracter house, and around the smelting rooms, which may amount to a couple of tons per month and is mostly silicious matter containing varying quantities of gold, therefore the slight excess of borax that may be used on the gold slimes smelt is not wasted, but assists in making the by-product smelt satisfactory.

THE DETECTION OF THE PLATINUM METALS IN CUPELLATION BEADS BY MEANS OF THE MICROSCOPE.

By C. O. Bannister, *Assoc.*R.S.M., F.I.C., and G. Patchin, *Assoc.*R.S.M.

The President: There has been no discussion on this paper, and consequently no reply is needed, but I have heard from the authors that they very much appreciate the kind way in which the paper was received, and the remarks that were made.

NOTES ON THE DUST ALLAYING PLANT AT THE FERREIRA DEEP.

By S. Newton (*Member*).

This paper gives an account of the experimental work carried out in devising a successful method of allaying dust on sand dumps, of the plant installed in applying the process, and of the practical results obtained during the six months, May to October, of this year. The months named represent the worst period of the year, from the termination of the rainy season to the first rains of this year, and gave the plant the severest test it is likely to undergo.

The original experiments, initiated by the Metallurgical Department of the Rand Mines, were carried out on the Deep Section of the Ferreira Deep, from November, 1913, to March, 1914, when the results were so satisfactory that the management of the mine decided to erect an efficient plant on the lines of the experiment, which had been carried out on a smaller scale.

In the first place, heaps of dry, fine sand, of about one ton each, were made in conical form in exposed positions, where they were exposed to normal atmospheric changes. The pumps used for spraying were the "Deeming" hand spray pumps. These experiments were carried out during the varying conditions prevalent during this period of the year, viz., heavy rains, and continuous light rains, and with heavy and light winds. The several heaps of sand were sprayed with any cake-forming substance that suggested itself to those carrying on the experiments, and daily observations and notes were made as to the efficiency and cost of material used. Among these were: Water glass in combination with magnesium chloride, molasses, cement, slime residue pulp, crude oil, waste oil, and acid mine water, but owing to the action of rains and bad weathering, and from an economical standpoint, some other material, both cheap and handy, was required to make a success on a large work-

ing scale. It was then suggested by Mr. Trump, the Manager of the Ferreira Deep, that salt had certain qualities which might solve the problem.

Salt solution was made up to the density of sea water, and also to saturation. The sea water test gave a slight crust, but was not sufficiently durable to resist the varying weather conditions. The saturated salt solution gave a good hard and effective surface which lasted a considerable period. At this time the question of mixing was suggested by the metallurgical staff of the Rand Mines, and those carrying on the experiments commenced by mixing certain parts of salt and residue slime pulp together, and spraying the mixture on to heaps of sand. This spray consisted of slime residue pulp (65% moisture), with 3·5% salt (on weight of pulp) added to it. The first result was so good that the heaps sprayed with it remained intact, while those which had been treated with other materials disappeared.

There is obviously much to recommend the use of slime, as it is a residue material costing nothing and easily obtained, while the salt also is a very cheap local product.

Spraying on a larger scale was then commenced by the erection of a cone-bottom vat holding four tons of pulp, fed by a branch from the delivery to the residue slime dam, the slime being agitated by air at the bottom of the cone, and elevated by a Knowles Air Pump. A 1½ in. main was carried up the west side of the sand dump, which was sprayed by means of various shaped nozzles fixed on a 1½ in. rubber hose. This spraying was carried on during the months of February and March, 1914, and owing to the promising results obtained under varying conditions, the management then designed a plant capable of dealing with the current residue dump.

The plant consists of:

Cone-bottom vat (capacity 50 tons of 1 to 1 pulp), 10 ft. in diameter, 12 ft. 6 in. in the vertical, cone 15°, ¼ in. plate, with 1 in. air connection at bottom of cone, and suction pipe 2 ft. above bottom of cone.

Pump-house, 20 ft. x 12 ft. One 6 x 8 in. Gould triplex power pump, 168 lb. working pressure, revs. 15, driven by a 25 h.p. motor, 500 volts, revs. 725 per min. On the pump is fitted a safety valve and also water service for glands to keep slime from coming up to the packing. For this work we have a 3 x 4 in. Gould triplex power pump, driven by a 5 h.p. motor, 960 revs., belt driven. Outside the house is a tank 4 ft. x 6 ft. to supply water to the gland pump.

A 4 in. *delivery main* leads from the pump along the up-track, round the working face, and down by the return track, with T-pieces, plug-cocks, and unions for connecting up to the fire-hose with 1⅛ in. nozzle. A 100 ft. length for the track, and a 250 ft. length for the long face of the dump is used.

The residue slime pulp is by-passed to the cone-bottom tank by a pipe from the discharge main to the dam. The amount of salt to be added is hoisted up and dumped into the pulp. After being agitated for about two hours, the mixture is in a fit condition for spraying purposes.

The best results are obtained by spraying on a pulp containing from 56% to 60% moisture, and made up as follows:—The residue pulp is elevated to the mixing tank averaging from 52% to 54% moisture; but the gland service also adds a certain amount of water, and so a good thick pulp is available which is efficient as a dust allayer.

On May 17th, 1914, the plant was started, and for the first month trials were made with various hoses and connections, and demonstrations made to prove efficiency on a large scale. From June 10th to 13th the whole of the current face and east and west tracks were sprayed so as to get the whole of the dump under control. Orders were then given to the contractor to tip at the south-east corner and continue in short tips, filling each one finally before moving on. As soon as each tip was full it was at once sprayed, and this method was continued, the idea being to discover if one spraying was sufficient until its turn came round again. But the filling of the whole current face of the dump took seven weeks, and during the very dry season this was found to be too long a period between spraying. The point arrived at was that slime coating on exposed portions of the dump face is very effective and may last three or four weeks, but when a portion shows signs of wear it is well to give it another treatment at once, otherwise dust will blow from this part on to other surfaces, and give rise to an amount of drift which will make the whole face bad, or rather look bad. During the very dry months of August, September, and October, spraying was done about three times a week; when the winds were very bad about two hours at a time. When we get rains a good coat

remains efficient for three or four weeks. Heavy rains are not at all detrimental, and steady rains improve the spraying surface. It is always more effective to spray a new tip when it is moist, or during or just after rain, as the slime has moisture to draw upon, and owing to the hygroscopic nature of the salt the cake remains moist much longer. The track requires to be sprayed more often than the current face, owing to the spragging, trampling, and general use of the track. An overturned truck or two, quickly drying, will raise dust and look bad without amounting to much in bulk.

As a result of the spraying, the contractor and boys work in a perfectly clear atmosphere, instead of being in a blinding sandstorm for days when, at periods, the trucks are not visible until close to the boys. After a week-end of blizzard weather it was a usual occurrence to have to spend some considerable time clearing tracks and ballasting a line that seemed to be suspended in mid-air. This now is never necessary. Incidentally, there is much less wear and tear on trucks; machinery in the near neighbourhood must benefit, and married quarters welcome the advent of the dust spraying.

The total area under control amounts to about 600,000 sq. ft. This includes the east and west flanks, which do not require much attention, as one good spray every five or six weeks keeps them in condition.

The Resident Engineer must be complimented on the design and erection of such an efficient plant, and the loyal support of the Cyanide Foreman and Cyanide Staff in making this method a success must be recorded.

The experiments were carried out by myself and Mr. Leyson, under the direction of Mr. Wartenweiler, of the Rand Mines Metallurgical Staff, and Mr. F. J. Trump, Manager, Ferreira Deep, whose help and suggestions made it the success it is.

APPENDIX.

Duty of spraying pump: about 50,000 sq. ft. per hour.

Cost, including salt, power, labour, etc.: 6d. per 1,000 sq. ft.

Spraying pump has run on an average 22 hours per week.

Cost of erecting plant: about £900.

Amount of salt used: 5 % salt to dry slime.

Total cost per month during the six months: £25.

Mr. H. A. White *(Member of Council)*: I would like to propose a very hearty vote of thanks to Mr. Newton for his most excellent and practical paper. He has certainly improved the paper considerably by the diagrams on the black-board, which have rendered the whole thing quite clear to us, and we can see the amount of work he has put into it to get the excellent results obtained. I have no doubt the Ferreira Deep dump is high enough to catch all the winds that blow, as Mr. Newton points out; it is therefore important to have some device of the kind mentioned. There is one point that struck me as being rather peculiar, and that is, what property of salt it is that he depends upon. He seems to refer to the hygroscopic nature of the salt; I should say that pure salt is not hygroscopic at all: the effect obtained has, possibly, something to do with the impurities in the salt he is using. I should like Mr. Newton to tell us in his reply if they tried calcium chloride at all; this, of course, strikes one as being very efficient for that sort of work, although the cost may be prohibitive; but in the experiments they have made they may have used it, and may have got some results. Also, it would be interesting to know what is the next best thing to salt. Mr. Newton has rather skipped that experimental work and has let us off rather lightly with reference to its details. It is interesting in experiments of this kind to know all the results obtained, even though there may have been many complete failures, for it would tend to prevent unnecessary repetitions.

Mr. C. Toombs *(Member of Council)*: I have much pleasure in seconding the vote of thanks to Mr. Newton. About two years ago, at Mr. W. T. Anderson's instigation, a considerable amount of experimental work was done on the East Rand Proprietary Mines in connection with the allaying of dust from dumps. Several different materials were tried, including assay slags and sodium silicates made on the mine, and containing varying ratios of Na_2O to SiO_2. These were very promising, but the crust was found to weather badly on the large scale, breaking up into sodium carbonate and silica. Slime residue was also tried and rejected, the reason for this being that it did not mix with the sand, but simply formed a cake over it, which did not stand up to ordinary wear and tear. The mixture of salt with the slime residue was not tried by us, and it is really difficult to see what

effect the salt has on forming a successful crust; as Mr. White says, common salt does not deliquesce. With regard to slime residue not costing anything, that depends entirely on whether a decantation plant is in operation or a filter press plant. If the former, it cannot be said that slime residue costs nothing—it costs exactly the value of the gold-bearing solution which would otherwise be pumped back from the slime dam.

The material finally selected at the East Rand Proprietary Mines was a black clay, of a pot-clay nature. Most of you know, I expect, that there is a seam of very good fireclay running through the East Rand district—at Boksburg North a company is making very excellent firebricks, slabs, and also assay crucibles. Near Driefontein sand dump there is a big deposit of this black clay, and it was found on trial on a small scale to give results sufficiently good to justify it being tried on a large scale, when it was found equally successful. It's advantages are, that it is very light and easily kept in suspension, but its chief advantage is that it actually percolates into the top layer of sand for about three inches, binding it firmly together. Occasionally trouble is caused by natives running over the crust and breaking it, the wind getting under the cracks and raising some dust, but on the whole it is a great success.

To Mr. R. Nethersole, who is in charge of this work, is due the credit of finding out the good qualities of the clay for this work, and perhaps he may later give us a contribution on this subject.

Mr. E. J. Moynihan: It has been very interesting to listen to this paper. It is a very good instance of the general fact that some things which do not seem to be profitable on paper, and have been so regarded for many years, when they are at last carried out for non-financial reasons, pay. We have had this dust blowing about the Rand for, I suppose, about 25 years. Until quite lately it has apparently never struck anybody to try and stop it. It has now been shown that it can be stopped quite easily at small expense. It is quite certain that the outlay is not really an expense at all; it is profit. It is a profit which the gentleman of a book-keeping turn of mind would never appreciate until he was shown it as a concrete fact. Any cheap method that stops this dust blowing about is a benefit. I think it likely that, later on, it will be found that there are many other efficient methods of doing the same thing: perhaps cheaper than this one. That does not detract, in any way, from the credit due to the people who have carried it out; for the man who first does a thing—it does not matter how inefficiently, compared with later methods—deserves an enormous amount of credit. It is comparatively easy to come along afterwards and say: "You can improve this," or "You can improve that." In this connection, I would like to point out some facts known to myself which might lead to another method. Some years ago I was cyaniding, on the Black Reef, a mixture of sand and slime. There was not very much slime in it, but there was enough to make it muddy when it was taken out of the truck. I never had any dust trouble at all. There was not enough slime to prevent percolation; yet it absolutely prevented any dust blowing about the dump. The dump just dried and caked hard. I suggest to try whether the mere mixture of a certain amount of slime with each truck, or alternate truck, of sand, would not produce the same result on a big scale.

Mr. E. M. Weston *(Member):* With regard to this prevention of dust, for some years past I have thought that nature had really been showing us a way in which to deal with this matter, and that people had been rather blind not to have taken the hint before. I believe that the real way to deal with this dust question on dumps, sufficiently and permanently, is simply to encourage the growth of vegetation on them. Now, that could be quite easily done. Two examples have been staring the people of the Rand in the face in different directions, almost every day, and we do not seem to have taken the hint. Passing the Robinson Deep dump, on the side of the dump nearest the road, it will be seen where, according to the practice on some mines, just bare ash from the furnace has been dumped with the sand, and it is surprising to see the amount of vegetation which has gained a footing. A still better example is shown on the dump at the Simmer & Jack, which can be seen by anyone travelling along the railway line. There is there quite a healthy crop of vegetation on the dump; a dump apparently consisting of pure sand and a little ash. It seems to me if the experiment were made of simply tipping along the lines which go up and which go down, say, a truck or two of red earth or soil, or ash and soil mixed, and allowing the weather to wash the ash and soil

gradually down the dump until the whole face of the dump was covered, and perhaps repeat this once or twice: that in a year or two quite a permanent and effective crop of vegetation would be found to have sprung up and cured the trouble altogether. I note that on the East Rand Proprietary Mines they are pumping a soil emulsion on the dump, on which, no doubt, vegetation will soon flourish. This method of establishing vegetation would, of course, apply only to the sides and tops of working dumps. On the growing end, no doubt the expedient favoured by the author will be found useful. I would like to point out that the side of the Robinson Deep on which the vegetation grows is that exposed to the prevailing winds, and therefore most likely to suffer from the shifting sand bogey, yet nature managed the job.

Prof. J. Cellier *(Mt. Member of Council):* Mr. Newton's remarks have been most interesting and instructive, and, if I may be allowed to ask him, would he give us some details regarding the other experiments made, for which some of the other members present have also asked. There must have been a certain amount of good resulting from the other experiments, and they would, no doubt, be useful. The property of salt referred to as hygroscopic is probably increased by a certain amount of magnesium chloride present in the cheap salt used in this country. Tipping ash on the dumps is a preventive, as Mr. Weston says, but there are some 30 millions of tons of ore milled, of which, say, 60 % is sand, and there are some two million tons of coal burnt, which do not give nearly two million tons of ash. The ash has a certain amount of alkalinity in it, and tends to neutralise the acidity produced by decomposing pyrite, thus giving the vegetation a chance to grow. Where the dump is so very acid, as the author remarks, vegetation is never found growing. The process of putting ash on the dumps would be to try to cover them with less than 1/15th part of the other material; and that is rather out of the question. On the Rand, the most obvious feature of mining is the number of sand dumps lying about, and they are certainly more dangerous, I think, to health than are the earth tremors about which we hear so much. On the whole, I think the experiments mentioned are very interesting and instructive, and especially from a health and, therefore, as has been pointed out, from an economical point of view they are to be welcomed.

The President: The point that struck me with regard to the growth of vegetation on dumps is, as one knows perfectly well, that it will not grow when the sand is acid, though it may when the acid is neutralised. In any case, the dump must be at rest and fairly well sheltered before anything can grow. When a dump is being removed by wind at the rate of 3,000 or 4,000 tons over the week end, it is impossible to get anything to grow on it.

Mr. John Watson *(Member of Council):* Could Mr. Newton please tell us where the salt is obtained from and what price is paid for it per cwt. or per ton?

Mr. S. Newton *(Member):* Ordinary commercial salt, I think, about 50s. a ton. It is just the ordinary salt from South African salt pans, that is used in the compounds.

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

(Read at May Meeting, 1914.)

By G. H. Stanley, A.R.S.M., M.Inst.M.M. *(President).*

REPLY TO DISCUSSION.

Prof. G. H. Stanley *(President):* I must first thank Mr. Martyn for his appreciative remarks, but shall endeavour to controvert his criticism. Obviously, he overlooked jigs among concentrating appliances, so I need not deal with them, but, in general, if we have on a flat table and subjected to the action of flowing water, two particles of the same size, but of different minerals, the one of lower specific gravity will, under any circumstances, tend to be removed first by the water. But as the size of the particle of heavier mineral decreases a point will be reached at which they will be moved with equal readiness, and as its size still further decreases it will, at first, be moved instead of the larger particle of light material, though after that, as its size approximates to the depth of the assumed stationary film of water, the difference becomes less and less, and eventually the case may be reversed, as Mr. Martyn states. So that if we consider a layer of particles, one particle deep, on a concentrating table containing mineral particles of such divergent size there is apparent justification for his remarks. But on what tables is such

a layer to be seen? Instead, on all shaking tables to which a classifier underflow is likely to be fed we have a thin layer admittedly, but still several particles deep, be it of the Wilfley type or vanner, and in such a layer separation or stratification is taking place, presumably, in accordance to a great extent with hindered settling laws.

On such tables, though, the particles appear to be subjected to two actions; one the rolling motion described by Mr. Martyn, and the other the tendency to settle. The relative importance of these in affecting the behaviour of the particles appears to me to vary with the absolute size of the particles, the former being more important with the finer sizes, and the latter with the coarser size, so we are both in a measure right; but the size of particles in which the first is important is small, so small that while a gentle current of water alone may effect separation, any other agitation keeps the pulp in suspension; and the second is more important where, short of giving an excessive slope, other agitation is necessary to enable the light particles to be transported readily by the water current and at the same time cause settlement of the heavier.

The first set of conditions would apply rather to the overflow (slime) than to the underflow of classifiers, which was the product I mentioned, and in which I think the second is relatively more important.

If a fine particle of heavy mineral happens to reach the table surface on being fed in admixture with gangue (which, having been classified out by a hindered settling classifier, is equal falling with it under hindered settling conditions) it will possibly behave as Mr. Martyn indicates, but otherwise it may never reach there, for it will not be able to sink because the gangue particles will have the same sinking tendency as itself. That is, the theoretical grounds for my statement were, that if the particles in the classifier product were equal falling under hindered settling conditions it would be useless to attempt further separation (or concentration) under the same conditions. Under free falling conditions the ratio of the diameters of equal falling particles is less, and if the product of a free falling classifier be fed to a concentrator working by hindered settling, obviously the heavy mineral particles, being on the average larger than from a hindered settling classifier, gangue remaining the same, will now tend to sink faster and allow of separation (concentration). And, of course, following up the line of argument, if sifting be used (with suitable sieve ratio) the particles may be still more nearly the same size and concentration still more easy and perfect; and in this respect I am afraid Mr. Martyn is in conflict with the general view. Further, particles so fine as to act in the manner considered above should be absent from the *underflow* of a really *efficient* classifier of either type; that they are present is due to inefficient operation, and losses in Wilfley tailings have been traced to the presence of "fine heavy mineral," the remedy suggested being more efficient preliminary classification!

Mr. Martyn apparently makes a contradictory statement later on, *i.e.*, "This is why closely classified products are far more easily concentrated than mixed products . . . ." Surely in closely classified products the divergence of sizes of particles present is less than in mixed products, and on this statement a free falling classifier product should be more easily concentrated than that from a hindered settling classifier with its wider divergence of sizes. I specified, however, ideal free-falling conditions as the basis of comparison, but these are so far from being realised in practice that a professed hindered settling classifier may give even a better classified product, judged by practical results in concentration.

Mr. J. M. Neill, also, I have to thank for his appreciation, and for expressing more clearly one of the objects of classification. He also shows that in gold metallurgy on the Rand and elsewhere, classification into equal falling groups is more valuable than classification into equal sizes, and with that, of course, I agree, since the heavy pyrite is required in the underflow for regrinding equally with, or even more than, the coarse quartz. From this point of view hindered settling should be preferable, since as the equal falling ratio is wider with hindered settling classifiers, finer pyrite should accompany a given size of quartz particle than with free-falling, and consequently the pyrite delivered in the overflow should be finer and yield, therefore, a higher extraction on cyanide treatment.

The thanks of all operators are due to him for his account of classification experiences at Benoni, with description of classifiers and ingenious regulating device, on which I congratulate him. This description and figures attached add greatly to the value of the paper. Still, the fact that 17·6 of −200 remains with the sand

underflow, and about 3 of +200 overflows with the slime illustrates that the classification is still far from *perfect*, though apparently quite satisfactory for the object in view.

Mr. Robertson states possible ways in which improved classification on the Rand might result in benefit. As he states, there is but little room for improvement in extraction. It is difficult also to see how shallower classifiers can be used without introducing some mechanical means for removing the settled coarse material, as in the "drag" classifiers used elsewhere, which might offset other advantages. I am afraid a development on the lines he suggests would soon result in a very large settling area being reached in order to deal with adequate tonnage, and necessitate mechanical means for removing and transporting settled sand, or otherwise height for fall would be needed. The second possibility he mentions, namely, reduction of sand tonnage tube-milled, is more important and would result in considerable saving both in capital and operating costs. It is, however, difficult to suggest a method for effecting improvement with the present character of the pulp without introducing some countervailing disadvantage; still, it may exercise the ingenuity of inventors.

In conclusion, I again thank the gentlemen named for adding greatly, by their contributions, to the interest of the paper.

ROCK TEMPERATURES.

(Read at May Meeting, 1914.)

By E. J. Moynihan (*Associate*).

REPLY TO DISCUSSION.

Mr. E. J. Moynihan: When I wrote my paper, which is a strong attack on the official views on the subject, I hoped to provoke a discussion of value and volume. I regret that there has hardly been any discussion at all, and what there was, was mainly of negative value. Most of the advocates of the official view have remained quiet; a not uncommon procedure on their part when vigorously attacked.

Referring to Dr. Moir's contribution, I am glad to see that he agrees with me that the conclusions of Messrs. Whitehouse and Wotherspoon are ridiculous and unwarranted. I do not expect to upset the "common opinion" that the Rand temperature gradient is 1° F. in 200 ft. I only claim to have shown that this conclusion is entirely unjustified in view of known figures. To upset an established belief is too much to expect from one paper or one man.

Dr. Moir says I am not entitled to state that the Rand gradient is "not abnormally low," as compared with European gradients. The words he quotes do not appear in my paper; nevertheless, I think I have shown that the statement he attributes to me is correct. The average gradient of rock temperature is usually taken as about 1° in 80 ft.; the air temperature in about a dozen of our mines gives gradients steeper than 1 in 100; and even assuming that these correctly represent the original rock temperature gradient, they are not *abnormally* low, and no amount of quibbling will produce from such data, with any shadow of reason, such an abnormally low temperature gradient as anything like 1° in 200 ft. —much less 1° in 250 ft. Dr. Moir says he is "by no means convinced that a freshly made bore-hole is necessarily cooler than the rock." I never suggested that it was cooler than the rock was at that point at the time the hole was made. A hole drilled dry might be hotter than the surrounding rock for half an hour or so after it was made. This criticism appears to be pointless unless he means cooler than the rock was *originally*. But of that Dr. Moir himself is convinced, apparently, for he says "most of us will accept the author's view—that in most cases the true temperature is higher than the bore-hole reading." Dr. Moir is of opinion that "it takes a lot of compressed air to counteract the (temperature) effect of breathing in a close space." It is easy to show the contrary. An ordinary man breathes something like 1½ cub. ft. of air in a minute, expired at about 99° F. Six men, therefore, breathe about 10 cub. ft. One ordinary big machine uses at least 120 cub. ft. of compressed air, and sends it out at not far from freezing point, far more than counteracting the effect, not only of the presence of men, but of illuminants also. Any one who has ever worked a rock drill knows that the temperatures of the working places, even if they are fairly big stopes, are materially lowered after the drills are started up. It is possible that, for a short time, an air observation in a non-worked dead-end may show a higher temperature than that of the adjacent rock-wall; but it is extremely im-

probable that it will show as high a temperature as the original temperature of the undisturbed rock at that point.

Replying to Mr. Meyer's criticism: Mr. Meyer seems to have materially misunderstood my paper. He attributes to me views which I have never expressed at all. I have never said, explicitly or implicitly, that "depth is the only factor governing temperatures." What it does govern, is the lowest mean temperature attainable by unrefrigerated ventilation. He asks: "In order to predict the *greatest* depth to which mining can take place without refrigeration, is it not more important to ascertain the condition of increase in temperature of the rock of opened-up mine workings?" This question shows a confusion of ideas. The increase of temperature with depth of opened-up mine workings itself depends on ventilation and refrigeration, whether intentional or accidental. It therefore cannot be determined at all, apart from the effect of these factors. In order to know their effect, the real, original, undisturbed rock temperature gradient must be ascertained. A mine may be ventilated as much as conceivable (without refrigeration) and a mean temperature gradient, even in the outer skin of the rock walls, lower than the gradient of the air due to compression will never result unless the air is refrigerated in some fashion, by accident or design.

In commenting on Messrs. Whitehouse and Wotherspoon's paper I pointed out that they had no right to attribute the whole of the lowered mine temperature between January and May to the installation of the fan, since there was a simultaneous large reduction in surface temperatures, sufficient to account for the slight reduction in mine temperatures. To controvert this, Mr. Meyer produces evidence that a closed place had its temperature reduced 1° in a year by a strong ventilating current which *may* have come from a much cooler part of the mine. There is no inconsistency between my statement and Mr. Meyer's. I am quite aware that the circulation of air for long periods cools the rock walls of a mine materially. I say so in my paper; and I know that at great depths seasonal variations are slight. Messrs. Whitehouse and Wotherspoon's own figures show that this is so in the lowest levels of the Village Deep, even when the accumulated fan effect for some months is added to that of the winter season. Mr. Meyer considers "that the cooling effect of a good current of air passing through the mine workings is much underestimated by Mr. Moynihan." Why he should do so I do not know: especially in view of the fact that one of the principal purposes of the paper was to show that this cooling effect is immensely greater than anyone (Mr. Meyer included) has hitherto supposed. I say, in particular, in one part of my paper: "If a variation of 13° in the surface temperature at Arendia can and does produce 10° variation of drill-hole temperature in five months, it is clear that the constant circulation for years of air very much below the original temperature of the rock exposed will produce large temperature changes in the rock even at a great distance below its surface." I think I am justified in concluding that Mr. Meyer had given but little attention to what I have written. He winds up by asserting once more, without submitting proof, and in implied contradiction to figures which he himself gives, that the rock temperature gradient is even less than has hitherto been asserted. The minimum possible air-gradient makes that which Mr. Meyer gives a physical and absolute impossibility. In any case the observations are not to be relied on (on account of inherent observational errors) to within 10% to 20%; and his figure implies an accuracy of 1·4%.

Referring to Mr. Gibson's contribution, there appears to be nothing in it of importance which has not been dealt with in answering Mr. Meyer and Dr. Moir.

It is customary to speak of the rock temperature gradient on the Rand as if it were one definite thing—almost at all points. There is no evidence whatever in favour of this supposition, and a great deal against it. However roughly the mine figures on record may have been taken, the great differences in the rate of increase for different properties is probably not due entirely to erroneous records or differences in ventilation. The old notion that the heat in the earth's crust is mainly due to conduction from a central hot core of the globe has been exploded in recent years. We now know that the radio-active contents of the crust rocks are sufficient to account for at least a large part of the heat which the earth radiates into space. We also know that the percentage of radio-active contents varies in different rocks. The total thickness of the Witwatersrand series varies materially in different parts of the Rand Goldfields. The distance of the mine workings from the plutonic rocks underlying them, therefore, also

varies. These facts make it probable that the original rock temperature gradient varies considerably in different parts of the Rand; and the variations are enough to explain very large apparent and local discrepancies. They might also explain differences of gradients at different depths. The law of increase of temperature may not be a straight line at all. It is probable that it is not.

My general contention that the rock temperature gradient, as obtained from shallow bore-hole temperatures in working mines, is too flat is borne out by a statement in the *Encyclopædia Britannica* as to the gradient obtained from artesian well temperatures. No specific detailed list of temperatures is furnished, but the average gradient is given as being between 1° in 40 ft. and 1° in 55 ft. This is about double the gradient usually taken to be the average.

If there were any real effective desire to learn the true temperature gradient of the various parts of the Rand, it could be gratified at slight expense. There are a large number of surveyed diamond-drill boreholes on the field. Some of them are no longer of any ordinary mining use, are not in reef, and are some distance from working mines. If electrical thermometers were put into them, in the way suggested in my paper, we should then be able to get some reliable information on this important subject. The cementing up of the holes would eliminate any appreciable error due to water circulating in or through them.

The history of this question is not creditable to the Rand. It is in the interest of company promoters to show that the temperature gradient on the Rand is unusually low. Apparently favourable figures are therefore harped upon. Those technical men whose views find favour with the mining houses talk: and nearly all those whose views do not, think it well in their own interests to remain silent. The existence of such a state of affairs is quite enough to explain the lack of discussion on my paper, and the inanimate condition of the professedly scientific societies of the Rand. A Rand scientific society exists ostensibly to discover, discuss, and publish scientific truth. The atmosphere of the Reef is such that to this must be added, in effect, "truth that suits the Stock Exchange gambling hell."

Having now thoroughly realised this position, I have made up my mind not to belong to any of the technical or scientific societies of the Rand. I have therefore resigned from this Society.

The President: I do not think I can let Mr. Moynihan's concluding remarks pass without challenge. I do not know whether he accuses this Society of only publishing such truth as is acceptable to the Stock Exchange?

Mr. E. J. Moynihan: No. I say that it is the effective atmosphere of the place. It reacts on the scientific societies. I am not accusing this Society, as a Society, of doing anything of the kind. I want to make that quite clear.

GOLD AND PRICES.

(Read at September Meeting, 1914.)

By Prof. R. A. Lehfeldt, D.Sc.

DISCUSSION.

Mr. E. M. Weston *(Member):* The object of Prof. Lehfeldt's paper appears to be to prove that in a given year owing to the influence of gold production, prices rose $1\frac{1}{2}$% to 2%. In endeavouring to do so he takes a given year—1911—and for this year states the facts of the case in a mathematical formula. Differentiating this equation he considers he has proved his contention by the answer he presents. In my reply to discussion on a former paper I stated the above formula:—as increase of price =

$$\frac{\text{amount of gold} \times \text{amount of credit money} \times \text{its velocity}}{\text{amount of goods bought and sold} \times \text{their velocity}}$$

(*i.e.*, the number of times the same article changes hands.)

I do not profess any great proficiency in mathematics; but to rightly apply mathematics one should be certain, first, that all the factors entering into an equation are put down, and, secondly, that they are all given their right value. Only so can one hope (with right procedure) to arrive at the correct result. I will not dispute the form GFV = PT, but when I come to the values Dr. Lehfeldt assigns I have criticism to offer. Turning to No. 1, I would like to ask him if it be not a grave error to arrive at his value for T in the manner he does. Let him remember that he has selected a definite year —1911—and that he has tried to prove from definite figures that the world's stocks of gold were increased for that one particular year by 3%. Now, in effect what the author

writes regarding trade is this: "Statistics show that the trade of the world over a period (he does not mention over what period of years) is increasing at 3 to 3½ per annum." "It may, of course, in a year of good trade increase 5, or in a year of financial panics only 2 or less, but 3 is a fair average, and I will take that value for my particular year, 1911." Is this the way of obtaining a reliable factor for the equation of a given year worthy of a mathematician? I submit that if the author takes any one particular year he should be able to give definite figures for the increase of trade for that particular year, not for any average year, otherwise the equation is valueless. It is not right to take an assumed average value over a term of years and give that value to a particular year. This is especially so when he is working with such a small allowable margin of error as 1 to 1½. In other words, he wishes to prove that had the Rand produced only £950,000 to £1,450,000 less of gold in 1911 a rise of prices might have been avoided. One would surely be careful to have one's values checked very closely before presenting such a result, especially when the other factors involved are hundreds and thousands of million pounds. I am not prepared to challenge the statement that trade is increasing 3 to 3½ per annum. I should, however, like to ask for proof; as in my paper I stated that trade had increased 7 per annum since 1895. I have burnt my notes and cannot verify my source of information, which, however, I believe to have been a reliable one.

Values furnished under 2 by the author are equally founded on assumption, and are thus unreliable and unable to support his conclusions. He estimates the total stock of gold in the world at 12,410 tons (excluding India and the East). Here, again, I must protest. Surely in (1) India and the East are included in the total trade of the world and, as I showed in my paper, the increase of trade in India in recent years has been phenomenal. It has formed no small proportion of the total increase of the world's trade; yet Dr. Lehfeldt puts India on one side of his equation and omits it on the other in obtaining his 1 to 1½ result! Surely India alone counts for more than that? I already have pointed out that apart from the hoarding of gold in India, all the wheat trade of recent years has been conducted on a gold basis, and sovereigns are now in circulation in parts of India where a few years ago nothing but silver was seen. Regarding the total 12,410 tons of gold estimated to be in the world in coin one can only remark that this estimate may well be in error over 3½. In a former paper I gave some figures showing the enormous wastage in the past by fire, war, wear, flood, mischance on land, and by shipwreck on sea. The estimate of the gold output for 1911 is also subject to a large margin of error. Some gold sent to mints represents remelted old stock, on the other hand a certain proportion of the gold output of every country is stolen. Witness the small parcels some Indians have recently been prevented from taking to India. These sources of error, which are well known, should surely make one very cautious in depending on results representing 1 to 1½ margins. Again, in dealing with the disposal of the gold production of 1911 the author, while making allowance for the gold hoarded in India, makes no allowance for the fact that Egypt also absorbs gold to the amount of several million pounds every year. Here it seems to me is yet another source of error not considered. Examining the equation still further in this respect we find the author's result is admittedly based on conjecture. The author estimates that 172 tons were used in the arts. I think Dr. Lehfeldt will admit that this amount is purely an estimate which might be wrong 5 or more either way. The question naturally arises, when so many of the factors are unknown quantities why try to solve the equation?

It will be noted also that Dr. Lehfeldt assumes that all credit money has as its only basis gold. In my reply to my previous paper I stated that many economists held that such currency tended more and more to be based on the whole wealth and credit of the nation. It appears to me that the events of the last two months confirms this contention as regards Great Britain. It is true that during that time the Bank of England has added 20 to 30 million pounds to its gold reserves, besides that added to the central gold reserve; but Mr. Lloyd George apparently believes that the £1 and 10s. notes now accepted as legal currency in England have other backing than any gold reserve. They are scraps of paper—with the whole credit of the nation behind them!

Regarding the world's gold output, I stated in my paper that this must decline, and speculated on the effects of such a decline. It was possible (and is perhaps possible even

now) by rapidly expending 20 million pounds or more, to develop the mines of the East Rand so rapidly that for a time the output of the Rand might have been increased. I am afraid, however, this will not happen, and we may expect a somewhat rapid decline in the world's gold output.

With regard to the paper generally, I am afraid that Dr. Lehfeldt has quite unconsciously adjusted his figures to obtain the result he desired. The question of gold output will not of itself nor even to any preponderating extent explain the variations of index prices.

CHRISTMAS AND NEW YEAR'S GREETINGS.

The President: As this will be the last meeting before Christmas, I would like to wish the members a merry Christmas and a happy and prosperous New Year.

The meeting then terminated.

Contributions and Correspondence.

CHEMICALS AND STORES FROM GERMANY, ETC.

With reference to the remarks made by Mr. John Watson at the last meeting of the Society, the following notes may be found of interest.

Permanganate of potash, which for the last ten years has been exclusively imported from Austria and Germany, is one of the principal items. The consumption of this article by the mining groups is approximately 200 tons per annum, at an average cost of £50 per ton, landed on the mine. Apart from the Mining Industry, this commodity is used as a disinfectant by the general public.

Lead Acetate.—The average consumption of this substance per annum by the Mining Industry of the Rand and outlying districts is 700 tons. The present suppliers are Holland and France, and up to the outbreak of hostilities, Germany. Since the declaration of war, the United States have made a bold bid to secure this market. The pre-war cost to the mines from the first-mentioned sources averaged £35 per ton. American consignments are being sold at approximately £50 per ton. The worst features of dealing with the States are the slow and uncertain deliveries and the prohibitive price. The available amount of pig lead on the Continent is rapidly diminishing. In fact, acetate of lead produced at present is chiefly made from scrap lead.

There is a golden opportunity for British firms interested in the manufacture of lead products. At present British manufacturers seem to content themselves with the production of white and red lead, litharge, and lead-pipe fittings.

Glassware.—This industry was almost entirely in the hands of Germans and Belgians. The nearest quotations from British manufacturers are not within 300% of their Continental rivals. This may be due to the unparalleled conditions existing in England to-day. Even America is 200% above pre-war prices, and this probably for German goods. The quality of the foreign article is much superior to anything produced in Great Britain. The foregoing remarks apply equally to all porcelain material, of which a considerable quantity is used on the Rand.

Manganese Dioxide.—The average consumption of the above in the Transvaal is approximately 100 tons per annum. This is now being produced by the Cornwall mines at 100% higher price.

Fine Chemicals and Reagents.—This necessity to all chemical laboratories and assay offices, embracing chemically pure acids, of which a very considerable quantity is used on the Rand, was entirely in the hands of huge German firms, such as E. Merck, of Darmstadt. The production of these chemicals seems to have been entirely neglected by British firms, who have thereby missed an opportunity of building up a lucrative adjunct to any chemical manufactory.

Dyes.—An enormous turnover was done annually by Germany in all aniline dyes. Foremost amongst them was the extensively used laundry blue. This industry, although originally, I believe, discovered and matured by the Mother Country, has been through excise restrictions, driven to alien shores.

A. MACDONALD (Member).

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

PLATINUM ASSAY.—"Dewey says that separation of Ag from Pt with HNO_3 and Au from Pt with *aqua regia* is impracticable. Also that Cd does not prevent solution of Pt in assay button." F. P. DEWEY, *Mining Sci. Press*, 109, 20. *Chemical Abstracts*, Aug. 20, 1914, p. 2859. (J. G.)

INDEFINITE ATOMIC WEIGHTS. — "Two rather revolutionary scientific papers, both coming to the conclusion that the atomic weight of an element has not a definite value were presented by Dr. K. Fajans, of Karlsruhe, and by Prof. O. Honigschmid, of Prague, respectively, at the Leipzig meeting of the German Bunsen Society.

In his paper 'on the final products of radio active disintegration series' Dr. K. Fajans said that until recently it had been assumed that a chemical element has always the same atomic weight from whatever mineral it may have been isolated. But when a year ago the author and Soddy brought the radioactive elements in line with the periodic system, the hypothesis was suggested that there are elements which while entirely identical in chemical respects may differ in atomic weight up to eight units and which differ otherwise only by their radioactive properties (rays, speed of disintegration). This hypothesis was first based only on indirectly determined properties of radioactive elements of short life and it was therefore important to prove experimentally that two elements which appear to us to be identical chemically can have different atomic weights.

The author then concluded that this test could be made with lead. He considered it as probable that the atomic weight of lead formed in the course of millions of years in uranium minerals as the final product of the disintegration of uranium would be different from that of lead obtained from ordinary lead minerals.

This conclusion has been tested and has been fully confirmed by the author's former collaborator, M. Lembert, in the laboratory of Prof. Theo. W. Richards, in Harvard. Lead from uranium minerals was found to have the atomic weight 206·6, while ordinary lead has the atomic weight 207·1. The possible error is at most one or two units in the last decimal and is therefore much smaller than the difference found experimentally.

Of greatest interest is the fact that these two kinds of lead are not only chemically identical, but also identical in spectroscopic respect. On the basis of similar considerations it is probable that lead from thorium minerals has a higher atomic weight than ordinary lead. Possibly ordinary lead is but a mixture of lead from uranium and lead from thorium. The question comes up whether the atomic weights of most elements are perhaps only mean values of the atomic weights of several chemically identical elements which therefore cannot be separated by chemical means. Atomic weight researches have therefore to meet an entirely new problem: that is to investigate how the atomic weights of the elements depend on their origin.

The paper by Prof. O. Honigschmid on a 'revision of the atomic weight of uranium' gives the results of a determination by analysis of uranium bromide. His atomic weight of uranium is 238·18, that is 0·3 units below the present international value. He considers the value 226.4 for the atomic weight of radium to be improbable, since the difference between the atomic weights of uranium and radium must be at least 12, that is, 3 × 4 (atomic weight of helium). His own value Ra = 225·97 is therefore considered to be more probably correct.

At the same time the author determined, jointly with Stefanie Horovitz, the atomic weight of lead obtained from uranium ore and found 206·73, that is 0·4 less than the atomic weight of lead obtained from ordinary lead ore free from uranium."

Metallurgical and Chemical Engineering, Aug., 1914, p. 495. (G. H. S.)

THE REMOVAL OF IRON FROM WATER.—"The removal of iron from water by means of aeration is no new idea, but difficulties are often encountered in order to remove the oxide of iron from the water. The successful removal of iron from well water supplied by the Champaign and Urbana Water Company provides an interesting example which may be useful under similar conditions. The water contains two parts per 1,000,000 of iron, 40 parts of carbonic acid gas, and no dissolved oxygen, and is pumped from a fine sand stratum. Long-standing trouble from crenothrix and stained porcelain led to experiments which proved that the iron could be removed by means of aeration and sand filtration without the use of chemicals. The water is aerated as it falls from the pumping main into a basin. From this basin the water is again pumped into the filters, rising as a jet and falling vertically. It is thus aerated. The filters are ordinary sand filters arranged on the principle of the mechanical filter with air and strainer systems for washing. The filters consist of 30 in. depth of sand and 8 in. of gravel. Each filter, being 12 ft. × 15 ft. in plan, deals with 500,000 gallons a day. After about six weeks of service, the sand grains became so badly coated that the ordinary washing arrangements were found to be inadequate. This difficulty has, however, been overcome by pumping the sand through a centrifugal pump, which cleans it satisfactorily. It is expected that the process will be perfected to such a degree that the sand of a filter may be washed by running it through a centrifugal pump, and then returning it to its place directly, without using much more water than is required for the ordinary daily wash of the filter. The sand of the beds has been washed in this manner twice in eight months. The occasional use of a fire hose in stirring and washing the sand in the beds has also been found to be effective, the amount of water used being about the same as that required for the ordinary daily wash. Experimental work carried out in order to ascertain whether the application of chemicals to the water would lessen the difficulties of washing has shown no advantage to accrue from this. It is, perhaps, remarkable that for such a purpose as this the fixed spray or rotary sprinkler, similar to those used for sewage filters, is not applied more frequently. It is certain that the aeration would be more perfect; the method would, however, be more applicable to slow sand filter beds than to the rapid mechanical filters. The method of washing the sand at the Urbana works by passing it through a centrifugal pump is certainly interesting, and may have valuable applications."—*Indian Engineering*, Aug. 29, 1914, p. 121. (J. A. W.)

DETECTION OF PLATINUM BY MEANS OF STANNOUS CHLORIDE.—"Stannous chloride gives a blood-red colouration (gold brown in very dilute solutions) with solutions of platinum chloride, and the colouring matter can be extracted with ether (Wöhler, *Chem.-Zeit.*, 1907, 938). The test is very sensitive and is not given by iridium, palladium, gold or iron. Humic substances, however, and filter paper give similar reactions after treatment with *aqua regia*, and the test is only conclusive of platinum, when the filter paper has been completely incinerated and when any humic substances present have been destroyed (*e.g.*, by fusion with pyrosulphate) prior to extraction with *aqua regia*."—E. LANGSTEIN and P. H. PRAUSNITZ, *Chem. Zeit.*, 1914, *38*, 802.—*Journal of the Society of Chemical Industry*, July 31, 1914, p. 769. (J. W.)

METALLURGY.

ORE TREATMENT AT THE ARGO MILL. "Rejection of the sand after concentration, the use of an electric cell to compensate the effects of reducing agents, and the addition of ammonium salts, are the three distinctive features which have helped make treatment possible of refractory low-grade sulphides of this district. Aside from the possible elimination of amalgamation, or change in the manner of practising it, a successful system is established in the Argo mill, and any further changes will be small details governed by convenience and not by a distinctive metallurgical change.

The ores treated are complex sulphides concentrating 4·9 tons into 1. The average heads for the first six months' period assayed 0·25 oz. gold; 3·01 oz. silver; 0·60% copper; 0·75% lead. The construction of the plant throughout is of steel and concrete, with the exception of wood for the ore bins.

Crushing and Sampling.—Running along the bottom of the 22 receiving bins, is a 24 in. Jeffreys pan conveyor, which delivers the ore over a 1½ in. grizzly into a No. 4 Gates gyratory crusher, dropping by gravity through three Snyder samplers, each taking out a 20% cut. Between each sampler the ore is recrushed, once by a Eureka crusher and once by a set of rolls. The final sample is sent to the bucking room, where four pulps for assay are used. Twenty per cent. of the ore is retained for resample until final settlement is made with the shipper. The balance of the ore goes to two 50 ft. elevators and is discharged onto a 24 in. conveyer belt, with reversible tripper, distributing in the 1,600 ton storage bin at the top of the mill. The elevators are of 14 in. Jeffrey belt with 5 in. × 7 in. malleable buckets set staggered, 7 in. centres. So far this arrangement has proved satisfactory.

The sampling plant uses four men on shift, and handles from 100 tons to 200 tons per nine hours, according to the physical condition of the ore. Power is furnished by three electric motors using a total of 35·5 h.p. Total costs in the sampling department amount to from 20c. to 25c. per ton of crude ore.

Stamping and Concentrating.—From the 1,600 ton bin, improved Challenge feeders deliver this ore to twenty 1,050 lb. stamps, in two batteries of 10 each, making 110 drops per minute through 6 in. Stamp crushing is done in ½ lb. cyanide solution, using from 5 tons to 6 tons of solution to 1 ton of ore, with a cyanide consumption of ½ lb. per ton of ore. Twelve mesh woven-wire screens are used on the batteries, the stamp duty being over 6 tons per stamp-day, or a total of ore 120 tons per 24 hours. The pulp from the stamps flows over four 48 in. × 54 in. copper plates. So far amalgamation in cyanide solution has not proven the success that was expected of it. The reasons are discussed later.

After passing the plates the pulp is led to a duplex Dorr classifier, where the overflow slime is removed, running to a 4 ft. × 4 ft. spitzkasten, followed by a 14 ft. × 22 ft. Dorr thickener, while the sand from the classifier is dewatered and delivered to a distributing barrel for table concentration. Solution titrating about ½ lb. cyanide is here mixed with the sand to give it the right consistency for proper table work.

There are five standard Card tables now used for the first sand concentration, with provision for installing a sixth table on the same floor. Two finished products are cut out on these tables, a lead and iron concentrate. The middling flows by gravity to a 4 ft. × 10 ft. Stearns-Roger tube mill and a half-size simplex Dorr classifier, operated in a closed circuit. This is made possible without pumping by an extra large scoop for feeding the tube mill.

Sand from the middling table joins the sand tail from the first five tables and flows by gravity to a three-compartment Dorr dewatering and washing machine. The pregnant solution with the small percentage of colloidal slime contained with the sand is removed in the first compartment. This product flows into a 12 ft. × 20 ft. Dorr thickener, the clear overflow from which is pumped back to the head of the mill. The thickened underflow from it is lifted by a diaphragm pump into No. 1 agitator, a 12 ft. × 20 ft. Dorr machine. In the second and third compartments, respectively, of the triplex Dorr classifier, a barren solution and a water wash is given the sand, which is then discharged at 15% moisture, carrying a trace in KCN and dissolved values, and run to waste as tailing, thus eliminating on an average of 56% of the mill tonnage without further treatment. Under normal conditions the gold extraction in the sand averages 90% and the silver well above 85%. The character and grade of the ore treated would make further handling a losing commercial treatment in a mill of this size.

Treatment of Slime.—A ¾ in. spigot underflow from the spitzkasten is concentrated on two Card slime tables, placed on the same floor as the middling table. A clean concentrate is here cut out and shipped as a copper product, thus eliminating a large percentage of the copper before agitation and contact with the stronger solutions. This spigot underflow also removes 90% of the +200 mesh product from the slime entering the spitzkasten. The whole remaining product from the above two tables gravitates to the tube mill circuit, together with the middlings from the middling table. The slime from this classifier has a screen analysis as follows:—

	Per cent.
+150	1·0
+200	5·0
−200	94·0
Total	100·0

It flows to two 10 ft. Dorr thickeners operated in tandem. The thickened overflow from these vats flows to the Dorr agitator. The clear overflow can either be sent to precipitation *via* the clarifying filter, a simplex excelsior filter, or may be pumped directly to the head of the mill.

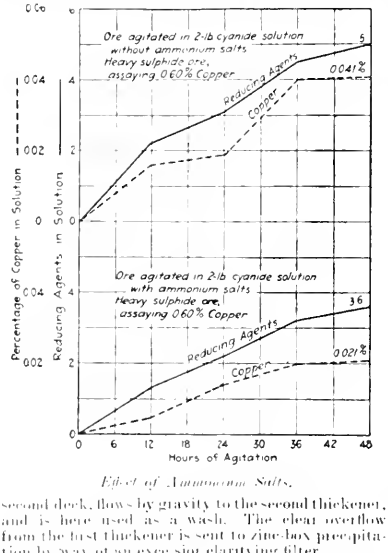
The overflow from the 14 ft. × 22 ft. Dorr thickener is handled through the same channels as the overflow from to 10 ft. thickeners. The underflow, which is thickened to about 34% solids, and has the screen analysis given in the following table, goes to the Dorr agitator.

	Per cent.
+ 80	Trace
+100	0·5
+150	1·75
+200	4·75
−200	92·5
Total	99·50

The combined pulp entering for agitation comprises about 26% of the mill tonnage. The underflow from the 10 ft. Dorr thickeners is considerably more dilute than the underflow from the other thickener, and will thin the total pulp agitated to 2½ of solution to 1 of dry slime. This is made up to 1½ lb. cyanide and 0·6 lb. lime per ton of solution in the No. 1 or Dorr agitator, which is connected in continuous agitation with a 12 ft × 20 ft. Parral agitator. The cyanide consumption passing these two agitators is about ½ lb. per ton of slime.

The overflow from the Parral vat passes to Dorr No. 1, the first of a series of three 12 ft. × 20 ft. thickeners, arranged for continuous decantation, the underflow pulp being passed on Goulds diaphragm pumps, while the returning clear overflows run by gravity.

The water used for washing the sand on the third deck of the triple Dorr classifier is raised 2 ft. by an air lift, and used as a wash for the last thickener. The barren solution used to wash the sand on the



Effect of Ammonium Salts.

second deck, flows by gravity to the second thickener, and is here used as a wash. The clear overflow from the last thickener is sent to zinc-box precipitation by way of an excelsior clarifying filter.

Under first arrangements, the underflow from the last Dorr thickener was concentrated on two Card slime tables, making a clean mineral, and the balance sent to the slime filter for final treatment. But this arrangement proved useless, as the amount and value of the mineral made was not worth considering. Therefore, the thickened underflow, 35 to 40 solids, is pumped by a diaphragm pump direct to 7½ ft. × 12 ft. Portland continuous-type drum filter, which is given a barren solution and a water wash, and from which the pregnant solution is combined with the barren solution wash by both being pulled through a 12 in. 10 in Gould vacuum pump, and then returned to the head of the mill or passed first through an available zinc box if desired. The water wash is passed through the filter by a 6 in. 6 in. triplex pump and used as a wash for the last Dorr thickener. The discharged slime cake, containing 0·20 lb. KCN and 10c. in dissolved values, falls into a launder and is sluiced into the creek.

Precipitation is by zinc shavings, both weak and strong solutions being precipitated in the usual way. The boxes are cleaned up every two weeks. Anything not passing a 30 mesh screen is put back. After drying, the precipitation is smelted direct without acid treatment. A No. 60 crucible in a Case oil tilting furnace is used.

Heating the Solution.—The solution throughout the mill are warmed by a 70 h.p. boiler, under 40 lb. pressure, from which lead two lines, the first going through radiators on the battery floor, thence through coils in the 14 ft. × 22 ft. Dorr thickener and discharging the condensed water into the last compartment of the sand tailing classifier, and the second a line going through coils in the three thickeners following agitation, discharging into the last. No other heat is used in the mill, except one line of steam going to radiators in the superintendent's office, a radiator at the solution man's titrating desk and radiators in the bucking room of the sampling department. An appreciable increase in extractions is shown by keeping the solutions above 60° F.

Lime is added at three places in the mill: the amount is varied and is regulated according to the ore treated. A small amount of slaked lime is first added at the batteries with the feed solution. Over the 14 ft. × 22 ft. thickener a continuous lime feed is made from a common whisky barrel, mounted on a shaft, and belt driven, filled with crushed lime. The barrel makes 6 r.p.m. and the feed is regulated by adjusting a slide over a small rectangular opening. A few small tube mill pebbles are placed in the barrel, keeping the discharge opening free. The balance of the lime is fed by slaking it in the water used to wash the last thickener.

As already mentioned, enough cyanide is added at the Dorr agitator to bring the solution up to the desired strength. Just enough cyanide is added at the tube mill to keep the weak solution at ½ lb. per ton at zinc box head.

The ammonium salts are added at the Dorr agitator, as experience has shown the zinc boxes will be kept free from precipitated copper by so doing, and the salts have the effect of increasing the extraction. The accompanying curves, Fig. 2, show the experimental results of the use of ammonium salts on the copper and reducing agents.

Electrolysis of Solution.—An interesting feature of the treatment is the use of electrolytic oxygen to overcome the effect of reducing agents. It is according to a modification of the method advocated by T. H. Aldrich, Jr., of Birmingham, Ala. The accompanying curve, Fig. 3 gives the reducing power of the solution with and without the cell installed. A hard rubber cell, containing fifty 4 in. × 5 in. lead plates ½ in. in thickness and ¾ in. centres, is placed in the head compartment of each zinc box. A 110 volt alternating current is used, and the cost of this current is less than 1c. per ton of crude ore milled.

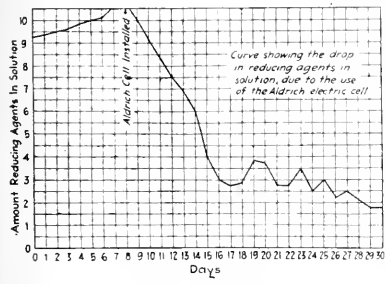
Lead salts in varying quantities have been used to offset the reducing agents, and while they had the desired effect, it was at a cost far greater than the Aldrich electric cell. Litharge was also tried in the tube mill and agitators as a possible aid to silver extraction, but showed absolutely no assistance. Precipitation, however, is materially assisted by dipping the zinc in a weak solution of lead acetate.

Handling Concentrates.—The three concentrate bins, one for each product, are placed on the lower floor so located that the concentrate can be weighed and wheeled into railway cars for shipment. Each class of concentrate from the upper floors gravitate to a one-fourth size simplex Dorr classifier, placed over the bins and used as a dewaterer. The concentrate are discharged from the classifiers, averaging 12 moisture, on a sloping cement floor, allowing further solution to drain off to the general mill sump. The concentrate is sold under contract to the Globe plant of the American Smelting and Refining Company, at Denver.

The mill costs per ton of ore treated during 1913 operations, when the mill never exceeded two-thirds of its capacity, owing to the lack of ore receipts, are as follows:—

	Per Ton Ore Milled.
Office and overhead expense ...	$0.138
Milling	1.092
Marketing products	0.802
Sampling	0.225
Total	$2.257

These figures cover all repairs and up-keep, but do not cover depreciation.



Effect of Electrolysis.

Operations during 1913 have shown mill extractions of 83% of the gold, 63% of the silver, 80% of the lead and 40% of the copper. The consumption of cyanide was 1·52 lb. per ton of ore treated, and lime, 5·09 lb. per ton."—A. H. Roller and H. T. Curran. —*Engineering and Mining Journal*, July 4, 1914, p. 1. (G. H. S.)

MINING.

Correct Methods of Priming Dynamite.—"A great many blasters think that the priming of a dynamite cartridge is of very little importance—if the dynamite goes off, that is all there is to it. It is not widely known that the success or failure of a good many blasts depends on the position of the detonator—blasting cap or electric blasting cap—in the dynamite. Not only the amount of work done depends in a large measure on the priming but also the safety of the loading operation is affected in many instances by the position that this detonator occupies. Many accidents have occurred and many men have been killed in loading holes with a primer made so that the detonator became jambed or wedged in the hole and exploded when the blaster endeavoured to force it down.

The points to be borne in mind in making a primer are the following:

The detonator, and by detonator is meant either a blasting cap or electric blasting cap, must be in such a position that it cannot become jambed in the bore hole, as in Fig. 1.

The detonator must be in such a position that it will not be struck by the tamping stick.

The detonator must be fastened to the cartridge in such a way that when the wires or fuse are held in one hand by the blaster, and the cartridge seated with a tamping stick held in the other hand, the detonator will not pull out.

The detonator must be entirely embedded in the dynamite.

The loaded end of the detonator, or the closed end, or the 'business end' should point toward the bulk of the charge (Fig. 2-A), not away from it (Fig. 2-B). The detonator should not be secured by metal wires, etc.

With an electric blasting cap, the wires should not be fastened on with a half hitch, as in Fig. 3, for although this practice is very common, there is always a chance of a short circuit occurring, especially when high-voltage power currents are used for firing.

With blasting cap and fuse, the fuse must not come in contact with the dynamite except for about a half inch next to the cap (Fig. 4). 'Lacing' fuse through dynamite causes more burned charges than anything else (Fig. 5).

The maximum detonating effect of a blasting cap or electric blasting cap is out from the closed end. If you place a dozen cartridges of dynamite, end to end in a straight line on the ground, and fire a blasting cap in the end cartridge, with the blasting cap pointing towards all the other cartridges (Fig. 6), the entire line will go off, but if you put this primer in the middle of such a line (Fig. 7), some of the cartridges back of the detonator, that is, those away from which the detonator is pointed, will often fail to explode. If you put the primer cartridge at the end of such a line, pointing it away from the line, that is, pointing it towards the open air (Fig. 8), generally the primer only will explode. This does not always happen in a bore hole, because

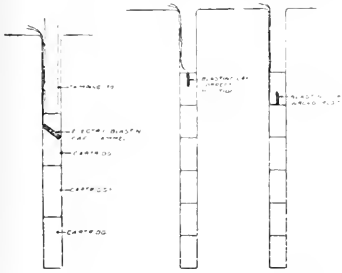


Fig. 1. Fig. 2. Fig. 2-B.

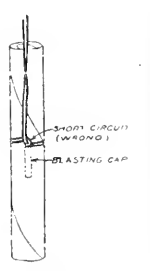


Fig. 3.

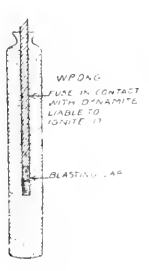


Fig. 4.

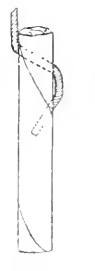
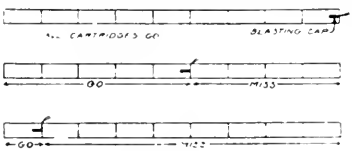


Fig. 5.

there the explosive wave is strongly confined, but this shows a tendency of the dynamite, which is in the opposite direction from the closed or 'business end' of the detonator, to misfire. Several years ago this was demonstrated to a sceptical individual in a quarry shot where there were 27 holes, each hole containing from 19 to 21 cartridges of dynamite; 13 of the holes having the detonator in the next to the last cartridge and pointing downwards, that is, toward the bulk of the charge; the rest of the holes having the detonator in the same cartridge, that is, the next to the last, but reversed, pointing upwards. These holes were all thoroughly tamped with sand and fired at the same time. While all the dynamite in both sides exploded, where the detonator pointed downwards the rock was broken and cleared from the ledge below the bottom of the holes; where the detonator was pointed upwards, about one-third of the distance from the bottom of the holes upwards, the rock was broken up but left in place. This showed that when the detonator pointed downwards, the full power of the dynamite was realized; where the detonator pointed away from the bulk of the charge, the full power was not realized, but a great deal was wasted.

When a cartridge of 40% straight dynamite is suspended vertically in the air, and a blasting cap



Figs. 6, 7 and 8.

supported at right angles to it, and pointing directly at it (Fig. 9) is fired, the dynamite will explode at a distance of 24 in. from the blasting cap, but if a cartridge and blasting cap are hung parallel with each other, that is, the blasting cap pointing either straight up or straight down and parallel to the dynamite cartridge, so that it does not shoot directly at the cartridge (Fig. 10), the dynamite will not explode until it is brought to a distance of 15 in. or less. When the blasting cap is horizontal and the cartridge of dynamite hung vertically and the cap supported at the level of the middle of the dynamite cartridge but pointed so that it does not point directly at the dynamite but a little to one side (Fig. 11), this distance where an explosion occurs is lessened still more. When the cap is pointed directly away from the dynamite cartridge (Fig. 12), it will not explode the dynamite cartridge every time even when within 1 in. of it.

The ideal primer with an electric blasting cap would be one where the detonator lies directly in

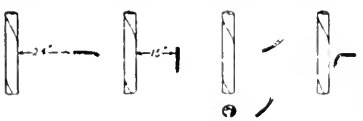


Fig. 9. Fig. 10. Fig. 11. Fig. 12

the axis of the cartridge near the middle, and with the wires secured to the cartridge, in such a way that they could not possibly pull out, by some strong non-conducting material like string. By using a curved puncher (Fig. 13) it is possible to get an electric blasting cap almost in this ideal position, where it may be held in place securely by taking a half hitch with a piece of string around the wires, is then carried once or twice around the dynamite cartridge and tied tightly with a square knot (Fig. 14).

Another way of priming with an electric blasting cap without using string is to punch a hole through the cartridge about 2 in. from the end and lace the electric blasting cap through this and insert it in the side of the cartridge.

With short wires, 4 ft. or less, the ends may be passed through this hole after the cap is inserted in the cartridge (Fig. 15), but with long wires it is apt to either make the hole so large that it does not make a secure fastening, or it may even cut the cartridge in two.

With blasting cap and fuse, inserting the blasting cap in the side of the cartridge about one-third from the top and tying it tightly with one or more pieces of string, keeps the cap away from the tamping stick, and if carefully done with a curved puncher, places the cap almost in the axis of the cartridge (Fig. 16).

The orthodox method of opening up the paper at the top of the cartridge, punching a hole in the dynamite, inserting the cap and fuse into that, and then tying the paper down around the fuse as in Fig. 17, gets the blasting cap in a good position but

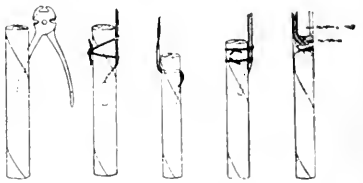


Fig. 13. Fig. 14. Fig. 15. Fig. 16. Fig. 17.

is a difficult operation to perform on account of the tendency of the string to slip off the paper and the difficulty of getting the string tight enough to prevent the fuse from pulling out. It also is liable to kink the fuse abruptly if the primer is seated firmly with the tamping stick.

Dynamite when strongly confined in a bore hole burns very rapidly when it is ignited, and on this account the lacing of the fuse through the dynamite cartridge is a practice which is strongly condemned by all explosive manufacturers. It seems strange that very much of the dynamite would burn before the cap would explode, but the fact is, dynamite burns very much more rapidly when confined than when in the open, and it has been known to burn up entirely in a bore hole which contained a perfectly good cap without any of the dynamite exploding, the cap of course exploding and doing no work. Burning dynamite makes a large quantity of exceedingly poisonous fumes, and a number of fatal accidents have occurred from dynamite burning up in the bore holes in mines and tunnels, due either to lacing the fuse through the cartridges, or from other causes, such as matches, sparks or burning oil dropping into the charged bore holes. These fumes from burning dynamite are so frightfully poisonous

that no chance whatever should be taken of a charge catching fire, especially when it is so easy to prevent it by taking the proper care in preparing the primer.

In priming holes where it is necessary to have two primers, one should be near the top, pointing downwards and one in about the middle, also pointing downwards. Where three primers are used, place one at the top, one about one-third way down and one about two-thirds way down; that is, equally distant from each other and from the bottom of the hole.

The bane of blasters in mines and tunnels is the smoke and fumes from the explosive, and while no explosive has yet been invented which does not give off more or less irritating fumes, the quantity of fumes given off can be materially decreased by insuring complete detonation of the charge by using the strongest obtainable blasting cap or electric blasting cap, by inserting the primer in either the last or next to the last cartridge in the hole, and placing the detonator so that the 'business end' points towards the bottom of the hole, and by using an adequate amount of good tamping material on top of the explosive charge. The reason why it is advisable to place one cartridge on top of the primer is that it serves to cushion the primer from the tamping stick when used for packing the tamping. The stronger the confinement and the larger the detonator, the more perfect the explosion will be, and the less amount of noxious fumes will be given off."—*Mining and Engineering World*, July 25, 1914, Vol. xli., No. 4, p. 149. (J. E. T.)

MISCELLANEOUS.

German Synthetic Drugs and the Patent Laws.—"Herman A. Metz, president of the Farbwerke-Hoechst Co., interviewed by the *American Druggist* regarding the manufacture of synthetic drugs in the United States, said:—'No, I do not think there is any possibility of engaging in the manufacture of these products here, even if there were patent laws to compel the carrying out of patents in this country and manufacturing goods protected under such patents. England has tried this, with the result that Germans established plants in England, where they carried out just enough of the process to cover the patent, depending upon Germany for all raw materials and intermediate products. The result is that, in spite of its laws, England is stranded high and dry. Only recently I received a cable from the Manchester branch of the German plant, asking if we could send any salvar-san to Europe for the English Government. . . . . It has taken half a century for Germany to reach its present place in the chemical industry. England originated it, and Germany depended upon England for a large part of its raw materials; then France took it up, but to-day France and England both are negligible factors in the industry as a whole. The net result of the British patent limitation has only benefited Great Britain to the extent that it has brought two or three plants in the chemical line to England, employing only a small amount of labour, because the chemical industry requires less labour in proportion to the value of the product turned out than any other industry. They obtain all their raw materials from Germany, and compel the English public to pay, in addition to the cost of the drug or dye in Germany, the cost of amortisation of the plant in England during the life of the patent."—A. Metz.—*Chemist and Druggist*, Sept. 26, 1914, p. 85. (J. A. W.)

Permeability Tests for Paints.—"In a report under the above title to the American Society for Testing Materials, the various methods of testing and various comparative standards of 'weight' and 'volume permeability' in the case of paints and varnishes spread on surfaces of wood veneer, cement, cloth, wire mesh and paper are discussed.

As to his definitions, 'weight permeability' is the weight of moisture vapour passing through a coat of paint or varnish in 24 hours divided by the weight of the coat itself. 'Volume permeability' is the weight of water vapour passing through the coat in 24 hours divided by the volume of the coat when dry on the given surface; it is easily deducible from the 'weight permeability' by dividing the latter by the specific gravity of the dried coating.

The tests are made by coating a disc of one of the above-named materials about 7½ in. diameter with the material to be tested, and the average weight of this coat after the siccatives, etc., have dried out is usually about 10 gm. Above this screen is a weighted dish of calcium chloride, and the screen is subjected to artificial rain and sunlight by means of a rotating garden spray and a mercury vapour lamp of high candle-power at short range. The coat is subjected to these special weathering tests in a cylinder of about 7 ft. in height and 4 ft. in diameter; descriptions of the tests are carefully given, and the results show unexpectedly large contrasts between the various materials used. Cements often broke down in two days, implement paints in three, varnishes in five days, and some outdoor paints could be examined for 100 days or more.

Heat has a great effect on permeability and osmosis takes place more readily towards the outside air than the reverse way.

Porosity is to be in no way confounded with permeability. The experiments are readily modified to discover the permeability of various coatings to obnoxious gases, acids and acid fumes, etc.

As regards moisture they are in very satisfactory agreement with the feeling of long experience with the more commonly known and used materials experimented with."—A. M. Muckenfuss. *The Chemical World*, Sept., 1914, p. 253. (J. A. W.)

The Cohesion Point, a New Constant of Glass.—"When two clean plane polished glass surfaces are pressed together a firmly adhering combination is obtained, in which however the surfaces are separated by an air layer at least 0·047 mm. thick, and a greyish blue interference colour is produced. The condition is merely one of *adhesion*, the intimate molecular contact of the glass being wanting which is the necessary condition for *cohesion* between the two surfaces. At high temperatures the adhesion slowly passes within a certain length of time into cohesion. The temperature required to bring this about in a specified time is named the *cohesion-point*. It is a constant of the particular variety of glass, which relatively defines its specific softening properties, and allows the course of liquefaction to be exactly followed. It is, in fact, a substitute for the melting point constant of crystalline substances. If cohesion points are plotted against corresponding times of heating a representation of the softening of the heated glass is obtained. In practice 30 minutes may be adopted as the standard time of heating required to produce coherence of the surfaces. The glass, in the form of dics, 10 mm. diameter and 2 mm. thick, adhering in pairs, is heated in an electric tube furnace (see Zschimmer and Schiele, *Silikat-Zeits.*, 1914, *2*, 45). The course of the coherence is easily

followed; a black spot surrounded by Newton rings first appears, and gradually increases in size, eventually at the completion of coherence extending over the whole disc. The end-point is easily recognised. Numerous tables and curves are given showing the values of the constant for many varieties of glass." —E. ZSCHIMMER, *Silikat-Zeits.*, 1914, 2, 129-135.—*Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 921. (J. A. W.)

EFFECT OF DUST CAUSING LUNG DISEASES.—"A study by Collis of the effects of dust in producing diseases of the lungs has led to the following conclusions:—(1) Inhalation of all forms of dust is accompanied by diminished power of chest expansion, with correspondingly high blood-pressure. (2) Animal dusts, apart from the presence of pathogenic micro-organisms, produce less effect than vegetable or mineral dusts. (3) Vegetable dusts tend to produce an asthmatic type of chest affection. (4) Mineral dusts which do not contain free silica tend to produce irritation of the upper air passages and respiratory diseases other than phthisis, those composed of calcium salts being least injurious. (5) Inhalation of mineral dusts which contain free silica is associated with an excess of phthisis; an excess which bears a direct relation to the amount of free silica present. (6) In general, dusts appear to be more injurious as their chemical composition differs from that of the human body, or from the elements of which the body is composed."—Annual Report of Chief Inspector of Factories.—*Journal of the Society of Chemical Industry*, Sept. 15, 1914, p. 896. (J. A. W.)

Abstracts of Patent Applications.

(C.) 201/14. George Moore. Improvements in processes for treating metal bearing material. 14.5.14.

This application refers to a process for treating metal bearing material, consisting of sand and slime, and comprises moving a compact column of such material downwardly while moving a fluid upwardly in contact with the material, and retarding the movement to prevent agitation and separation of the sand and slime. An apparatus is described for the purpose of carrying out the work.

(C.) 322/14. Henry Moore Sutton (1), Walter Livingston Steele (2), Edwin Goodwin Steele (3). Improvements in processes of and apparatus for sizing or classifying comminuted material. 16.7.14.

This application relates to improvements in process and apparatus for sizing or classifying comminuted material. The inventor claims to be able to classify according to size, regardless of any difference in specific gravity, without screens of any kind. This constitutes the process claim.

The apparatus consists of a series of adjustable deck frames, each frame being so mounted that the inclination thereof can be adjusted independently.

Each deck frame is provided with spaced rollers, around which pass endless flexible belts, having roughened or corrugated surfaces, which latter are progressively increased in size from the upper to the lower edge in order to progressively increase the resistance to the material.

Driving mechanism is attached to the spaced rollers, which impart movement to the endless belts.

Arranged below the belts are graduated tapering angular rollers, which impart an accelerated movement to the crushed material passing over the sizing surface of the belt. These rollers are controlled independently from the driving mechanism of the endless belts, their effect being to create varying vibrations and undulations of the sizing surface, which, acting in conjunction with the cross channels resulting from the corrugations of the belt surface, are intended to give the classifying effect claimed by the inventor.

(C.) 354/14. Orenstein-Arthur Koppel, Ltd. (1), Martin Christensen (2). Improvements in tipping trucks and the like. 19.8.14.

This patent claims improvements in the brackets supporting the pins in "two-pin" side tipping trucks. The usual arrangement is to have a bracket of flat bar bent to receive the pin and rivetted to the side of the end standard of the truck, and the improvement claimed is to carry down the end of this bracket to take a rivet in the foot of the standard.

(C.) 380/14. Mather Williams Sherwood. Power drilling machines. 8.9.14.

This application embodies 10 claims, and in general the claims refer to the following:—

In a drilling machine, the combination, with the drilling engine, of means for effecting the rotation of the drilling tool, including a fluid pressure actuated motor operatively connected with the tool, the exhaust of the motor communicating with the inlet of the engine for operating the latter.

(C.) 381/14. Charles Herman Haeseler. Power drilling machines. 8.9.14.

This application embodies 7 claims, and in general the claims refer to the following:—

In a drilling machine, the combination of a drilling engine including a cylinder, a hammering piston, reciprocable therein, a rotatable tool holder at the front end of the cylinder, a back head for the cylinder, a rotary motor mounted in the back head and operating connections between the motor and tool holder.

(C.) 388/14. James Mineis Holman (1), John Leonard Holman (2). Improvements in or relating to pumps. 26.9.14.

This application comprises a fluid operated pump, preferably of two cylinders, specially designed to be clamped to a single stretcher bar, such as is used with rock drilling machines.

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PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M.

This paper is submitted for discussion, both to this Society and to the Institution of Mining and Metallurgy, according to the arrangement existing between the two bodies.

PREFACE.

The Subject. Mining is an art to which science is applied with a view to economic result. Science is based upon ascertained fact and inductive reasoning therefrom. Any perversion of fact or any acceptance of untruth vitiates science, which then becomes non-science, commonly identified with nonsense. As mining is based upon the application of scientific principles, it is the aim of the intelligent miner to eliminate error, more especially the veiled falsity that masquerades in the guise of an attractive generalization. Of all such generalizations the most attractive, and, if untrue, the most dangerous, is the one that asserts the indefinite continuance of ore in depth.

This subject is of vital importance to the profession of mining engineering. If it be true demonstrably that veins are not impoverished vertically, then, an outcrop having been found, it is bad practice to sink shallow shafts and peck at an ore deposit by a succession of workings from levels only one hundred feet apart. If the ore can be assumed confidently to continue for thousands of feet below the surface, it is a blunder in technique to exploit such ore timidly and on a small scale; it is as incorrect as the extraction of ore by underhand stoping from a winze.

On the contrary, if a mining engineer be justified in assuming an indefinite downward prolongation of his ore deposit, within the walls of his vein, he should sink his shaft bravely to four or five thousand feet and develop his ore reserve boldly, as, for instance, has been done in the case of the deep-level mines of the Rand. The extent of development is then limited only by the capital available and the time required to redeem it. The economic unit of size in the mill, and, therefore, in the tonnage of ore developed, is proportioned to the capital involved. Finally, the amount of capital it may be expedient to expend on a certainty is dependent upon the value of money, that is, the current rate of interest.

In more senses than one, the persistence of ore is a basic problem in mining. Yet it is one either carelessly ignored or perversely misinterpreted. If I venture to urge upon you the consideration of the subject it is not without justification, for I have made it a matter of inquiry for over 20 years. When my attention was first drawn to it, the idea of persistence assumed an even more fallacious form. At that time it was commonly asserted that ore-bearing veins became richer as they were followed downward. Deep sinking used to be advocated as a practical method of improving the prospects of an impoverished mine. Such advice was frequently the last resort of a perplexed engineer, but as often it was the first resort of a charlatan.

In 1892 I cut the following paragraph out of a prospectus appearing in *The Illustrated London News:*

"The mine has been inspected and reported upon by several eminent engineers, who have declared that the quartz lodes are true fissure-veins; and in true fissure-veins it is well known that there is no fear of giving out or not paying as depth is reached, the experience being that they get richer the farther they go down."

Such statements were almost stereotyped in those days. This particular report referred to the Phœnix mine, in Otago, New

Zealand. I had examined the mine a year earlier, but, needless to say, I was not one of the "eminent" engineers willing to prophesy such smooth things. In that late Victorian period the notion still lingered that what was termed the 'true fissure-vein' had a peculiar sanctity; it was a privileged kind of ore deposit that penetrated the very core of the earth and continued in bonanza to depths far beyond the reach of man. The current theory was that the ore in such veins was deposited by hot solutions welling upward from the earth's interior; and from this theory the inference was drawn that the nearer the mine workings approached the source of the ore the richer that ore necessarily became. This was as logical as the assumption that the quantity of water in a river must increase as it is followed to its source.

Discussion of the subject in print has been meagre. Even those who questioned current opinion were slow to express themselves in technical publications. One notable exception was an article by William P. Blake in the *Engineering and Mining Journal* of January 7, 1893. It was entitled 'The Persistence of Ore in Lodes in Depth.' In this article the late Professor Blake confessed that "the tendency of the theories of lateral secretion seems to be to shake our faith in the full mineralization of lodes at great depth," and he expressed himself as satisfied if "it can be shown that lodes can be expected to maintain their average value at any depths we are ever likely to reach in mining operations." He undertook to prove this, and cited sundry examples. On the invitation of the editor, Richard P. Rothwell, I controverted this opinion, demurred to much of the evidence, and concluded my reply to the professor's article by saying: "Let us not bend facts to theories, but rather bring our theories in accord with the facts. When we do so we shall, I believe, find that ores do not, as a rule, persist in depth, much less grow richer, but we shall recognize that the exceptions to this general experience are fortunately numerous." That was written 21 years ago; reading it now, the statement seems to me to err on the optimistic side. It all depends upon the interpretation of "numerous."

English opinion on the matter is expressed, for example, by J. Arthur Phillips, whose book on ore deposits was for a long time the best in our language. Referring to the belief of "the earlier miners in California" that "the outcrops were more productive than the deeper portions of the same veins" he proceeds* to summarize the matter thus: "Within the last thirty years, however, experience has entirely changed their opinion upon this subject, as deeper workings have led to the conclusion that there is no evidence tending to indicate a progressive falling off in the yield of gold." Now, after another 30 years, we have learned that the early miners were not far wrong, after all; also we know a little more concerning the effects of oxidation in modifying the distribution of sulphide ores, more particularly.

On the next page Phillips summarizes the matter further by saying that "comparatively few mines have reached four hundred fathoms, but there is reason to believe that true veins extend to depths to which the miner will be unable to follow them." As it is the miner and not the geologist who is to follow them, we may assume, as is also otherwise suggested by the context, that Phillips meant that the 'true' veins would continue to contain ore to the depths indicated. In short, at that time the best informed writers assumed a persistence of ore beyond the reach of man and based all their ideas concerning the limitations to deep mining not on the probability of ore persistence, which was taken for granted, but on the invention of better methods of hoisting and ventilation.

Later Views. Twenty years ago geologists gave a ponderous sanction to the wild ideas concerning persistence and enrichment of ore in depth. Since then the application of geology to mining has advanced, and with it the geologist's knowledge of the conditions governing the exploitation of ore deposits. He has come in touch with the realities of mining, and appreciates that it is not a scientific pursuit but a means of making money by extracting metals from the rocky crust of the earth. Hence the latest geological writings on the subject under discussion are marked by an approach to the facts disclosed by mining operations. At the International Geological Congress at Toronto, in 1913, James F. Kemp summarized the position in a short but comprehensive paper.† His own teaching and practice have familiarised him with the economic phase of the prob-

* 'Ore Deposits,' by J. Arthur Phillips, 1884, p. 60.

† 'The Influence of Depth on the Character of Metalliferous Deposits, by J. F. Kemp, Professor of Geology, Columbia University.

lem. Therefore, it is not surprising that he should arrive at two conclusions:

"(1). While there seems to be nothing to prevent precipitation at greater depths than we have yet reached, yet conditions seem to be specially favourable in those portions which lie between the present surface and 2,000-4,000 ft. in depth.

(2). Secondary enrichment has increased the yield of those portions of many veins which are above 1,000 ft. in depth, the vertical extent of its action being limited to a relatively short stretch below the ground-water level."

Professor Kemp reviews the evidence derived from actual mining, and states that "in most cases, values in ores decrease in depth after a moderate section of the vertical extent of the vein has been passed. This experience is not universal, but it is the rule." He quotes my friend F. Lynwood Garrison and myself as witnesses covering a wide range of experience.*. He then outlines the conditions governing secondary enrichment, and explains why they are limited in vertical range. He refers to the effects of erosion, and states that it is clear in many cases that the ores mined to-day, "and which have been unaffected by secondary enrichment," were originally precipitated "at much greater depths than the present workings." Therefore, "the physical conditions involved in vertical depth, down to 6,000-10,000 ft., would not seem to be of themselves prohibitive of the precipitation of commercial ore." But as exploration becomes increasingly expensive in depth, the search for ore is more easily discouraged than in the zone of shallow mining. These latter considerations are well worthy of attention. That of erosion, I have discussed elsewhere in this paper in connection with California and Nova Scotia. As regards the increasing cost of deep exploration, my contention is that usually so much money is spent in shaft sinking that none remains for lateral exploration, which, from Professor Kemp's standpoint, as from mine, is commonly neglected.

Definitions. Before we proceed further in this discussion—for it is a discussion, I trust, not expecting or wishing to have the last word—we must ask one another what we mean by such terms as 'ore,' 'richness,' 'depth,' and 'numerous.'

First comes 'ore.' Whatever it may be to the mineralogist or the geologist, to the miner 'ore' signifies metal-yielding rock that can be exploited to advantage, which ordinarily means pecuniary profit. Profit is the ultimate excess of yield over expenditure. The ore extracted profitably by aid of forced labour, may, when mined and milled by high-priced labour, yield no profit, and becomes no better than waste rock. On the other hand, the waste of yesterday may become the ore of to-morrow, by the discovery of new metallurgical processes or the development of improved methods in mining. In short, 'ore' is an economic, not a geologic, term.

'Richness' in an ore may be defined as a composition ensuring a yield considerably above the cost of exploitation. An assay of one ounce of gold per ton means richness on the Rand, whereas it affords but a dubious margin of profit on the Gold Coast. However, the test by assay does not suffice to measure the commercial value of an ore, for that depends not only on the quality but the quantity. The geologist would call 5 oz. gold ore at 3,000 ft. an evidence of richness in depth, but if the quantity of 5 oz. ore proved to be insufficient, it would not repay the miner for his trouble in finding and removing it. Many mines have been quoted in proof of deep enrichment because some rich ore was discovered in their bottom workings, but their subsequent history has been singularly barren of achievement. One swallow does not make a summer. A sample does not make a mine.

'Depth' also is a relative term. Time and place are modifying factors. Twenty years ago a mine having workings 2,000 ft. below the surface was called 'deep.' To-day several mines have gone down to 5,600 ft.; hence, those that are less profound are made to seem shallow. In a new district, 500 ft. is deep; in an old one, 1,000 ft. is shallow. Here, also, I speak in terms of economies, for, to the geologist, the deepest shaft is but a pin-prick in the crust of the earth. When 'depth' is used in context with 'persistence of ore' it does not signify a fixed distance from the surface; it is comparative. Anyone making the statement that the ore of a particular lode persists 'in depth' means that the ore continues as far as it has been mined and as far as it is likely to be mined. If any limit can be assigned, within the possible reach of the miner, then the ore is not persistent 'in

* 'Decrease of Value in Ore-Shoots with Depth,' by F. Lynwood Garrison, *Trans. Can. Min. Inst.*, Vol xv., 1912, pp. 192-209.
'Persistence of Ore in Depth,' by T. A. Rickard, *Min. & Sci. Press*, 1912, pp. 232-233, 264-267

depth' but only to the vertical extent specifiable. If, on the other hand, it be argued that, of course, all veins, lodes, or other kinds of ore deposits do not extend interminably, or even to the depth attainable in mining, then we may ask whether the number of persistent ones is such as to warrant the generalisation now the subject of discussion.

This brings us to the fourth term needing to be defined: 'numerous.' Twenty-one years ago I wrote of the "numerous" exceptions to the general experience of non-persistence. Of those numerous mines the big majority have succumbed to Anno Domini, and are now no longer quotable either on the Stock Exchange or on the record of the optimistic engineer. As they became older, they were deepened; as they became deeper, they were exhausted. Of the exceptions to impoverishment quotable in 1893, only one or two survive, precariously; younger ones have come forward to take their place, but the number is relatively small, having regard to the multiplication of mines consequent upon the geographical expansion of the industry.

It may be argued that the increase of cost due to increase of depth suffices to explain the apparent tendency to discontinue mining in depth. This I deny. On the whole, big mines are operated at a lower cost when they are deep than when they are shallow. With increase in depth comes experience, which provides greater skill in metallurgical treatment, better equipment, better organization, and more systematic methods generally. No, the actual increase in the expense of winding and ventilation is an insignificant factor in this problem, although I do not ignore it. Similarly, excessive water, bad management, refractoriness of ore, and the other conventional reasons given locally for the abandonment or idleness of mines in sundry old districts are not in the least convincing. Excessive water is usually not a concomitant of great depth; on the contrary, deep mines are dry. The zone of maximum water is generally between 600 and 1,500 ft. Nor is bad management a sufficient reason; a mine having a persistent and profitable lode will not be closed down on account of bad management; the fact, if it be a fact, that it has suffered from such a cause, will be so notorious as to stimulate clever people to reopen the mine. Refractoriness will kill a poor mine, of course; but a rich lode of refractory ore does not remain neglected for long. Consider Broken Hill. Another factor is the price of metals; this may not affect gold mining directly, but it affects it indirectly through changes in the cost of the labour and supplies involved. It does affect the mining of the base metals in an obvious manner, but not sufficiently to invalidate the purpose of the present inquiry. The operation of sundry mines has been discontinued for varying intervals on account of a depression in the market for the metal or metals produced by them, but such discontinuance of operations has rarely been final, and it will be found in most cases that the smaller value of the product was due not only, or even chiefly, to a drop in the price of the metals, but to a fall in the grade of the ore. In short, eventual impoverishment in depth is normal.

Enrichment in depth. The idea of persistence was a sequel to that of enrichment in depth. The older fallacy, and, to some extent, its successor also, arose from a confused understanding of the changes observed at the base of the zone of oxidation. At first the limit between shallow and deep mining was fixed by the boundary between oxidised and sulphide ores. This proved an unscientific division, for sulphide ore may reach to the surface in glaciated regions, and oxidised ore may extend as deeply as 1,000 ft. in arid regions. It depends upon erosion. Another datum line was the water-level, which is the top of the zone of ground-water; but this also is subject to regional variation, and, in many cases, does not coincide with the present limit of oxidation. However, some of the notions concerning enrichment are undoubtedly traceable to the confusing evidence obtained at the water-level and just below it. It was recognised that when the miner struck wet ground, the ore changed, in gold and silver veins especially, but in copper and lead lodes also. Thus the water level became associated with a decreased yield of metal and a smaller profit, mainly because the ore assumed a refractory character and became more expensive to treat metallurgically. Even if its metal-contents did not decrease, it became, to the miner, a poorer product. In many districts the first penetration into refractory sulphides marked a period of gloom. When, by successful experiment, the miner learned how to treat his less docile ore and when, as happened almost concurrently, he ascertained, by a little deeper sinking, that just be-

low the water-level his lode or vein was much enriched, owing to secondary precipitation, he hailed it as a sign that courage to sink was all that was needed to ensure success. This pleasing experience was usually restricted to an inextensive zone, but in rare cases, as at Butte, the effects of sulphide enrichment extended so far as to warrant the use of the term 'depth.' If, at the same time, the method of mining was cheapened or the milling practice improved, the general effect was to associate the higher returns with the idea of shaft-sinking. In the majority of cases, of course, the rich secondary ore under the water-level was succeeded by primary ore of a lean character. In such cases further sinking was apt to coincide with impoverishment, disguised by patches of rich ore, sufficient to encourage deeper exploration. When a vein 'peters out,' it does not usually exhibit a gradual diminution of its payable contents; sometimes it narrows, so that, without a decrease in the grade of ore, it becomes a fading source of wealth; but much the more frequent sign of approaching exhaustion is a poor lode carrying irregular patches of rich ore, enough to please the assayer and perplex the superintendent.

DEEP MINING.

The deepest mines are those of the Comstock (in Nevada), Pzribram (in Bohemia), Bendigo (in Victoria), the Lake Superior copper region (in Michigan), the Witwatersrand (in the Transvaal), the Kolar goldfield (in India), and the Minas Geraes (in Brazil).

On the Comstock lode, the greatest depth attained was 3,350 ft., in the Union workings. The bonanza ore-bodies of the celebrated California and Virginia mines extended from 1,100 ft. to 1,860 ft. in depth. The lode became poor below the Sutro Tunnel, which is the main adit, and is 1,850 ft. below the outcrop. In the bottom workings only patches of rich ore were found. More money was spent below 2,000 ft. than that obtained from the ore extracted.

The Adalbert shaft at Pzribram reached a depth of 3,640 ft. This mine has been idle for many years, work having been discontinued when it became unprofitable. The ore follows fissure-faults associated with diorite dikes in Cambrian sandstone.* In the deepest workings it shows the same composition as in the upper levels; only it is poorer.

Bendigo. For many years Bendigo was the pride of the deep sinker. The maximum depth attained was 4,614 ft. in the Victoria Reef Quartz mine. The deepest ore mined in the district was in a saddle formation cut at 4,154 ft. in a neighbouring property, the New Chum Railway. It was proposed to sink the Victoria Reef Quartz shaft to intersect this orebody on its pitch at 4,800 ft., and the work was actually commenced, but the principal financial contributor withdrew, despite a subsidy from the Victorian government. In the New Chum Railway this orebody had averaged 6½ dwt per ton, a yield insufficient, in my opinion, to warrant the effort to exploit the continuation of the same orebody at a much greater depth. All the deep mines on the New Chum 'line of reef,' or lode, have ceased sinking. Work has been diverted to the shallow ground, neglected in the days when there was a lively competition to see which local company could boast the deepest shaft. When I was first at Bendigo, in 1890, it was an axiom that if a mine became poor it needed only to be deepened. Up to a point this had some sort of justification in the peculiar structure of the district,† but the craze for deep sinking became a mere obsession. I hinted as much, and advocated lateral exploration on the intermediate lodes. In 1912 the folly of a further deepening of the mines in the face of unpalatable evidence was recognised, and the "shallow ground" theory, as it was called, became vindicated.† On the whole, it may be asserted that mining at Bendigo below 2,000 ft. was not profitable; a few mines for a short time earned dividends, but the general result was unfavourable financially.

Lake Superior.—The deepest metal mine in the world is the Tamarack, at Calumet, Michigan. This mine is the 'deep-level' of the celebrated Calumet & Hecla, and was planned to exploit the rich copper conglomerate where it passed on the dip beyond the Calumet & Hecla's side-line. The No. 5 shaft of the Tamarack cut the lode at 4,835 ft., and was sunk to 5,253 ft. vertically, while the No. 3 cut the same lode at 4,662 ft., and was deepened to 5,281 ft. The deepest workings, however, are those of the No. 5 shaft, for they extend to 5,368 ft. below the surface. The Tamarack enter-

* 'The Genesis of Ore Deposits,' by Franz Posepny, *Trans. A.I.M.E.*, p. 84.

* See 'The Bendigo Gold-field,' by T. A. Rickard, *Trans. A.I.M.E.*, Vol. xxv., pp. 465-545.

† *The Bendigo Advertiser*, June 10, 1912.

prise was a disappointment.* As long ago as 1900 it was recognized that the wonderful run of ore had been bottomed. The last dividend was paid in 1907, and the Tamarack was absorbed by the Calumet & Hecla in 1911. Meanwhile, the Calumet & Hecla itself had rapidly become poorer; in 1873 the yield of copper averaged 5 %; in 1900, 3 %, and in 1910 only 1½ %. The future of the Calumet & Hecla company is known to depend upon the yield from its shallow subsidiary properties.

The Rand. Next we come to the Witwatersrand. As I have already suggested, those who believe in the persistence of ore in depth should enjoy a large perspective, and open-up their mines on a grandiose scale. To sink a couple of hundred feet and then commence to stope a lode that is believed to continue rich indefinitely downward is puerile engineering. It is fair to say that to any such charge the men of the Rand can plead 'not guilty.' Those who started the big-scale exploitation of the Main Reef series were true to their quasi-geological convictions, and designed the development of their mines in a manner entirely consistent. The doctrine preached on the stock exchanges of Johannesburg and London, ten years ago, was that the beds of gold-bearing conglomerate had the uniformity of a coal seam, and would continue profitably productive to a depth beyond human reach. At that time the leading engineers of the Rand were discussing projects for tapping the Main Reef series at vertical depths ranging from 5,000 to 8,000 ft., the latter figure taking it for granted that profitable 'banket' would extend to 18,000 feet on the dip. At that time I ventured to protest against such flouting of world-wide experience in mining, and wrote a leading article under the sardonic title of 'Even Methuselah Died.'† Since that was published the irrefutable logic of facts has been ignored persistently by those responsible for the direction of mining affairs in South Africa, with the exception of one or two perspicacious engineers, whose utterances have been disregarded. In 1907 Frederick Hellmann testified before the Mining Industry Commission, at Johannesburg, that a decrease in the grade of the ore was "most marked" in "all mines" with which he was acquainted on the Rand. He added that in the Driefontein this impoverishment was especially noteworthy.‡ Mr. Hellmann was then general manager of the East Rand Proprietary. Nevertheless, in March, 1910, Sir George Farrar and Sir Lionel Phillips testified to the shareholders of the East Rand Proprietary that the Angelo Deep§ had "added to the evidence that the Witwatersrand series were not decreasing in value" in depth, and Sir Lionel Phillips asserted that "if they excluded the surface area down to 200 or 300 ft., there seemed to be no evidence whatever that the gold contents at the deepest levels were not fully as high as they were at the surface, or within 300 ft. of the actual surface." This statement is confused, but its general intent is plain enough; it is in direct contradiction to the ex-manager's testimony, which must have been known to both of these directors. In the following year, before the Institution of Mining and Metallurgy, a reference to the subject, in a paper by C. O. Schmitt, provoked H. F. Marriott, consulting engineer to the Central Mining Corporation, to state that "the assay-values of samples of actual reef-widths taken throughout all the mines" of the group controlled by his employers showed "no substantial evidence" of general deterioration in the richness of the ore with increasing depth. I referred to this assertion editorially in *The Mining Magazine*,‖ and laid stress on the importance of furnishing the proof of it, in the interest alike of economic geology, the mining industry, and the share-market. Obviously, if Mr. Marriott had 'made good' it would have been to the interests of everybody concerned. He did not accept the challenge. It is fair to assume that he had not the facts to support his assertion.

And now, since I began¶ to prepare this paper, comes the annual report of the Consolidated Gold Fields of South Africa, for the year ending June 30, 1913, with an admission by H. H. Webb that "there can be no doubt that from our experience and in our mines the average value of the ore developed over large areas has been getting lower as greater depth is attained. High-grade areas and patches are encountered in depth, but these are not as frequent or extensive, nor are they in all cases as high in value, as they were nearer the outcrop, consequently the general average over large areas has fallen." No one now connected

* 'The Copper Mines of Lake Superior,' by T. A. Rickard, 1905, p. 51.

† *Engineering and Mining Journal*, May 2, 1903.

* One of the mines included in the East Rand Proprietary.

‖ In the issue of August, 1911, p. 98; and in that of November, 1911, p. 341.

¶ Early in November, 1913.

with Rand mining is in a position to speak with greater authority than Mr. Webb. It is evident that Methuselah is going the way of all flesh.

The deepest mines on the Rand are the Jupiter, 5,040 ft., the Cinderella Consolidated, 4,627 ft., the Simmer Deep, 4,780 ft., and the Village Deep, 4,245 ft. Of these, the first two are idle, for the sufficient reason that it is unprofitable to operate them. The Simmer Deep is struggling with an ore so low-grade as to warrant the prediction that it also will be shut down at an early date. The Village Deep is still a profitable mine, but the ore it produces is not as rich as that of the Ferreira Deep, which is above it on the dip, and considerably less rich than that yielded by the Ferreira and Wemmer, which were the corresponding outcrop mines. By way of comparing outcrop and deep-level mines, that is, shallow and deep properties in vertical succession, I may quote the Geldenhuis, Geldenhuis Deep, and Jupiter series; or the Bonanza, Robinson Central Deep, and the Robinson Deep; or the Durban Roodepoort and the Durban Roodepoort Deep. Surely the mention of these is enough to give pause to the unscientific optimist. They represent facts to which the promoters of the Rand have tried to shut their eyes for many years, but they can no longer be ignored by the members of our profession.

Kolar. Next we come to the Kolar goldfield. The prevailing country-rock is the Dharwar schist, which is hornblendic and pre-Cambrian.* This is traversed by lodes that in their strike 8° west of north) and in their dip (40° to 70° west) conform to the foliation of the schist, so that these ore deposits may be regarded as veins formed along planes of fracture coinciding with the foliation.† The coincidence is not perfect everywhere, for, as might be expected, the later fractures in places found an easier way by diverging slightly from the earlier structural lines of weakness.‡

All the noteworthy mines are on one lode the Champion Reef—but several subordinate veins have been exploited profitably. The orebodies represent the richly gold-bearing portions of the quartz veins; even where a vein becomes too poor to be mined the quartz may persist. From Plummer's shaft in the Mysore mine to Kennedy's shaft in the Nundydroog, a distance of three miles, the main lode is continuous except where intersected by dykes and cross-courses. At some levels this lode has been persistently profitable for 7,000 ft. on the strike. The folds in the country-rock do not interrupt this continuity; on the contrary, the ore-bodies are most generous in width where the schist has undergone a sharp fold, making a pleat, the axis of which marks the pitch of the shoot. The longitudinal sections of the four principal mines indicate that the various orebodies are comparatively uniform in their pitch, at an angle of 45° north. This fact has had an important bearing on the history of individual mines, for it has caused sundry ore-shoots to pass from one property into another. The width of ore averages about 4 ft., but stopes as wide as 35 ft. are recorded.

Of the seven properties linked along the main lode, four have become great mines; the other three need not be discussed in connection with the persistence of ore in depth. The length of the four productive properties and the depth of their workings is given herewith, in feet.

	Extent of Property on Lode.	Depth on the Dip.	Vertical Depth.
Mysore ...	7,847	5,000	3,465
Champion Reef	3,506	4,700	4,340
Ooregum ...	3,355	5,100	4,670
Nundydroog ...	6,198	3,200	2,756

For many years the yield from these deep mines was so steady that the Kolar goldfield became a stock example of persistence, if not enrichment, in depth. This hasty inference was as fallacious as the suggestion, made in 1904, that a flattening of the lode presaged a speedy end to a highly productive industry.* Leaving geological inferences of doubtful value, I find that the statistical record indicates impoverishment in depth, but sagacious management has decreased the cost of exploitation to such an extent that the economic result has continued to be most satisfactory.

Of the four principal mines, the most northerly, the Nundydroog, has two ore-shoots, 1,800 ft. apart. The lower levels became decidedly poor in 1908 at a depth of 1,200 ft. Subsequently the mine gave

* J. Malcolm Maclaren, 'Gold,' p. 15.
† F. H. Hatch, 'The Kolar Goldfield,' *Memoirs of the Geological Survey of India*, Vol. xxxiii., Part 1, 1901.
‡ For this note I am indebted to Mr. Arthur Llewellyn, formerly in charge of the Survey and Geological Department of the Kolar group of mines.

* 'Geology of the Kolar Goldfield,' by A. M. Smith, *Trans. Inst. M.M.*, Vol. xiii., pp. 152-180.

good results, and the output of ore was gradually increased. Eighteen months ago, however, the bottom workings appear to have passed through the principal ore-shoot at 2,500 ft., the output becoming curtailed pending further developments.

On the Ooregum, which comes next southward, the ancients followed the richest spots down to 250 ft.,* and dug into the vein along the entire length of outcrop within the limits of the property. The first orebody exploited was traced from these old workings to 750 ft. in depth, where it diminished in length and richness; but an ore-shoot pitching north from the Champion Reef crossed the southern boundary, and enriched the lode from 1,200 to 2,000 ft. Under this came a third shoot, which was broken by a dyke near the south boundary, but made again 1,000 ft. north on the 3,000 ft. level, from which it has been traced to the bottom of the mine at 5,100 ft. on the dip.

In the Champion Reef the old workings reached to 275 ft. This mine did handsomely until about six years ago, when it was 3,700 ft. deep. Since then the length of stoping ground has contracted. An examination of the qualitative stope-section shows that impoverishment began at the south end and on the 9th level. Down to the 16th the mine was 'sick.' Then the upper Tennant's shoot from the Mysore crossed the boundary, and for five levels ore was almost continuous from boundary to boundary—3,500 ft. Again for 12 levels the lode was poor and patchy. Then an improvement became apparent, on the 33rd level, where, at the south end, the lower portion of Tennant's shoot crossed the south boundary and gave renewed encouragement.

The Mysore is the great mine of the Kolar district. It extends for 1½ miles along the lode, and includes ten distinct ore-shoots. In places three or four orebodies are found overlapping. The ancients worked to a depth of 300 ft. at the north end of the mine, where Gilbert's shaft was sunk, but this orebody proved disappointing. Below it, however, came Tennant's shoot, which did not outcrop, but ended at about 150 ft. from surface. Under Tennant's, and close to it, come another. Indeed, in this mine the three principal runs of ore have the form in each case of twin ore shoots. Crocker's shoot is the most regular and persistent on the goldfield; it has had a length of 1,500 ft. of continuous high-grade ore. Impoverishment appeared at 1,600 ft., followed in depth by alternations of yield, the latest stage being impoverishment for 700 ft. McTaggart's shoot, also in two divisions, one pitching under the other, reaches to within 200 ft. of the surface, but does not outcrop. It is enriched at 1,660 ft., and for 1,000 ft. in depth, becoming poor at 2,800 ft.

In this gold-mining district the existence of ancient workings proved as much an obstacle as a guide in the first operations, because the unknown former workers had removed all the rich and easily accessible ore to a depth reaching nearly to 300 ft. under the surface. The present industry owes its origin to an Irish soldier named Lavelle, who began prospecting in 1871 and obtained a concession two years later. In 1876 he transferred his rights to a syndicate—the Kolar Concessionnaires—formed by the late General de la Poer Beresford. The first working capital was furnished by Arbuthnot & Co., of Madras, but this famous mercantile house retired from the venture at an early stage. In 1880 the firm of John Taylor & Sons was invited to assist the enterprise. This was done after the late Bell Davies had reported favourably. But for three years the results were most disappointing, and it was due to the pertinacity of the engineers, more particularly Mr. John Taylor, that the work was pushed to eventual success, aided by the financial support of the late Sir Charles Tennant. How complete that success has been is proved eloquently by the statistical record.

	Ore. Short Tons.	Gold. Fine Oz.	Dividends. £
Mysore ...	3,917,109	3,603,112	7,414,344
Champ. Reef	3,179,185	2,502,292	4,018,966
Nundydroog	1,526,554	1,241,808	2,002,768
Ooregum ...	2,402,346	1,562,829	2,214,320

In his earlier paper, Dr. Hatch gives the yield of gold per ton in 1899; this can be compared with that in 1913, by converting the standard gold as recorded in the annual reports into the fine gold quoted in the geological memoir. Another correction must be made for the use of the short ton (2,000 lb. avoirdupois) in some of the later reports as compared with the long ton (2,240 lb.) of the earlier statistics. Making these corrections, I find that the yield of the four

* Unless otherwise stated, all the mine measurements are taken on the dip of the lode.

* The extra impost has been paid to the Government since the renewal of the leases.

be extracted, not as a counter in Stock Exchange manipulation. To these two facts is

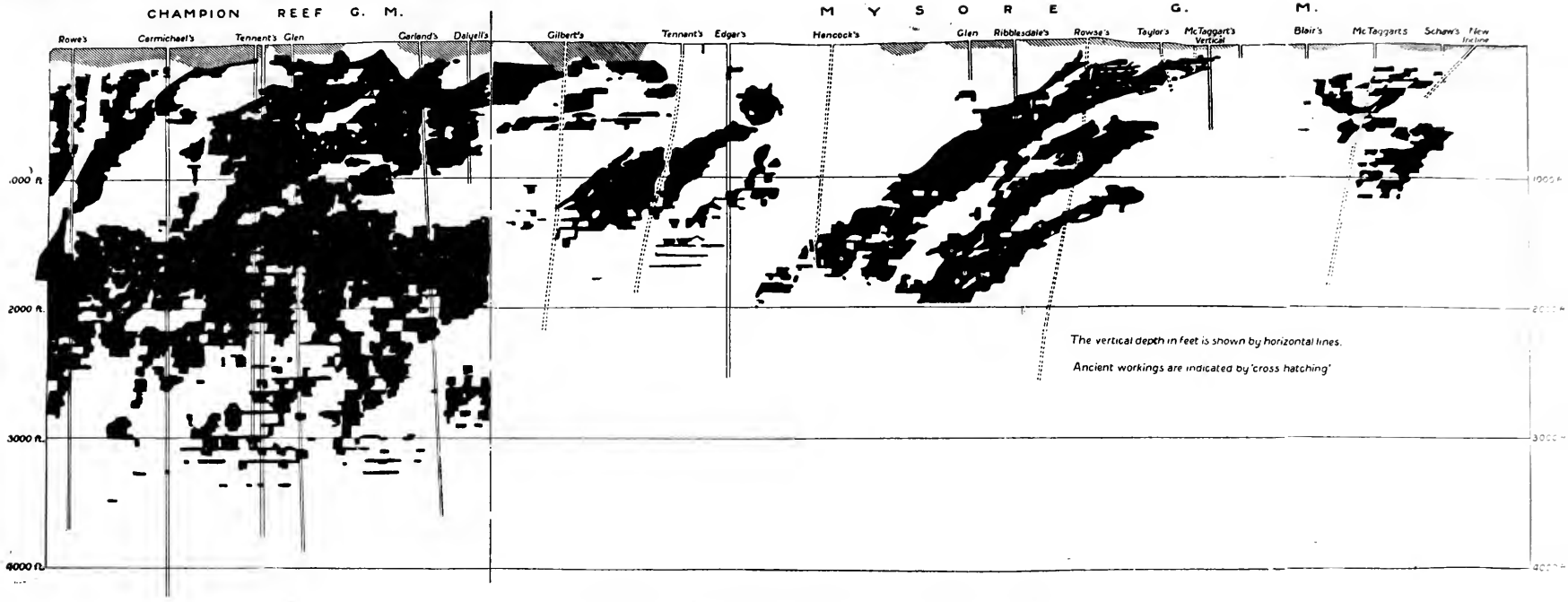


Fig. 1.—Vertical Section of part of Kolar Goldfield, showing portions of Lode-Area Actually Stoped.

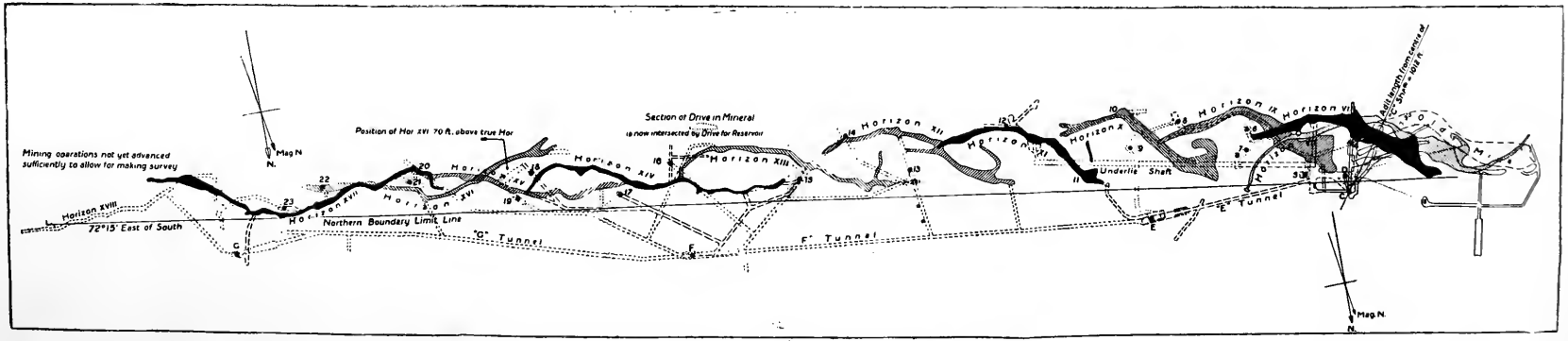


Fig. 3.—Plan showing Shape of St. John del Rey Orebody at Successive Levels

big mines compares as follows in dwts. of fine gold per short ton.

	1899.	1913.
Mysore ...	27·68	14·29
Champion Reef	26·41	10·92
Nundydroog ...	19·60	15·95
Ooregum ...	14·04	13·92

The decrease in yield is apparent; whether it is due to the treatment of a larger tonnage or whether, conversely, the necessity for treating a larger tonnage arises from the lower grade of the ore, is not obvious. Other evidence suggests that there is a real impoverishment, which has been largely compensated by a cheapening of the processes of exploitation. This is indicated by a comparison of the cost incurred at the two periods, in shillings per short ton, as follows:

	1899.	1913.
Mysore ...	33·36	24·24
Champion Reef	42·34	26·55
Nundydroog ...	39·28	30·33
Ooregum ...	40·53	22·68

These figures do not give the total cost. Among the items omitted are (1) the royalty paid to the Mysore government, at the rate of 5% of the gross value of the gold, *plus* 2½ on dividends distributed since March, 1910,* (2) the English income tax, and (3) the depreciation of, or expenditure on, shaft-sinking, plant, and machinery, this item being differently treated at the two periods under comparison. The actual total cost is about 50 more than the 'working' cost or revenue expenditure. However, that does not affect the comparison, for it applied as much in 1899 as in 1913. On the other hand, while the cost has not decreased as much as the yield, the dividends have increased considerably in some cases, as the following comparison indicates:

	1899.	1913.
Mysore ...	£350,000	£381,250
Champion Reef	275,000	130,000
Nundydroog ...	72,600	99,050
Ooregum ...	65,000	129,250

* The extra impost has been paid to the Government since the renewal of the leases.

This favourable result is due mainly to the increased tonnage:

	1899.	1913.
Mysore ...	103,424	362,662
Champion Reef	104,296	220,511
Nundydroog ...	39,424	90,650
Ooregum ...	71,800	153,636

Obviously these statistics do not bear directly on the geological problem before us, simply because mineral exploration is not conducted with a view to elucidating scientific questions.

In Fig. 1 is shown the portions of lode-area that have actually been stoped. This does not show all the ground that is ore-bearing, of course, but it serves to indicate the shape, size, and trend of the ore-shoots.

The question of continuity of ore in depth is also complicated by the flat pitch of the successive ore-shoots. This feature need not, of course, perplex any intelligent observer now, but in the early days of the mines, before the geological distribution of the ore had been ascertained, it was undoubtedly a hindrance to confident exploration. The Mysore, for instance, passed through several periods of depression. In the company's office a transfer of 200 shares at 9d. is still preserved as a memento of the dark days of 1883. The ore under the old workings near Gilbert's shaft appeared to 'peter out,' although it really sloped flatly across the adjoining boundary into the Champion Reef property. A long exploratory level at 1,100 ft. had been extended for 3,000 ft. without finding anything of particular promise, for the rich part of Tennant's shoot was underfoot and Crocker's orebody was farther south. Such eras of depression have been repeated not only in the history of the Mysore mine, but in that of the Kolar goldfield as a whole. It is the barest justice to state that the persistence of intelligent effort on the part of the management cannot be discounted even by one who finds it necessary to question the persistence of the ore. Two outstanding facts in the history of this group of mines are (1) the uninterrupted technical control of operations by the oldest mining firm in London, and (2) the complete detachment of those operations from share-gambling. The lode has been exploited for the gold to be extracted, not as a counter in Stock Exchange manipulation. To these two facts is

owing, largely, the ability of the mining companies to obtain financial support, whenever necessary, from a large body of absolutely loyal shareholders. The history of Kolar constitutes one of the best pages of the mining record.

Returning to more technical considerations: A careful examination of the longitudinal sections of the four principal mines will show how the successive ore-shoots slope one under the other, often in pairs, at an angle of 45° with the horizon, tending, however, to spread apart in depth northward. This spreading is due to a flattening of the fold, the structure of which was a determining factor in the deposition of the ore. The shoots are not always continuous bodies of ore, but represent favourable structural conditions, having a fairly regular pitch, which, as I have said, coincides with the axes of folds in the schist. The decreasing pitch of the ore-shoots and the apparent spreading or fanning of their axial lines points to a diminution in the acuteness of folding or intensity of plication. This is an unfavourable change in the geological structure.

This structure explains many of the vicissitudes recorded in local history. Vertically the distance between ore-shoots is considerably greater than the interval separating them at right angles to their pitch. When an ore-shoot, like that at Gilbert's shaft, was lost in depth, it was not until long afterwards that the stope-section showed that it had passed flatly across the boundary into the adjoining mine. At least two ore-shoots in the Mysore did not outcrop, and therefore were not easily found. The general effect was to accentuate unduly the alternations of depression and prosperity, and to exaggerate the intervals of poverty and richness. On the other hand, as the mines have become healthily enlarged, the managers have provided against fluctuation of output by allowing large reserves of ore to accumulate, whereby to help the lean years and obviate anxiety to their shareholders. Thus, at the present time, the Mysore has 1,377,000 tons of ore in reserve, this being equal to over four years' supply. As regards the immediate subject of this paper, the output for successive years does not correspond with increasing depth because the ore milled in 1914 may come from stopes two thousand feet higher than the bottom level; in other words, the depth of the mine in any year bears no direct relation to the quality of the output. Of this, of course, the economic geologist must be aware, lest he fall into inferences of a most misleading character.

St. John del Rey. Two important gold mines are being operated in the state of Minas Geraes, in Brazil. Of these the Ouro Preto has no pretensions to profundity, but the Morro Velho mine of the St. John del Rey company has workings that have reached to 5,350 ft. vertically,* or about 7,600 ft. on the pitch (45° decreasing to 40° east) of the orebody. This mine was bought for £56,434 by the present company in 1834. In 1867 a fire and the collapse of the open-cut workings necessitated the cessation of operations and the sinking of two shafts, 1,122 ft. and 1,422 ft. deep, respectively; in 1874 work was resumed, but a second caving of the ground ensued in 1886, and the mine again became unproductive. Two new vertical shafts were sunk through the orebody to a depth of 2,264 feet, this work being completed in 1892. Since then the mine has been developed by a series of internal shafts, each 1,200 ft. deep, in step-like succession, the first being started 610 ft. east of the main shafts sunk from surface, and each of the succeeding ones 1,420 ft. farther east, to be connected by a level parallel with the lode at a safe distance north in the country-rock. This method is in itself an indication of the cautious policy of the manager, Mr. George Chalmers, who has never felt justified in looking sufficiently far ahead to incur the heavy expense of a more comprehensive system of development.

According to Orville A. Derby, of the Brazil Geological Survey, the lode itself† in calcareous schist, which is older than the Upper Silurian and possibly of even pre-Cambrian age. This schist, or calcareous phyllite, is, in part at least, a sheared representative of an older eruptive rock intercalated within an ancient sedimentary series that has been highly metamorphosed in consequence of orographic movements of considerable extent. Granite intrusions, in the form of bosses and dikes, exist in the vicinity of the mine. The ore is mainly a massive mixture of pyrrhotite, with arsenopyrite and common pyrite. Quartz is a subordinate ingredient. The typical composition of the ore is as follows: Pyrrhotite 28·5 , arsenopyrite 5 , iron pyrite 6·5 , quartz 24 , carbonates of iron, lime, and

* In 1916 the shaft should reach horizon No. 20, at 5,826 ft. below the surface.

† As reported by him to the St John del Rey Company in 1901 and 1904.

magnesia together making 36%. Copper pyrite is usually present to a minor degree. According to Dr. Derby the lode occupies a fault-plane dipping eastward at an angle of about 45°, the shape of the orebody being modified by systems of jointing in the schist.

The annexed section (Fig. 2) of the mine is the one published in the company's annual reports. It looks like the cross-section of a thick vein dipping at about 45°. This is not the case, as Dr. Derby's description has already indicated. The real character of the deposit is suggested by the

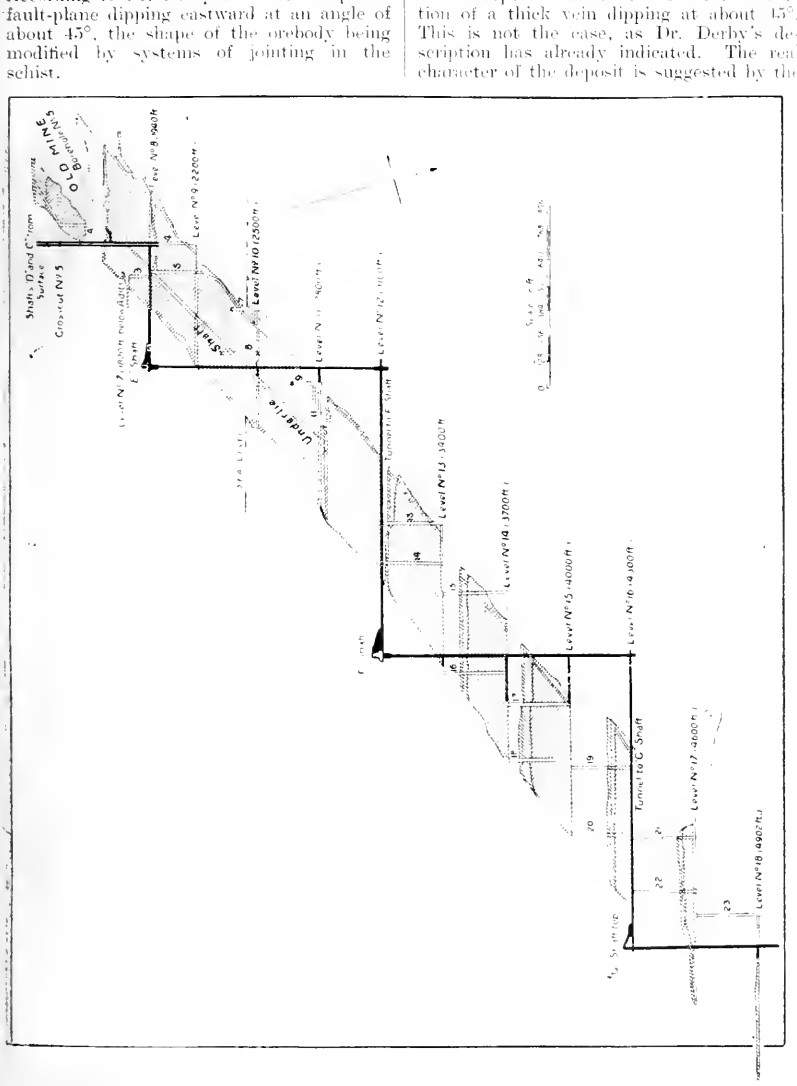


FIG. 2.—Longitudinal Vertical Section of St. John del Rey Orebody.

plan (Fig. 3*), in which I have shown the shape of the orebody at successive levels by difference of shading so that the superposition is made evident. The foliation of the schist is nearly vertical and the strike is 17° 45′ east of north. Owing to the sinuous outline of the orebody and the progressive modification of shape downward, I am emboldened to suggest that faulting and jointing will not suffice to explain the structure of this deposit. The facts, to my mind, suggest a lenticular orebody within a folded schist. It is irregularly lenticular in horizontal section and incompletely funicular to vertical view. Apparently it lies in the vertical plane of the schist and pitches eastward in sympathy with an acute torsional fold, accompanied by slipping and shearing of the encasing rock. This interpretation is more favourable to the relative persistence of the orebody than one based on the structural conditions specified by Dr. Derby.

At the 9th level a "horse" or inclusion of country splits the ore in its western and widest part. Indeed, down to the 13th the mine was in a parlous state, but on the 14th an enrichment of the eastern end became evident and has persisted ever since. Below the big open-cut the mine is dry. In 1901, however, on the 11th horizon (3,126 ft. below surface) some water was found; and it is said that this water contained "all the constituents of the lode." An analysis would be of great interest. In his report for June, 1913, the manager, Mr. Chalmers, refers to the problem of persistence in depth. He mentions the careful study made by Dr. Derby, who expressed a favourable opinion at a time (in 1901) when the orebody showed one of those contractions to which it has been subject at intervals. Mr. Chalmers proceeds to state: "Since Dr. Derby gave his opinion we have explored the lode down many hundreds of feet, and find it has altered a little in shape, a very little in quantity, and improved in contents." He concludes that the increase of temperature, unless mitigated by improved ventilation, is the only obstacle to the deeper exploitation of the lode. The temperature increases 1° F. in 126 ft., so that the rock-temperature at 6,100 ft. will be 117°. This would be no serious obstacle, if the mine had vigorous ventilation through two main openings; as it is, the air comes down through a tortuous series of shafts, to ascend by an even more tortuous series of winzes. Ventilation is stimulated by a Sirocco fan of 150 h.p. at the top of the C shaft. Another factor prejudicial to deeper mining is the system of hoisting, in steplike series up the various intermediate shafts and along their connecting levels. This must be costly. An incline-shaft, on the pitch of the orebody, was tried from the 8th to the 11th horizon, that is, for a distance of 900 ft., but it was found to be too dangerous, as the schist "blisters" and comes away in dangerous falls of ground. In depth the orebody has become longer (about 1,000 ft.) and narrower (12 to 15 ft.) as compared with the nearly uniform length of 600 ft. and the width of 45 ft exposed in the upper workings.

From 1835 to 1892 the output of the mine was 2,958,593 tons, yielding £5,167,592, or nearly 35s. per ton; while in the period from 1892 to 1913 the output has been 2,715,933 tons, yielding £5,498,352 in gold, equivalent to 40s. per ton. In the first period 7·3 % was rejected or sorted; in the second period only 5·7 %. The total output was 5,671,526 tons mined or 5,313,028 tons milled for a yield of £10,665,944, which is equal to a yield of 39s. 11d. per ton crushed. The mine was unprofitable for the first five years: between 1842 and 1867 the total dividends and bullion tax amounted to £896,500; from 1874 to 1886 the profit, as above specified, was £556,600; from the reopening in 1892 to 1913 the profit was £730,266, so that the total dividends and tax payments amount to £2,183,366. During the last period a bond interest of £277,700 also has been paid. From the commencement about £1,000,000 has been raised for working capital.*

Reviewing the history of the mine according to the three productive periods, 1850-1865, 1875-1886, and 1900-1913, I find that the working cost (not the total cost) ranged from 17s. to 23s. in the first period, from 26s. to 30s. in the second, and from 25s. to 31s. in the third. During these respective intervals the extraction in the mill has ranged between 52 % and 80 %, 70 % and 75 %, 71 % and 93 %. Dividing the history of the mine into two parts, before and after 1892, in accord with the statistics available concerning tonnage and yield, I find that the extraction averaged about 65 % in the early period and 87 % in the late one, improving to 92 % during the last seven years. Now, if we make these allowances for loss in milling, we shall find that the 35s. per ton extracted from the shallower workings.

* See inset opposite p. 117

* For these facts I am indebted to a historical account prepared by Mr. George Chalmers, the general manager, in 1913.

represented gold contents equivalent to nearly 54s. per ton against the 40s. extraction and 46s. gold contents in the ore mined in the deeper workings. On the face of the evidence, allowing for difference in sorting, it seems to me that the lode cannot be said to have become either richer or poorer as it has been deepened, and that the St. John del Rey is an example of persistence to great depth. Yet I consider that the experience of other mines warrants us in commending the cautious policy of the management, which does not assume the continuity of the orebody for more than two or three hundred feet as measured at any time from the bottom level.

If the orebody follows a fold in the schist, it is obvious that similar folds along the strike of the schist may enclose other orebodies. Looking at the longitudinal section published in the reports, it is remarkable that so little exploratory work has been done outside the immediate limits of the known ore-bearing ground. Having regard to the unusual vertical extent of the workings, it would appear that lateral search for other orebodies is abundantly justified.

North Star. Another satisfactory instance of relative persistence of ore in depth is afforded by the North Star mine, at Grass Valley, California. The vein traverses diabase; the ore consists of quartz containing 2½% iron pyrite, a little galena, and gold. The vein has well defined walls, 3 ft. to 1 ft. apart; the space between being filled usually with one foot of quartz, while the rest is country rock, crushed and impregnated with pyrite. The dip of the vein varies, but the average is about 23°. The main ore-shoot has a decided pitch westward.

The vein was discovered in 1851, and was first called the French lead, having been located by two Frenchmen. Other compatriots secured adjacent claims. They combined to form the Helvetia and Lafayette Gold Mining Company. This came to grief in 1858. In 1860 J. C. Pascoe, Edward Coleman, and others bought the property for $15,000 and joined it to the North Star group of claims under the name of the North Star Quartz Mining Company. In 1867 a re-organisation was effected, and a company called the North Star Gold Mining Company was incorporated, with a capitalization of $1,000,000. In 1875, when the mine was 1,200 ft deep on the incline, a decrease in the yield compelled the cessation of operations. In May, 1884, a new company, the North Star Mining Company, was incorporated, the mine was re-opened and the shaft was sunk. Since then the mine has been continuously productive, although for two or three years prior to 1902 the output was meagre. The late James D. Hague was the guiding spirit of this last and most successful phase of the enterprise, aided by Arthur DeW. Foote, who has been resident superintendent since 1896, and is now succeeded by his son, Arthur B. Foote.

Between 1884 and 1891 the shaft was sunk to 2,300 ft. on the dip. Rich ore was found on the 1,800 ft. level near the crossing of a fault. Good ore also was extracted between the 1,900 ft. and 2,100 ft., raising the yield in 1893 to $31 per ton. At the 2,300 ft. level, however, the quartz ceased, and the vein itself became pinched. Despite the poor outlook, it was decided to sink a vertical shaft to cut the vein at 4,000 ft. on the dip. This work was commenced in 1897. To provide funds for this new development and to acquire additional protective claims, the capital of the company was increased in 1894 from $1,000,000 to $2,000,000. Again in 1899 a new company, the North Star Mines, was incorporated with an issue of $2,500,000 to carry out this large-scale scheme of development. The vertical shaft, begun in 1897, cut the vein at 4,000 ft. on the dip and 4,630 ft. vertically on March 31, 1901. The heavy flow of water and the completion of the necessary equipment hindered exploratory work, but the rise on the vein, to connect with the old incline-shaft, was completed on November 22, 1903. Subsequent operations have fully justified the sinking of the new shaft, for the ore on the deeper levels has proved highly profitable. The most productive level has been the 3,700 ft., for this 'lift' will yield altogether over 200,000 tons of ore. Moreover, this part of the workings is remarkable as having furnished a pocket from which $40,000 worth of ore was taken to surface in 20 candle-boxes.

Since 1901 the development of the mine has been continued by the sinking of the main shaft and the opening of new levels down to 5,300 ft. on the dip, or 2,000 ft. vertically. The old incline was continued down to 5,390 ft.

The following statistical information* affords an interesting comparison between

* For which, as for the historical facts herein recorded, I am indebted to a statement prepared by William Hague and W. B. Pagan; see also *Mining and Scientific Press*, Oct. 10, 1914.

the produce of the upper and lower workings that is, above and below the 2,700 ft. level, which vertically is 1,200 ft. deep. Work in the upper mine, through the incline-shaft, began in 1851 and ceased in 1911; work in the lower mine, through the vertical shaft, began in 1901 and is still in progress.

	Tons.	Average	Cost.	Profit
Upper ...	438,000	18.30	12.90	5 40
Lower ...	732,599	11·87	6 38	5·48
Total ...	1,170,000	14 28	8 82	5.46

The totals compare thus:

	Gross Value.	Profit
Upper	8,019,000	2,369,000
Lower	8,696,596	4,020,224
Total to Dec. 31, 1913	16,715,000	6,389,000

It is estimated that prior to 1884 the mine yielded $2,625,000 at an expense of $2,000,000 from 108,000 tons.

This is a good example of successful mining. While the average yield of the ore has decreased to one-half, the cost has gone down at the same rate, so that the profit has been fully maintained. This is due to engineering skill as expressed in a large and systematic scheme of development, supplemented by an up-to-date reduction plant.

It has been shown that the yield of the ore has declined in depth. On the other hand, it appears that the yield per square foot of vein-area has increased, because a bigger width has been broken. In the lower portion of the mine the vein and the quartz are both said to have widened somewhat, and more stringers have traversed the country between the walls. This helps to explain the lowering in the grade, but as against this modifying factor must be mentioned the improved metallurgical treatment. In the earlier period the extraction cannot have averaged more than 80 of the gold contents; in 1913 it had risen to 97½ . Thus the impoverishment of the vein remains a fact, although it has been disguised and overcome by the skill of the management in making a reduction of cost that has marched with the diminishing yield of the ore.

As regards lateral exploration, no effort was made before 1900 to prospect the ground eastward. In that year the 500 ft. level was extended through a barren vein for some distance, and then passed through the top of a short shoot of rich ore. At the surface no indication was given of this ore-shoot, but the 1,100 and 1,900 ft. levels were pushed eastward promptly and found the downward trend of it. Another was found a few hundred feet east of it; together they yielded 182,000 tons of ore between the 1,100 ft. and 2,700 ft. levels, before becoming exhausted in 1911. Some of this ore was rich, averaging at one place $37.50 for 18,000 tons.

In the U.S. Geological Survey's report on this district,* Waldemar Lindgren asserts that "it can be confidently stated that there is no gradual diminution of the tenor of the ore in the pay-shoots below the zone of surface decomposition." There is a diminution, I submit, although it may not be gradual. Moreover, I believe that in some cases the impoverishment comes by way of a shortening of the length of particular ore-shoots. I mention for what it may be worth the fact that the North Star Mines company has tried, from time to time, to buttress its business by acquiring and developing adjacent mines, among which may be named the Stockbridge, New York Hill, Gold Hill, Granite Hill, and Cincinnati Hill. In all of these ventures a loss has been scored, the total for the five aggregating $125,000. The earlier North Star Mining Company purchased the neighbouring Massachusetts Hill property, which yielded $80 per ton for several years after the first discovery, in 1861. Prior to 1896 this mine produced 125,000 tons, from which four to five million dollars worth of gold was extracted. The ore-body continued to about 600 ft., where the vein became barren. The workings were re-opened by the North Star Mining Company in 1896, and the shaft was sunk to 1,200 ft., where a new orebody was cut. This yielded $1,000,000 from 68,000 tons. The shaft was then sunk to 1,800 ft., and new levels were extended, but no ore was found. The mine was closed down in 1901. The profit from this operation was small, but it kept the company active during a lean period in the North Star mine's history.

Taken in conjunction with the later efforts of the North Star Mines Company,

* "The Gold Quartz Veins of Nevada City and Grass Valley Districts, California," by Waldemar Lindgren, 1896.

the successor to the North Star Mining Company, it will be seen that six mines, all rich at surface and near it, were tested, with a sagacity that cannot be impunged. However, one fairly persistent mine in seven is a good average. It is better than the average in the Grass Valley, or any other district.

Charters Towers. A recent incident in Queensland illustrates the practical application of the ideas under discussion. It appears that in June, 1913, a number of mine operators at Charters Towers applied to the Secretary of Mines for a subsidy to be used in the sinking of a deep shaft for the purpose of testing and developing the known veins to a depth of 1,500 ft. below any existing workings. This shaft was to be 4,000 ft. deep. The promoters of the project asked the Government for £60,000, to be spent in three or four years, and proposed to capitalize the undertaking at £300,000. The principal vein system, famous through the Day Dawn and Brilliant groups of mines, has yielded gold valued at £12,400,000 down to a maximum depth of 2,600 ft.

This application for Government aid to deep sinking was duly received by the Secretary of Mines, who then requested E. O. Marks, Assistant Government Geologist, to report upon it.* That report is worthy of careful study. The keynote of it is expressed in the statement: "Mining is a business, though a speculative one, and any expenditure of capital is only to be made when the prospect of reward is commensurate with the risk taken."

The deep workings on the Day Dawn lode are in unproductive ground. "In Mill's United," Mr. Marks says, "a great deal of unsuccessful prospecting work has been done, and the bottom level, 2,543 ft. in vertical depth, could not be very much more discouraging." Again, he says of this mine: "The fact that highly profitable ore was mined in the levels above in irregularly shaped shoots does not give, in my opinion, any real reason for supposing that payable ore is more likely to be found at greater depths below these shoots than elsewhere in shallower ground along the same line of lode."

On the Brilliant lode, the Brilliant Deep has workings down to 2,600 ft. vertically. Mr. Marks says that in these deepest workings "the reef is unfortunately rather patchy in thickness and quality, but the fact that such high-grade ore [crushings yielding up to £14 18s. per ton have been obtained, he states elsewhere] is found at this depth is certainly very encouraging, and shows that depth alone can hardly have been the determining factor in the deposition of the precious metal, and that therefore, depth alone need not be regarded as necessarily unfavourable to the occurrence of ore in the ground, deeper still, in the vicinity of the proposed shaft." But I suggest that the patchiness is ominous. However, that is not the reason for Mr. Marks' unfavourable verdict. He points out, with considerable acumen, that these lodes can be tested much more cheaply by exploring from the bottom of the existing workings than by sinking a shaft, the cost of which may exceed £200,000. Even if the shaft is sunk, the extension of the existing workings will be required. Therefore, he concludes: "While desirous of seeing further prospecting done from the present workings, I am strongly of opinion that the present prospects do not justify the enormous outlay which the shaft contemplated would necessitate."

The Government decided that, in view of the report of the geologist, the application from Charters Towers could not be entertained.

To those who can recollect the temper in which similar projects were discussed 20 years ago, this episode is deeply interesting. Geologists have learned that "mining is a business." Time was when they were neither economic nor economical, but visionary and vain. At Charters Towers, we presume, Mr. Marks is regarded as a repellant pessimist. To me he seems by no means lacking in optimism. He states the case with admirable fairness and yet with some of that indomitable hopefulness that is essential to mining. I was in the mines of Charters Towers when they were in their youth, and recognise that to talk of the persistence of ore in depth in this context is like talking of physical immortality to a man who is about to sign his will.

Tasmania. Another recent incident illustrates the change of sentiment on this subject. The Tasmania mine, at Beaconsfield, Tasmania, was first operated in 1877 by a local company, which in 20 years, up to 1896, had mined 299,000 tons of ore for 371,408 oz. of gold, equivalent to an average yield, by amalgamation, of 24·84 dwt. per ton. In the next seven years the com-

* *Queensland Government Mining Journal*, Aug. 15, 1913.

pany crushed 198,850 tons for a return of 190,435 oz., or 20·05 dwt. per ton. Of the output in this second period, 15·37 dwt. per ton was extracted by amalgamation, and 4·68 dwt by chlorination, suggesting a change in the character of the ore, from perfect docility to incipient refractoriness. The total winning of gold by the local company was £2,094,833, from which dividends to the amount of £772,072 were distributed.

In 1903 the mine (then 800 ft. deep) was acquired by a London company. For the next 10 years an output of 524,842 tons was made, and from this gold to the value of £1,071,114 was won. But no dividends were paid by the London company; this unhappy result being due to an excessive inflow of water and a gradual decline in the yield, from 16·2 dwt. in 1904 to 7·8 dwt. in 1913. Concurrently the proportion of base-metal sulphides increased, so that the yield by amalgamation declined from 13·25 dwt. in 1904 to 4·16 dwt. in 1913. Meanwhile, the mine had been deepened to 1,500 ft., and the prospect was gloomy. The question arose as to whether there was any chance of the ore improving in grade with greater depth. Could an increase of tonnage compensate for the declining yield? Could the cost be lowered so as to turn the current loss into an eventual profit? To decide these points, the managers, John Taylor & Sons, sent Arthur Llewellyn to make a report, in consultation with the superintendent, C. F. Heathcote. Mr. Llewellyn points to the fact that the local company obtained its ore above the 815 ft level, while the London company's operations have been confined to the lower workings. The "recoverable value," that is, the gold extractable, in this part of the mine is estimated, from systematic sampling at the successive levels, to be as follows:

Level. Ft.	Length. Ft.	Average. Shillings.
915	1,200	76·8
1,000	1,200	54·5
1,100	1,200	33·2
1,250	1,200	22·7
1,370	1,200	23·2
1,500	900	28·6

While an improvement is obvious between the 1,370 ft. and 1,500 ft. levels, it must be noted that the lowest level has been driven through three-quarters of the ore-body, the remaining quarter being normally the poorest portion. Making a correction on this account, Mr. Llewellyn considers it likely that the 1,200 ft. of ore on the 1,500 ft. level would average not more than 25·75s. per ton.

However, even this favourable indication must be discounted because the bottom level is notably patchy, the best assays being confined to a length of only 300 ft., instead of being distributed, as was the case in the higher levels, along the entire extent of the shoot, with a gradual impoverishment westward. Therefore, Mr. Llewellyn concludes that "the outlook is not favourable for an improvement of the lode to anything approaching its former productiveness."

The vein follows a fissure at right angles to beds of sandstone, but its productive portion is confined to a relatively short length where conglomerate predominates in the enclosing rock. Beyond the conglomerate the vein splits, and the branches become rapidly unproductive. Efforts to develop the vein along its strike in the limestone and in other sandstone beds beyond the limestone have been "invariably disappointing," so says Mr. Llewellyn. For these and other economic reasons, he joins with Mr. Heathcote in advising against any further expenditure of capital in deeper exploration.

The conclusion is commendable. Twenty years ago a management with such excellent financial backing and with such 'symptoms of indications,' as the Cornishman says, would have sounded the loud trumpet of persistence in depth, and raised £100,000 for the deepening of the mine, uselessly.

THEORIES.

Introduction. So much for the facts of mining; what of the theories of geology? Does the study of ore deposits give grounds for assuming the indefinite persistence of those valuable aggregates of mineral that constitute the essential product of the mining industry? Keenly interested as I am, and as you are, in geological theories, is it worth our while, as mining engineers, to weigh them against the brutal facts of experience?

And still we may be curious to ascertain what geology has to say. As long as the young science was too proud and too ignorant to be utilitarian, it spent itself on fossils and faults, disdaining to dispel the mystery surrounding the origin of ores and veins.

Until a few years ago the geologist gave a formal assent to the talk of persistence, and many was the F.G.S. appended to foolish documents asserting that a particular vein was, or would be, richer in depth. That was in the infancy of economic geology. When, however, the United States Geological Survey, under Clarence King, began to devote itself to the scientific investigation of the mineral resources of a continent and specialists made systematic studies of important mining districts, then we began to see, not the immediate killing of an old fallacy, but the gradual accumulation of the facts that tend to exterminate unsubstantiated generalizations. For a time the amiable desire not to hurt the feelings of the miner and not to dampen his indomitable enthusiasm, caused well informed geologists to leave so delicate a subject severely alone. Hence a discreet silence was observed.

When, at Denver in 1894, I mentioned the matter to John B. Farish, a successful veteran among American mining engineers, he acquiesced in my suggestion that data on the subject of ore persistence should be collected, but he expressed the hope also that such data, having been presented to the Colorado Scientific Society, would be burnt subsequently. When in 1896, while State Geologist, I told the students at the University that men were not born equal, and that mines did not become richer in depth, the local press raised an outcry and protested that I was depreciating the mines of Colorado and undermining democracy in America!

Not until the effects of oxygenated water within the vadose zone were elucidated by Posepny, and the water-level was recognized as a factor in the distribution of rich ore, did the geologist begin to find an explanation for conditions so disconcerting to the miner. When Emmons* and Weed† brought forward an explanation for the secondary enrichment of copper lodes, and others, including myself,‡ followed suit with a discussion of the conditions affecting concentrations of gold in veins, then the miner sat up and took notice, as it were, for the geologist began to touch upon the one subject most important to him, namely, the ascertainment of the factors favouring the making of those extraordinary concentrations of metal for which he has sought since first a pick struck the earth. The miner does not seek merely for gold in quartz, galena in limestone, or cinnabar in serpentine, for example; he seeks for those few spots where the gold, the lead, or the mercury is in a form or in a mass so concentrated as to yield him a handsome profit. Mining is the art of exploiting enrichments, not universal disseminations. There is gold in sea-water, what profits that?

So when theories of secondary enrichment, with all the deductions arising therefrom, began to make headway, they, being true, prevailed over the old generalization, being untrue. One of the first systematic statements on the subject was made, most fitly, by that keen observer and most effective teacher, Waldemar Lindgren, in 1905. In the first volume of *Economic Geology* he wrote an article entitled 'Ore Deposition and Deep Mining.' That article summarizes current theory on the subject we are discussing. Turning to what Professor Lindgren has to say concerning "deposits in which the metal is of later origin than the surrounding rock," we find that he considers contact-metamorphic deposits to be non-persistent. "Slight changes of composition and texture of the rock influence their susceptibility to contact metamorphism to a very surprising degree. Few mines on contact deposits have been worked at a greater depth than a few hundred feet."

Concerning the deposits "in which the ores have been deposited either by filling or replacement along fissures or other paths for underground waters," he states that the views now generally accepted agree in supposing the metallic contents to have been "deposited by ascending hot solutions—in fact, ascending hot springs—and most observers also agree that the metals were taken up in solution by the water at some point considerably below the point of precipitation." He allows that "the conditions for deposition grow less favourable as depth increases, but the change takes place slowly. The richest ores are those near to their original apex." Here he is referring to the erosion of the upper portion of the vein as originally formed. Finally, he concludes, on theoretical grounds, that "ores deposited by ascending hot waters are more likely to show a decrease than an increase in value when developed by deep mining."

* 'The Secondary Enrichment of Ore Deposits,' S. F. Emmons, *Trans A.I.M.E.*, Vol. xxx., pp. 177-217.

† 'Enrichment of Mineral Veins by Later Metallic Sulphides,' W. H. Weed, *Bulletin Geol. Soc. Am.*, Vol. 11, pp. 176-206.

‡ 'The Formation of Bonanzas in the Upper Portions of Gold Veins,' T. A. Rickard, *Trans. A.I.M.E.*, Vol. xxi., pp. 198-220.

* Vol. I, pp. 34-36, Oct.-Nov., 1905.

Erosion. However, this is not all that Professor Lindgren has to say. It is important to note his suggestion that in some cases erosion may have removed so much of the vein or lode that the present outcrop "may be thousands of feet below the point where the fissure reached the surface during the vein-forming epoch." Applying this idea, he distinguishes between veins traversing volcanic rocks of relatively recent geological age and those penetrating older intrusive rocks known to have consolidated at great depth. He argues therefrom that "the date of vein formation" in the latter case "is likely to be remote," as compared with the former. He places the veins of the Southern Appalachian region, of the California gold belt, and of Australia under the first category, while he mentions the mining districts of western Nevada, the Hauraki peninsula in New Zealand, Owyhee county in Idaho, the San Juan region and the Cripple Creek district, both in Colorado, as in the younger class of deposits. Professor Lindgren argues, for example, that because "the thickness of crust removed by erosion in the foot-hill region" of California is 3,000 ft., and the most persistent orebody as yet discovered has been mined to 3,000 ft. vertically, therefore "deposition of free gold ores in the California veins has proceeded to depths from the surface of about 6,000 ft.

This argument assumes that the ore as now found is much the same in character and richness as it was at the time when the vein was first formed; it infers the geological age of the orebody from that of the vein in which it lies. To this I demur. The vein, of course, cannot be older than the rock by which it is encased, but it may be much younger; moreover, the ore as now found by the miner in the vein may be considerably younger than the fissure or other receptacle in which it lies. Furthermore, we are discussing the vertical distribution of enrichments, not of the mere vein-filling, for the orebodies exploited by man represent the selected parts of a lode, not the whole of it, and certainly not its poorer portions. I submit that these rich parts may be the result of later chemical reactions long post-dating the waters that first circulated along the fissure and precipitated the primary ore. In short, I submit that most orebodies, that is, the rich parts of a lode, are geologically young.

The suggestion made by Professor Lindgren, that the present veins may be only the stumps, as it were, of those that at one time, previous to erosion, reached to a former surface several thousand feet higher than their present outcrops, has been advocated with some enthusiasm by persons of less acknowledged authority in matters of this kind. A good example is afforded by the literature on the goldfields of Nova Scotia. In that Canadian province the small veins of gold-bearing quartz follow thin beds of slate belonging to an extensive series of slates and sandstones of pre-Cambrian age. This sedimentary formation has not only been metamorphosed, but it has also been subjected to violent plication resulting in sharp anticlinal folds, so that the quartz veins assume the shape of saddles, which by the further action of cross-folds, become domes.* Apart from these structural features, it is a remarkable fact that this sedimentary formation has a known thickness of 30,000 ft., of which 18,000 ft. constitutes the division characterized by gold veins. It is estimated by E. R. Faribault, J. E. Woodman, and other geologists who have studied the series, that erosion has removed several miles thick of overlying rock. Mr. Faribault writes† of the veins having been "truncated by extensive denudation." Again he says: "Some of the sharpest and highest folds have been truncated to a depth, as far as we know, of over eight miles, exposing at the surface a section of gold-measures of over five miles in thickness." This means that 8 miles has been denuded, and that the gold veins persist through the underlying and surviving 5 miles of rock. His diagrams express the same idea. Several less known writers on the geology of the Nova Scotian goldfields have argued likewise: that the little quartz veins to be seen nowadays at the surface are the stumps of veins that once reached three or four miles to the surface of a former period, and therefore, that these same little veins may persist a few miles deeper, thus arriving at the cheerful conclusion that gold mining in Nova Scotia is "in its infancy."

This reminds me of the cynic's definition of faith as believing things you know to be untrue. It is a brutal fact that of the gold veins that are supposed, on a geological diagram, to have continued several miles overhead and to persist yet for several miles underfoot, not one has been exploited pro-

* 'The Domes of Nova Scotia,' by T. A. Rickard, *Trans. Inst. M.M.*, Vol. xxi., pp. 506-566.

† 'The Gold Measures of Nova Scotia and Deep Mining,' by E. R. Faribault, *Mining Society of Nova Scotia*, 1899.

fitably to a vertical depth of more than 1,100 ft.* Most of the little mines in Nova Scotia that have given good results were abandoned when they were 200 ft. to 300 ft. deep. When a man tells me that a 6-inch quartz vein once penetrated rock, now eroded, for several miles, either way, up and down, from the spot where I now see it, I marvel at his imagination; and when I find that a prospect hole, or at most a shaft only a couple of hundred feet deep, suffices to follow that vein to its vanishing point, I turn to him and say to him that he has stated something that is scientifically untrue.

Eruptives. Even if the actual story of mining in Nova Scotia had not given the laugh to any such assumption of persistence on quasi-geological grounds, we should be warranted in rejecting the theory of continuity on such knowledge as is extant concerning the origin of ore deposits in general. Mr. Faribault and others assume that the granite, of Silurian age, was extruded after the gold veins had been formed, "for granite dikes and veins have been observed to always cut the inter-stratified quartz veins wherever they come in contact with them." This is open to doubt. At Forest Hill the quartz of a vein can be seen penetrating into the granite, and gold is visible both in this quartz and in that of a vein near-by.† Mineralization is usually the sequel to eruptive activity. In Colorado, almost without exception, the deposition of ore over a wide region of great geological diversity is known to have taken place in post-Cretaceous time, and in most cases it can be proved that the orebodies hitherto discovered were formed after the extraordinary eruptive activity of the early Tertiary period.‡ I could quote scores of other examples indicating the geological recency of the agencies with which we are warranted in connecting the deposition of ore, not of the precious metals only, but of the base ones as well. Even if we can find evidence indicating that a vein was formed at an early date in geological history, before erosion had removed thousands of feet of younger rock, it still remains to be proved that the metals enriching that vein were precipitated where we now find them, even at so early a period; and even then it is incumbent on those who argue persistence of ore on the basis of the geologic antiquity of the veins, to prove that the richest portions, namely, the orebodies, were not the product of activities belonging to a much later time. We are entitled to make this last stipulation, because, as I have said, and have elsewhere proved, so many orebodies are definitely known to be the products of the thermal and igneous activities that came into play at a period long subsequent to that in which the veins themselves were first formed. Let me quote the case of the Homestake.

That celebrated lode, in South Dakota, consists of a mineralized zone in schist, known to be of pre-Cambrian age. In the overlying Cambrian conglomerate are found boulders recognizable as eroded fragments of the lode-schist. The conglomerate is a "banket" deposit, for it contains enough gold to be exploited as ore. So here, it has been said, we have proof of a gold-bearing lode of pre-Cambrian age, for is not the debris of it and the gold of it found as a constituent of a Cambrian conglomerate?

As early as 1882 W. B. Devereux indicated that part of the gold in the overlying Potsdam conglomerate was not of mechanical origin, derived from the erosion of the Homestake outcrop, but had been deposited as a chemical precipitate, for he found it as a film between the folia of the schist and in the overlying quartzite.* In 1889, F. R. Carpenter, long resident in the locality as a mine manager, testified that the vicinity of intrusive rock is favourable to the richness of the ore; he concluded that 'porphyry' produced "either an enrichment of the deposit or a further concentration of what gold originally existed in it." In the neighbouring district he ascertained that the silver-bearing galena ores, in Cambrian quartzite, are found "only in connection with the igneous intrusions." In the overlying Carboniferous limestone, ore of similar character is found "adjoining the porphyry, where it cuts through the limestone." He then expressed the opinion that the ore-bearing rocks of the region are "co-extensive with the intruded igneous rocks."† In 1897 F. C. Smith‡ stated that

* The deepest gold mine in Nova Scotia is the Libbey property at Brookfield, Queen's County. This was worked to a vertical depth of 1,062 ft. The incline was 1,807 ft. long, on an angle of 74°. The Lake lode, at Caribou, in Halifax County, was mined to a vertical depth of 1,000 ft. Not more than a couple of dozen mines were exploited to a depth greater than 500 ft. vertical.

† *Trans. Inst.M.M.*, Vol. xxi., pp. 565-566.

‡ "Geological Distribution of the Precious Metals in Colorado," T. A. Rickard, *Mining and Scientific Press*, Jan. 8, 15 and 23, 1910.

* "The Occurrence of Gold in the Potsdam Formation, Black Hills, Dakota," by Walter B. Devereux, *Trans. A.I.M.E.*, Vol. x., pp. 465-475.

† "Ore Deposits of the Black Hills of Dakota," by Franklin R. Carpenter, *Trans A.I.M.E.*, Vol. xvii., pp. 570-589.

‡ "The Potsdam Gold Ores of the Black Hills of South Dakota, by Frank Clemes Smith, *Trans. A.I.M.E.*, Vol. xxvii., p. 405.

neither the Potsdam quartzites " nor those of the Carboniferous are ever sufficiently mineralized with the precious metals to be called ores, except in certain localities where they are cut by dikes, covered by flows, or intercalated with sheets of igneous rocks."

In 1896 I had an opportunity of going underground in the Homestake mine and of visiting neighbouring workings; in 1899 I examined the Hawkeye-Pluma mine, which is in the gold-bearing conglomerate overlying the Homestake lode. I formed an opinion then that the richer portions—that is, the orebodies—of the Homestake schist are associated with the dikes of rhyolite that penetrate not only the overlying Cambrian strata, but transgress a formation of shale belonging to a period so late as the Cretaceous. In other words, it seemed likely that the enrichment of the lode was consequent upon thermal action of post-Cretaceous age.

In 1903 J. D. Irving published a paper on the subject.* He stated that the fact of the Homestake lode being " mineralized and gold-bearing " in Cambrian time " is proved by the presence of gold in the basal or lowest rocks of the sedimentary series which lie in the isolated patches about the outcrop of the orebody." However, Mr. Irving also says, in regard to the Homestake lode: " It appears, first, that there have been several different periods of mineralization, at least one of which has preceded the deposition of the Cambrian rocks, as is distinctly shown by the presence of placer gold in the lowermost gravel beds of the Cambrian series; second, that there have been periods of mineralization which followed the entire deposition of the sedimentary rocks and were later than the intrusion of the dikes and bodies of rhyolite. It is probable that this belt has been the seat of much fracturing and crushing from very early geological time until the present, constituting a line of weakness along which mineralizing waters were permitted to circulate more freely elsewhere."

Finally in 1901, S. F. Emmons and T. A. Jaggar published a paper on the economic geology of the district.† Emmons concluded that while some of the gold in the Potsdam conglomerate is detrital, the high-grade portions of it are indebted to secondary enrichment. This enrichment was contemporaneous with the introduction of the pyrite, which was not an original constituent of the pebbles, but serves now to distinguish the rich parts of the deposit. The pyrite occupies fissures and cavities in the pebbles. Barren conglomerate is cemented by quartzite or by calcite free from pyrite. He refers to the films of gold in the schist.

It is noteworthy that Emmons considers the increase from a grade of 1 dwt. gold to the higher average of 10 dwt. per ton to be due to secondary enrichment. In other words, the formation of that part of the deposit that has an economic value—and is therefore " ore "—is ascribed to the later processes whereby the original detrital deposit was enriched by secondary precipitation. In short, the Cambrian conglomerate became an orebody by reason of chemical precipitation associated with post-Cretaceous, or Tertiary, thermal circulation.

In regard to the Homestake lode, Emmons ascertained that intrusive bodies of rhyolite, of Eocene age, dip and strike with the orebodies, through which they also pass. The breccia formed along their contact with the schist is gold-bearing and pyritic, showing that the " mineralizing solutions must have entered along the channels produced by these movements," which, be it noted, took place in the Eocene period. Thus the gold-bearing ore of the Homestake, once supposed to be wholly of pre-Cambrian age, proves to be largely of Tertiary age; in other words, the enrichment or concentration that changed a zone of crushing into an orebody was not ancient, but young, geologically.

An interesting example of ore deposits undoubtedly due to more than one period of formation is afforded by the copper veins of Butte, in which chalcocite has originated* in at least three ways, and at three different periods:

(1) As a primary deposit from ascending solutions.

(2) As a secondary deposit from ascending solutions, and

(3) As a secondary deposit from descending solutions.

Indeed, the idea that ore deposits as now found came into being at one moment of geological time and have survived unaltered through the continual interplay of natural forces, operating for periods during which the Alps and the Andes might come and go like a child's sandcastles before the encroaching tide, is contrary to all the teachings of geology. On the contrary, the evi-

* " Contributions to Economic Geology," *U.S. Geol. Survey*, 1903, p. 178.

† Professional Paper, No. 26, *U.S. Geol. Survey.*

* Austin F. Rogers, *Economic Geology*, Vol. viii., No. 8, p. 741.

dence, especially that gathered during recent years in connection with secondary enrichment, all goes to show that the orebodies found by man to-day represent the temporary balance between solution and precipitation, between dissemination and concentration. Man, by removing the ore and subjecting it to his own processes, interrupts the work of Nature, who otherwise would continue to modify her handiwork indefinitely. The idea of fixity in the composition of an orebody is as repellant geologically as the notion that the present face of the earth is permanent.

Country-Rock. Among those that have demurred to the conclusion I am seeking to establish is J. Malcolm Maclaren. Two years ago, in discussion of a couple of articles* on this subject written by me, he objected† to one generalization being replaced by another; that is, while he could not support the old idea of general persistence, he considered my sweeping contradiction to be a "pernicious doctrine, if universally applied." He makes a big exception for ore deposits in Archean and Palæozoic rocks, and even several in Mesozoic rocks. Imputing the ore-bearing character of such lodes to a favourable country-rock, he concludes that the richness will continue downward until the lode passes out of the favourable rock. He instances the Mother lode in California, the Treadwell lode in Alaska, and the Champion vein in India to support his conclusions. He endeavours to make a point of differentiating between geologic structure and depth *per se*.

Having regard to Dr. Maclaren's knowledge of economic geology, this may be considered the best argument yet advanced in favour of an idea that I consider ripe for final sepulture. The argument is ingenious, but it amounts only to a red herring across the trail we are trying to follow. Suppose we accept geologic structure as more decisive than depth, does that falsify the argument from world-wide experience? Of course not. Increase of depth involves change of geologic structure. It is most unlikely that a mine opening can be extended downward for many thousand feet without exposing a change of rock or of rock-structure. The probability of change is implicit in the idea of depth.

He refers to the Champion lode, in the Kolar goldfield, and says that "if it should fail at greater depths, it will not on account of the depth, but because the fissure has passed from hornblende-schist into the granite, and so far as I [Dr. Maclaren] know, there is no evidence to indicate such a change is likely to take place." To this I reply that the Champion lode, without marked change of geologic encasement, has become impoverished in depth.

That impoverishment or even abrupt cessation of ore may ensue from a change in the structure or composition of the wall-rocks, is, of course, true. This is an explanation, not a rebuttal, of my argument. We are all aware that ore-shoots have a pitch that takes them beyond one line of vertical descent, and may bring them across another line of descent, as happens to shafts A and B in Fig. 4.

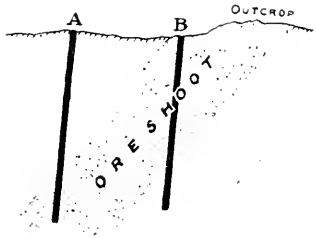


FIG. 4.—Pitch of Ore-Shoot.

We miners are also aware that a 'congenial' rock exercises an important influence upon the distribution of ore. Cornishmen have known that since the days when the Phœnicians came to the Cassiterides. See Fig. 5, where veins are shown more productive in granite than in slate; or else

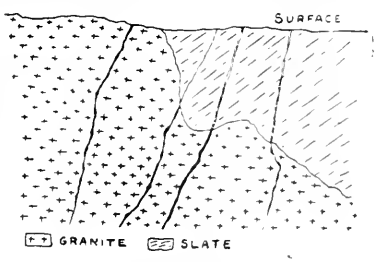


FIG. 5.—Influence of 'Congenial' Rock on Ore Distribution.

* 'Persistence of Ore in Depth,' *Mining and Scientific Press*, Aug. 24 and 31, 1912.

† *Mining and Scientific Press*, Oct. 26, 1912.

they carry copper when in the slate, and tin when in the granite.

Another thing that miners have learned, is that the crossings of veins are likely places for ore, as Fig. 6 illustrates in plan.

FIG. 6. Ore-Body at Crossing.

This is due to the meeting and mingling of diverse solutions at a place where the shattering of the rock causes arrest of the flow and opportunity for reactions, not only between the solutions, but between them and the shattered wall-rock.

Again, the contacts of limestone with dikes or intercalations of irruptive rock constitute conditions familiar to those who mine silver, copper, and lead ores: see Fig. 7.

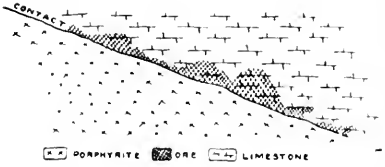


FIG. 7. Contact of Limestone with Irruptive Rock.

A recognised variation from the foregoing is the deposition of ore from vertical veins at horizons where they meet a soluble dolomite or gypsum roofed by impermeable shale, as illustrated in Fig. 8.

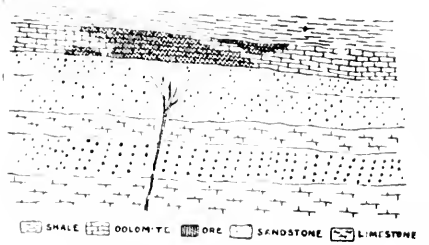


FIG. 8. Influence of Impermeable Stratum on Ore Deposition.

Other examples of the localization of ore might be given, but these suffice. Structural conditions have gone far to determine the formation of ore-bodies. Such conditions may recur in depth, of course, but there the effect of them is marred by an increase of temperature and pressure, acting as a deterrent to the precipitation of ore-forming minerals. Favourable structural conditions *and* favourable chemical conditions are required.

Peculiar geological conditions such as those that conduced to the formation of the Oroya pipe* or the Yankee Girl† chimney become likely as the distance from surface increases. Fractures become less numerous in depth. This is deduced from geophysical principles, and it is confirmed by observation even within the relatively shallow zone penetrated by mine workings. In short, geologists may theorize as much as they please, but the miner can assert sadly that he has never failed yet to bottom any ore deposit he has found. The scientific reasons for it are interesting; the scientific fact is compelling.

Heat and Pressure. On the other hand, the experience of mining finds theoretical confirmation. Deposition of ore is the result of precipitation from solution. Such precipitation is aided by decrease of temperature and of pressure; in depth both temperature and pressure increase. The increment of heat varies between wide limits, being least in undisturbed strata and in regions distant from centres of volcanic unrest, thermal activity, or crustal movement. The average has been determined from systematic tabulation of recorded observations in wells and shafts. Prestwich put it at 1° F. in 48 ft.; a committee of the British Association determined the increment in coal mines as 1° in 60 ft.; at Bendigo it is said to be 1° in 80 ft. by one authority and 1° in 135 ft. by another; in the Lake Superior copper county 1° in 105 ft. is given; in the St. John del Rey mine it is 1° in 126 ft.; while on the Rand 1° in 208 ft. and 253 ft., respectively, has been measured, although later investigations throw doubt upon these earlier measurements, and indicate that the increment is more in accord with observation elsewhere. The increment is not constant vertically, owing, among other reasons, to the diverse thermal diffusivity of rocks. From data now available, 1° in 90 ft. is a fair approximation, with a much more rapid rate in regions

* *Mining and Scientific Press*, Oct. 26, 1912, p. 595.
† *Trans. A. I. M.E.*, Vol. XXXI., pp. 212-214.

of geologic unrest, which are also the places where ore deposition would be in progress. This means that at a depth of 10,000 ft. the boiling point of water may be reached, and at 30,000 ft. the critical temperature of the same liquid would be attained.

As regards pressure, a column of water at a depth of 10,000 ft. would be under a pressure of 5,000 lb. per sq. inch, or 340 atmospheres. The movement that would cause basalt or porphyry to yield, would involve a pressure of 25,000 lb. per sq. inch, while the actual crushing of such a rock would entail a pressure of 60,000 lb. per sq. inch. The temperature evolved by such crushing would be 212° F., apart from the increment due to depth. With these data* as a suggestion, we can infer roughly what must be the heat evolved in the plication and brecciation of rocks enclosing ore deposits.

Deposition of ore is affected by structural conditions. Among those tending to promote precipitation of metallic minerals in concentrated form is the existence of fractures, fissures, or other cavities suitable for the reception of a deposit. Another favourable condition is furnished by sheer-zones and lines of crushing, facilitating those processes of replacement and impregnation to which many large orebodies are traceable. In depth the excessive pressure tends to check the formation of openings in the rocks, which then yield to pressure by undergoing deformation rather than fracture. It has been proved by experiment that many rocks are capable of deformation without rupture when under great pressure.† On theoretical grounds, it is likely that amid the conditions obtaining at great depth, say, 30,000 ft., the only openings in the rock are of sub-capillary size, which means that water would not be free to circulate.‡

How much of the water that acts as an agent in ore deposition is of magmatic or deep-seated origin, and how much of it is of meteoric or superficial origin, is not yet known. In any case, the meeting of these diverse solutions is a prime factor in promoting precipitation, but the zone wherein they mingle is limited by the descent of meteoric water. Deep mines become dry. Hence, on theoretical grounds, we should expect a decrease in the deposition of ore, after passing the bottom of the water-zone.

* Mallet, *Phil. Trans.*, Vol. 163, p. 147.
† Frank D. Adams, *Journal of Geology*, Vol. xviii, No. 6.
‡ Van Hise, "Genesis of Ore Deposits," *A.I.M.E.*, p. 293.

CONCLUSION.

Lateral Exploration. After reviewing the evidence available on this subject, it appears to me—and to my readers also, I expect—that continuity of ore is exceptional, even for limited depths; that the idea of unlimited continuity is untenable in the face of the brutal facts: in effect, non-persistence is normal.

Some there are doubtless to whom my views—or those that I desire to emphasize—may seem destructive of that hopefulness on which mining is nourished. All that I ask is for a transfer of that hopeful spirit from directions in which it is likely to be wasted to others in which it may become justified. If the money so often squandered on impoverished mines in the vain expectation of betterment in depth, or of indefinite persistence, were diverted to lateral exploration in localities highly mineralized, or spent in systematic trenching in new regions, we should, I believe, give to such capital the fruitfulness that is the economist's constant aim. In short, we must employ the cross-cut more and the shaft less, except, of course, where the orebody can be kept in view.

Incidentally, I desire to commend the increasing use of the diamond-drill as a means of ascertaining the position of a lode and its general character in depth or at a distance from existing workings. While the drill is not a safe instrument for sampling a lode, it provides a highly economic means of obtaining that preliminary information on which wise development depends.

The first rule in mining is to find ore; the second is to follow it. If the crust of the earth in a productive mining region were suddenly rendered transparent to the radio-active vision of a transcendental geologist, he would see, I feel assured, many an orebody that had been missed in the hurry to follow the lure of deep sinking. The fallacy of enrichment in depth has been the cause of much useless expenditure in deep exploration; the idea of persistence has led to the neglect of lateral development. Dr. Maclaren has said* that "future industry must more and more concern itself with concealed orebodies; it is here that a universal acceptance of Mr. Rickard's proposition would prove exceedingly harmful both to the mining engineering profession and to mining capital." I refer to this remark not in any particular antagonism to Dr. Maclaren, but because I regard him as

* *Mining and Scientific Press*, Oct. 26, 1912, p. 547.

one of the foremost exponents of mining geology. If I assert that orebodies usually become impoverished in depth and finally become unprofitable, it is not to imply that prospecting in depth is universally undesirable. But I do mean that it is unlikely that every known orebody will persist indefinitely, and that evidence of eventual impoverishment vertically must be faced in a scientific spirit. The corollary is not that deeper search must be abandoned hastily, especially in mines that are relatively shallow, but that in many cases lateral exploration, while cheaper, is also likely to prove more successful than further sinking. As regards four big mines* with which I was professionally connected, I took the view that their prospects in depth were poor, but that lateral exploration looked promising. Three of these mines were in Colorado, namely, the Enterprise, Independence, and Camp Bird; the fourth was the Commonwealth, in Arizona. Only in one case—the Camp Bird—did I consider that the geological conditions in depth were at all favourable. The Enterprise orebodies were on a shoot-gypsum contact fed by small veins that ended at this contact and were themselves impoverished within 100 ft. to 150 ft. below that contact. My diagnosis has been confirmed. Deeper prospecting has proved futile. The Commonwealth lode is in a rhyolitic breccia lying on sandstone; when the lode reaches the sandstone it dies: the breccia-sandstone contact pitches eastward, hence the lode is productive more deeply in that direction. This interpretation of facts has been corroborated. In examining the Independence in 1898, I found that the vein on the 8th level was thin, although rich, and on the 9th level it was not worth sampling. I did not recommend the sinking of the shaft, because depth entailed departure from the particular conditions known to favour the finding of ore in this mine. On the contrary, I advised lateral exploration along and between the series of veins exposed in the workings. This proved highly successful. The shaft was sunk, several years later, to a depth of 1,400 ft., but the deeper search for ore proved decidedly unprofitable. The Camp Bird, at the time of my examination, in 1899, had a fine orebody that promised to extend westward, in which direction the contour rose so as to increase the distance to surface from 400 ft. to 1,200 ft., affording, therefore, a block of promising ground fully 800 ft. more in height. I advised exploration in that direction. The subsequent history of the mine has shown that this reasonable expectation was warranted, while the sinking of the shaft below the main adit yielded unsatisfactory results, ending in the exhaustion of the mine.

My own experience has been that mine-owners often squander their funds in expensive shaft-sinking when the chances of finding ore by cross-cutting or diamond-drilling have been scarcely tested. I take issue with Dr. Maclaren when he suggests that the effort to smash a fallacious generalization is likely to prove "exceedingly harmful both to the mining engineering profession and to mining capital." The profession must recognize facts if it is to make a scientific use of current knowledge in the exploitation of ore deposits. Capital is not going to gain, legitimately, by spending money on forlorn hopes in depth, while neglecting chances in the shallow zone of mining.

* Each of them has yielded over $10,000,000, and the four of them have made a gross output of $75,000,000.

GOLD AND PRICES.

(Read at September Meeting, 1914.)

By Prof. R. A. Lehfeldt, D.Sc.

Discussion.

Mr. Alex. Aiken (*Life Member*): Prof. Lehfeldt's definition of Currency appears to me to be both inaccurate and incomplete. He states that it consists of metallic and paper money and bank deposits. In so far as his attempt is concerned to compare currency with the total sales of goods, it is clear that his definition of currency must be very largely extended. Large volumes of goods pass hands in the first place on merely entries in books. The debts so recorded may be liquidated in various ways, some by bills drawn by the seller on the purchaser, others by cheques remitted by the purchaser, and in still other cases little currency passes at all where people deal with each other as both buyer and seller. It is clear that the total sales of goods cannot under such circumstances agree with the product of the currency and its velocity of circulation unless currency is to have a wider meaning than the Professor has given it. On the other hand, his definition is inaccurate in that he includes bank deposits.

as currency. He may mean by this phrase, cheques which may fairly be termed currency, but cheques are not the same thing as bank deposits. The latter, which simply consist of the credit balances due to the customers of the banks, are not the limit of the ability of the community to draw cheques. That limit is the credit which the banker is prepared to give by way of overdraft to his clients. It is possible that the Professor considers this credit to be limited in turn by the total deposits lodged with the banks, in other words, that were all the clients of a bank to withdraw their balances the bank would not be in a position to grant an overdraft. In this sense it is competent to argue that the total of bank deposits is the limit of the power of a community to issue currency in the form of cheques at any particular moment. This may entitle the Professor to claim bank deposits as potential currency, but it is clearly inaccurate to call them actual currency. Take, as an instance of the underlying fallacy, the case of a person who wishes to borrow £100. If he goes to his banker, and the latter, as often happens, opens a special account for the advance and places the £100 at the credit of the borrower's ordinary operative account, the banker has, according to the Professor, increased the currency of the community by £100, and after the customer has passed his cheque, the bank's deposits are at the same figure as before the operation. But assume that the banker declines to make the advance and the borrower gets a cheque from another client of the bank, as a loan, then the currency is decreased on the Professor's theory by £100. There is no difference in the real currency which has passed in both cases.

The expression "Goods," as used by the Professor when dealing with the total sales of goods in a community, must, I presume, include services which have also got to be paid for by currency. Will they, however, also include purely financial operations which frequently largely affect questions of currency?

The inference from the Professor's paper is that the cause for the rise in prices during the last few years is due to an increased production of gold. In other words, that had the production of gold been less, the rise in prices would not have happened. This is very controversial, and I do not think that anyone has succeeded in proving the theory satisfactorily. That the quantity of gold must have some bearing on price I know of no one that will deny, but so many other things than the quantity of gold also have an effect on price, that there are many economists who maintain that the effect due to the changes in the quantity of gold is slight compared to the results of other factors. There is one point in regard to the quantity of gold which I personally have never been able to satisfactorily explain on the quantity theory. The rate of discount settled week by week by the Board of the Bank of England is based on the amount of surplus gold it possesses, having regard to the bank's liabilities. Now, if more gold were coming into the market than it required to carry on the business as it stood at any particular moment, it is legitimate to assume that in the free market for gold, of which London is the centre, it would show itself, it nowhere else, and were it to do so, the first effect one would anticipate would be a reduction in the discount rate. This has never been used as an argument by the quantity theorists, for the reason that the trend of discount rates fixed weekly by the Bank of England do not support the theory, nor have I seen any attempt to explain why the hypothetical phenomenon has not appeared. Accordingly, for my part, I think that the Professor's thesis is still unproved.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Detection of Antimony in Qualitative Inorganic Analysis.—"The sulphide precipitate, containing possibly arsenic, antimony and tin, obtained in the course of qualitative analysis, is treated with sodium peroxide, in the presence of a little water, the mixture heated to boiling, and any precipitated stannic acid filtered off. Antimony is indicated by a separation of sodium metantimoniate on cooling. Arsenic may be detected in the same solution by adding concentrated hydrochloric acid and then Bettendorff's reagent (stannous chloride in fuming hydrochloric acid)."—J. Petersen, *Z. anorg. Chem.*, 1914, *88*, 108.—*Journal of the Society of Chemical Industry*, Sept. 15, 1914, p. 891. (J. A. W.)

Action of Hydrogen on Metals.—"From an aqueous solution of $HgNO_3$ the author succeeded in separating crystals having the composition $2Hg_2O$, $N_2O_5H_2O$ by treating the solution in a sealed vessel with hydrogen at a temperature of 160° C., and 130 atmospheres pressure. The crystals exhibited, under microscopic observation, distinct spherical globular quicksilver drops. At a temperature of 225° C. metallic Hg mixed with Hg_2O separated from the solution, the latter having an acid reaction."—A. K. Starynkiewitch.—*Australian Mining Standard*, Sept. 3, 1914, p. 216. (J. A. W.)

DETERMINATION OF CARBON MONOXIDE.—"A modified form of the Haldane apparatus is used. The measured gas sample is first treated with bromine in potassium bromide solution, followed by potash; the loss in volume gives the carbon dioxide, hydrogen sulphide, sulphur dioxide, and unsaturated hydrocarbons. It is then passed backwards and forwards, by means of the potassium hydroxide solution containing it, through a U-tube filled completely with powdered iodine pentoxide previously heated for 3·4 hours in a current of dry air at 200° C. Before and after this operation the U-tube stands in cold water of known temperature; during the passage of the gas in water at 90—100° C. The loss in volume gives the carbon monoxide. Mixtures rich in hydrogen and carbon monoxide require dilution with air to bring the hydrogen below 20 and the carbon monoxide below 8; otherwise hydrogen is oxidised. An accuracy of 0·02 is possible."—J. L. GRAHAM and T. F. WINMILL, *Chem. Soc. Trans.*, 1914 *105*, 1996-2003. *Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 944. (J. A. W.)

DETERMINATION OF THE LIME REQUIREMENTS OF SOILS. "From 10 gm. to 20 gm. of the soil are treated in a bottle with 300 cc. of approximately *N*/50 calcium bicarbonate solution, the air in the bottle is displaced by carbon dioxide, and the mixture shaken for 3 hours. The solution is then filtered, and 150 cc. of the filtrate are titrated with *N*/10 acid, using methyl orange as indicator. The difference between this titration and the titre of the calcium bicarbonate solution is taken to represent the amount of calcium carbonate absorbed by the soil. The calcium bicarbonate solution is prepared by suspending an excess of calcium carbonate in water, saturating with carbon dioxide, filtering, and diluting the filtrate with one-third of its volume of water." H. B. HUTCHINSON and K. MACLENNAN, *Chem. News*, 1914, *110*, 61-62.—*Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 932. (J. A. W.)

A NEW METHOD OF SEPARATING MANGANESE FROM IRON. "O. Hackel describes an analytical method of separating Fe and Mn by precipitating quantitatively the iron from a neutral solution of ferrous sulphate, by boiling with potassium chlorate, adding zinc oxide (the MnO remaining in solution). The proportions used are 0·2 gm. of Fe, requiring 2·5 gm. of $KClO_3$ and 1·3 gm. ZnO, being diluted with 500 cc. H_2O. The estimation of Fe and Mn is to be carried out volumetrically. The method gives most exact results, and the author proves by a series of control-analyses that the best results are obtained by dissolving the iron-precipitate in HCl, converting this solution into the sulphate, reducing with zinc and titrating the $FeSO_4$ thus obtained. The manganese-precipitate is to be evaporated with HCl, converted into the sulphate, neutralised by Na_2CO_3, acidulated with HNO_3, and the Mn titrated after Volhard's method."—Sonderabdruck, Jahrbuch Imp. Roy. Geolog. Reichsanstalt, Vienna, Vol. 63, p. 151. —*The Australian Mining Standard*, Aug. 6, 1914, Vol. liii., No. 1343, p. 133. (J. E. T.)

SEPARATION AND DETERMINATION OF TUNGSTIC ACID. A NEW USE OF NITRON.—"The weighed portion of WO_3 is moistened with H_2O and dissolved by aid of heat in smallest possible amount of strong NaOH. Dilute the clear solution to about 150 cc. with boiling H_2O, acidify with AcOH, heat to boiling and with vigorous stirring add slowly an excess of Busch's nitron acetate solution; cool and let stand till supernatant liquid is clear. Filter on paper, Gooch, Neubauer or other filter and wash with a mixture of 4 cc. of precipitant + 96 cc. H_2O till filtrate is free from alkali. Burn the moist or dried precipitate in a Pt crucible, igniting at about 800° to constant weight and weigh WO_3. With alkali tungstates dissolve in H_2O and proceed as given. Excellent results are reported, using WO_3. The precipitate cannot be weighed direct, decomposing readily, though very difficulty soluble in H_2O, dilute AcOH, alcohol, etc. The white precipitate soon becomes yellowish, apparently due to NH_3 in air. If solution is boiled after adding precipitant the supernatant liquid remains turbid and filtration is very slow."—A. GUTBIER and G. L. WEISE, *Z. anal. Chem.*, *53*, 426-30.—*Chemical Abstracts*, Aug. 20, 1914, p. 2858. (J. G.)

MICA AND ITS USES.—"The recent discovery of mica deposits on the north bank of the Oliphant's River, Transvaal, is likely to have some effect upon the mica market. Mica at present comes principally from India, the chief producing districts being those of Hazaribagh, in Bengal, and Nellore, in Madras, but Canada and the United States also produce small quantities. Of recent years the uses of mica have largely increased. For insulating purposes it is without an equal, and is used in large quantities in modern high tension electrical appliances. The spread of wireless telegraphy has also largely increased the demand for mica, while it has varied uses as transparent fire resisting material. Waste mica is used in the manufacture of lubricants and as an absorbent of nitro-glycerine in the preparation of certain explosives. The present value of mica ranges from about £100 to £3,000 per ton, according to the size of sheets of clear mica produced."—*The Chemical World*, Sept., 1914, p. 266. (J. A. W.)

CYANIDE TO BE MANUFACTURED IN THE UNITED STATES.—"The stoppage of the foreign supply of cyanide and cyanogen compounds creates a serious situation, not only for the mining industries (extraction of gold and silver), but also for many other electrical and chemical trades using cyanide. The means to meet this temporary difficulty can be easily devised. The raw materials from which cyanogen and its compounds are prepared are abundantly found in our country. This source of raw products are many and will be briefly reviewed.

A considerable quantity of cyanides escapes from the blast furnaces, as fumes and flue dust, from which large amounts of potassium cyanide can be recovered, by methods analogous to those in practice to purify coal gas. Cyanide is produced not only in iron blast furnaces, but also in large quantities in copper and lead furnaces and in a general way in any metallurgical apparatus wherever the nitrogen of the blast acts on hot carbon mixed with alkalies. Fumes and gases issuing from these producers can be condensed, and the cyanide contents separated thereof.

In the purification of coal gas by hydrated iron oxide, large amounts of cyanide are continually obtained; this has been an old-time source of supply.

The extension of the by-product coke ovens, replacing the wasteful bee-hive ovens, constitute an abundant source of nitrogen products from which large quantities of cyanides can be obtained.

The beet sugar industry is another well known source of cyanide, which is obtained as one of the by-products, by the treatment of residual vinasse.

Formerly most of the potassium ferrocyanide from which other cyanogen compounds were made was obtained by fusion of dry refuse animal matter at a red heat, together with potassium carbonate and iron filings. About 3,000 tons of cyanides were yearly produced by this method. The scarcity of hoofs, horns and scrap leather and the competition of synthetic cyanides have put an end to this old method.

Cyanides can also be made from atmospheric nitrogen: by synthesis between carbon disulphide and ammonia; by ammonia, charcoal and alkali, and from sulphocyanides.

There are several hundred patents (many of them in operation) which have been applied for during the last 20 years, relating to the preparation of cyanides from carbides, from nitrides, etc.

The enumeration of the more practical methods for the preparation of cyanides by these various methods would be too long. These brief notes show plainly the abundance of materials which we have within our reach, to create an independent national industry which could not only produce all the cyanide required to supply our home industries, but also to reverse our former trading practice of importing cyanides by becoming exporters of our excess of production to other sections of our continent, to Australasia, South Africa, and to the eastern gold producing countries."—Baxeres de Alzugaray. —*Mining and Engineering World*, Aug 22, 1914, p. 334. (J. C.)

Annual Report of the Chief Inspector of Explosives for the Union of South Africa.—"*Accidents.*—Six accidents have occurred in manufacture, of which the chief were: A fire of empty nitrate bags; an explosion in a packing house in which cartridging, since stopped, was also being carried on (see Special Report, this *J.*, 1913, 627); a fire in a nitration house, due to the inadvertent supply of more glycerin than could be taken by the distributor, which, the stirrer having stopped, fired the supernatant nitroglycerin; an explosion in a kneading house supposed to be due to the dropping of a box of gelatin, or to a blow on the edge of a kneading pan being charged (see Special Report, this *J.*, 1913, 1171). In addition there were 61 accidents with explosives or other dangerous substances, including 'outrages,' but excluding mining accidents. *Importations.*—Gelatins, gelignites, dynamites, and cheesa sticks are made in South Africa; blasting and whaling powders, safety fuse, detonators and electric detonators, sporting powders and ammunition, and fireworks are imported. Statistics are as follows:—

	Blasting Gelatin.	Gelignite.	Dynamite.	Blasting Powder.	Permitted Explosives	Total.	Detonators (Number).
Manufactured, cases ...	484,314	240,097	183,729	—	1,643	909,783	
Imported, cases ...	1,371	1,300	2,638	4,363	1,993	11,665	13,562,900
Exported, cases	10,863	17,734	948	—	—	29,545	31,525

20,321 cases of 500 coils each of safety fuse were imported."—A. B. Denne.—*Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 943. (J. A. W.)

Dissociation of Hydrogen into Atoms.—"Evidence in favour of the view that at high temperatures hydrogen is dissociated into atoms is summarised as follows:—The loss of heat from electrically heated tungsten wires in nitrogen, mercury vapour, argon, and carbon monoxide varies with the temperature exactly in accordance with theory (*Phys. Rev.*, 1912, *35*, 401). In hydrogen, however, the loss of heat rapidly increases above 1,900° K (° signifies absolute scale of temperature) and at 3,500° K is more than four times the calculated value; no secondary electrical effects capable of explaining the increase can be detected. At low pressures the loss of heat from tungsten wires is greater than at atmospheric pressure in hydrogen, but not in other gases. When a wire of tungsten, platinum or palladium is heated above 1,300° K. in hydrogen at a very low pressure (0·001—0·02 mm.), the hydrogen slowly disappears. It is not absorbed by the wire but is deposited on the glass (sometimes in tubing a long distance from the bulb), especially when this is cooled by liquid air; ordinary hydrogen is liberated when the cooled portions are heated. The hydrogen which deposits in this way on glass is very active chemically, reacting with oxygen and phosphorus at the ordinary temperature. It is capable of diffusing through long tubes (at low pressures) and will then dissolve in platinum (at 50° C.), causing a marked increase in the electrical resistance of the metal and a corresponding decrease in its temperature-coefficient. The active hydrogen is not affected by an electrostatic field and therefore does not consist of hydrogen ions. Tungstic oxide, platinum oxide, and other substances contained in a tungsten filament lamp together with hydrogen at very low pressure, are rapidly reduced when the filament is heated above 1,700° K., although otherwise they are not affected by hydrogen. (See also Langmuir, *J. Amer. Chem. Soc.*, 1912, *34*, 1310; Freeman, *ibid.*, 1913, *35*, 927.)"—I. Langmuir and G. M. J. Mackay, *J. Amer. Chem. Soc.*, 1914, *36*, 1708-1722.—*Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 919. (J. A. W.)

Preparation of Colloidal Solutions of Precious Metals.—"Dr. H. Gramert, of Halensee, uses the following method for the preparation of colloidal Au, Ag, Pt, Pd, and Ir, which is quite a departure from all known processes. He prepares an alkaline solution of protalbine-acid or lysalbine-acid, and silver or gold, and by heating and dialysing the solution, the gelatinous residue contains the colloidal metal which, in the case of Ag, is obtained through the reducing action of the gelatine upon the silver nitrate, the action being facilitated by adding formaldehyde. According to the results, it is proved that serum-albumin can be utilised as an excellent protecting agent for colloidal metals. A lasting solution of colloidal silver is prepared by reducing silver-nitrate in presence of blood-serum. An addition of alkali is not required in this case, and if proper reduction-agents are used, even the dialysis can be dispensed with. Similar Ag, the other precious metals, gold, platinum and iridium can be converted into colloidal solutions in presence of serum-albumin. For instance, 45 gm. of blood-serum are mixed with a solution of 0·85 gm. of

nitrate of silver dissolved in 4 gm. H_2O and adding a 50% solution of hydrazin-hydrate until the $AgNO_3$ is completely reduced. After an active liberation of nitrogen a solution is formed which appears dark red-brown in convergent light, representing an absorption-compound of silver and serum-albumin. In ordinary light the solution has a greenish-grey colour."—H. GRANERT.—*Australian Mining Standard*, Sept. 3, 1914. p. 246. (J. A. W.)

THE TOXICITY OF OIL PAINT VAPOURS.—"The hygienic importance of the vapours given off by drying paints had not until quite recently been realised, but now that some of the effects of these vapours have been determined, a great deal of attention is being paid to the subject. It was not till 1912 that the subject received any attention of note in England, but since that date Goadby,* Armstrong and Klein† have in different ways approached the question, and as a result have established the fact that serious physiological disturbances are produced by the inhalation of paint vapours, the toxic effects of which they ascribe to the volatile thinning constituent of the paint. These volatile constituents produce definite toxic effects, and have been described.‡ Recently Gardner** claims to have traversed the work of former workers, and in a paper, entitled 'The Composition of Paint Vapours,' describes certain experimental work which leads him to the following conclusions:—

(1) When linseed oil or similar drying oils are spread in thin layers, the absorption of oxygen which takes place is accompanied by the evolution of considerable amounts of carbon dioxide and organic substances. Carbon monoxide is also evolved in small amount.

(2) Oil paints containing lead or zinc pigments do not emit volatile compounds of a metallic nature.

(3) Drying paints evolve water-soluble acid substances, such as formic acid, as well as acid substances which are apparently of a fatty nature. Carbon dioxide and carbon monoxide are also present in the vapours from the drying paint. The type of pigment used in the paint may directly affect the amount and character of the volatile substances produced. Basic pigments apparently stimulate the evolution of such products.

(4) Aldehydic substances are present in the vapours from drying oil paints. These substances probably have a marked bactericidal effect upon pathogenic bacteria, thus accounting for the sanitary value ascribed to oil-pigment paints.

In a subsequent communication,†† entitled 'The Toxic and Antiseptic Properties of Paints,' by the same author, read before the thirtieth annual convention of the International Association of Master House Painters and Decorators of the United States and Canada, held at Indianapolis, on February 12th, 1914, the subject is further discussed, and the hygienic bearing of the work is enlarged upon.

In further discussion as to the influence of vapours of turpentine or similar volatile thinners on blood corpuscles (which, according to Armstrong and Klein, is such that the observed changes can be ascribed to these vapours) Gardner states that he "is convinced that the change in the blood corpuscles of a person who had breathed the fumes of drying paint in an unventilated room is more likely due to the effect of the carbon monoxide evolved by the paint. The use of turpentine doubtless stimulates the oil present to give off larger quantities of such gas, but the effect of turpentine should be cited as a contributory effect in this respect. Carbon monoxide if present in appreciable percentage, is really the most poisonous constituent of the vapour from drying oil paint.'

The rest of the article is a polemic against Gardner's conclusion that CO is formed when paint dries, the statement being made that experiments now in progress indicate that the CO formed is due to the action of the fuming sulphuric acid on the formic acid which is undoubtedly formed, no precautions having been taken to eliminate this substance before passing the gases into the fuming sulphuric acid preparatory to the determination of the CO by the liberation of iodine from I_2O_5."—C. A. KLEIN.—*The Chemical World*, Sept., 1914, p. 252. (J. A. W.)

NEW TEST FOR MORPHINE [AND COPPER].—"On adding a few cc. of hydrogen peroxide and a small quantity of strong ammonia solution to a solution of morphine or one of its salts, and stirring with a copper wire, gas is evolved and a deep port-wine coloration is produced; 0·00002 gm. of morphine can be detected by the test, but with very small quantities, a few drops of potassium cyanide must be added after stirring, to destroy any blue colour due to copper. Monomethylamine may be used in place of ammonia, but not dimethylamine nor caustic soda. Apomorphine gives an orange coloration under similar conditions, but codeine and dionine (ethylmorphine), which are indistinguishable from morphine by ordinary tests, do not give the reaction, nor do strychnine, brucine, atropine, narcotine, heroine, cocaine, nicotine, cephaline, emetine, veratrine, physostigmine, pilocarpine, and coniine. Copper (0·00001 gm.) may also be detected by the test."—T. H. OLIVER, *Chem. and Drug.*, 1914, *85*, 249.—*Journal of the Society of Chemical Industry*, Sept. 15, 1914, p. 885 (J. M.)

INVESTIGATIONS ON THE WHITE WINES OF SOUTH AFRICA.—"Statistics are given relating to the imports of wines from Cape Colony to the United Kingdom from 1857 to the present time. The value of the annual imports at present is £1,000—£2,000, representing about 2% of the value prior to 1860, when the heavy preference given to Colonial as compared with foreign wines was discontinued. The wines formerly imported were mainly fortified products of the type of port and sherry. Of the light unfortified wines of to-day the red varieties are much more satisfactory and uniform in quality than the white wines, as the former requires less skill in the making, and are much less sensitive to certain diseases. The white wines of the Cape are peculiarly liable to *casse* or turbidity, accompanied or preceded by an increase in the colour of the wine, a more or less pronounced bitter flavour, and a very characteristic earthy odour or *terroir*. The malady, which possesses the general character of *casse ferrique* is not of biological origin, but is conditioned by purely chemical changes. The essential conditions for its appearance are access of oxygen, the presence of iron salts which act as carriers of oxygen, and the presence of certain oxidisable substances of the wine, notably tannic acid. A mere trace of iron, such as may be extracted

* Goadby, 'Lead Poisoning and Lead Absorption,' Arnold, London, 1912, 108.

† Armstrong and Klein, *Journal Soc. Chem. Ind.*, 1913, Vol. 32, 7, 320.

‡ Klein, *Chemical World*, 1914, Vol. 1, 7.

** Gardner, *Journ. Ind. and Eng. Chem.*, 1914, Vol. 6, 91.

†† Bulletin 11, Educational Bureau Paint Manufacturers' Association of the U.S., reprinted in *Oil and Colour Trades Journal*, 1914, Vol. 45, 1009.

during filtration through asbestos or decolorisation with animal black, sulfides, by alternate oxidation and reduction, to produce a large cumulative effect when the supply of free oxygen is not limited and sufficient time is given. The *casse* itself consists of a colloidal combination of iron with products derived from the limited oxidation of tannin and certain proteins, and it may be distinctly perceptible when amounting to only a few milligrams per litre. At ordinary temperatures the whole of the oxidised products do not separate from a wine which has undergone *casse*, and a distinct improvement in the flavour and colour of the wine may be effected by chilling for some days at 31°–32° F. (0° C.) and filtering cold. Addition of sulphurous acid (potassium metabisulphite), even within the limits allowed by the Adulteration Act of the Union, 1913 (200 mgm. per litre for dry wines, and 357 mgm. for others), has a marked effect in preventing *casse* and in restoring faulty wines to their original state. Citric acid diminishes the tendency of a wine towards *casse*, but tartaric has little effect. Addition of tannin to white wines is deprecated, and in order to prevent its presence in more than a trace, it is advisable to effect a rapid separation of the juice from the skins and a judicious cutting off of the 'press wine' from the bulk. Fining with isinglass or egg white, or better still, filtration, is preferable to the tannin-gelatin process of fining. Further means of guarding against *casse* are quick fermentation with selected yeasts and well controlled by attemperation, cold storage and maturation of the wine, and the avoidance of residual air in the bottles. Several tests are described by which the tendency of samples of wine to develop *casse* may be compared :—(1) Treatment with '10-vol.' hydrogen peroxide at the rate of 10 cc. per litre; most wines will show enhanced colour or turbidity within 6—7 hours, and faulty samples within a few minutes. (2) Treatment with 0·2 cc. of a 2% solution of ferrous sulphate per 100 cc.; a satisfactory sample may show no turbidity for 14 days. (3) Ten cc. of the wine are treated with 0·5 cc. of *N*/10 sodium hydroxide solution in such a manner that the latter sinks in the wine and forms a separate layer. At the junction of the two liquids, *i.e.*, at the zone of neutrality, there is a more or less purple ring below which is a zone varying in colour from straw yellow to a very dark sherry colour. The intensity of the colour of these two zones corresponds to the tendency of the wine to undergo *casse*. There appears to be no connection between the presence of oxydases in wine, and the *casse ferrique*. The grape contains, in quantities differing according to the variety, an oxydase (peroxydase) which occurs in the inner cells of the epidermis, and in the vascular bundles underlying the skin and those connecting the pips with the stalk. Freshly expressed grape juice contains no oxydase, but oxydase reaction (with benzidine and hydrogen peroxide) appears when the juice has been in contact with the skins for a short time. Juice which contains a considerable quantity of oxydase has a tendency to become brown on exposure to air. This is particularly notable with the white French variety of grape which is specially rich in oxydases. The so-called 'light' wines of South Africa are much higher in alcoholic strength than the average European wines of a similar class, owing to the higher gravity and sugar content of the must before fermentation, dilution being forbidden by law. This in itself appears to favour *casse* in the white wines, and certainly detracts from their value as beverages for general table use. The author is convinced that a dilution of the must to arrest the gravity of the musts of the light European wines would tend to improve the character of the South African wines and that the prohibition of this dilution by law and of the addition of citric acid are directly detrimental to the quality of most of the wine produced there." H. T. BROWN.—*Journal of the Society of Chemical Industry*, xxxiii., July 31, 1914, p. 762. (J. A. W.)

METALLURGY.

DETERMINATION OF IRIDIUM IN PLATINUM IRIDIUM ALLOYS.—"The following methods were used for the analysis of two platinum iridium alloys containing respectively 20 and 25% Ir:

Silver Method.—0·25 gm. of alloy and 1·5 gm. of pure silver were cupelled (finally at a high temperature) with 10 gm. of assay lead, the buttons rolled in to cornet form as usual, and 'parted' with boiling concentrated sulphuric acid (2 successive portions of 40 cc.); the residue was digested first with 10 cc. and then with 5 cc. of dilute *aqua regia* (HCl 4, HNO_3 1, H_2O 10, parts by vol.) to dissolve the platinum, then washed with water, dilute ammonia (to remove AgCl) and water in succession, ignited at a high temperature, and weighed as iridium.

Lead Method.—0·5 gm. of alloy was heated for one hour (in a wind furnace) with 5 gm. of assay lead in a carbon crucible contained in a covered clay crucible packed with charcoal, and allowed to cool slowly. The lead button was digested with two successive portions of 50 cc. of dilute nitric acid (1 : 10), the insoluble residue treated with dilute *aqua regia*, and the process concluded as in the preceding method. In both methods platinum was determined (in the *aqua regia* solution) by precipitation with pure zinc and subsequent ignition of the metal in hydrogen; but it is customary to take this constituent by difference. The results obtained were as follow :

Approximate Composition of Alloy.			By Silver Method.	By Lead Method.
		Per cent.	Per cent.	Per cent.
Iridium	...	20	20·20	20·10
			20·40	20·40
Platinum	...	80	79·76	79·30
			79·52	79·60
Iridium	...	25	25·20	25·32
			25·24	25·28
Platinum	...	75	74·68	74·52
			74·56	74·60

C. O. BANNISTER and E. A. DU VERGIER, *Analyst*, 1914, *39*, 340-346.—*Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 922. (J. A. W.)

ORE TREATMENT AT KALGOORLIE.—"The ore landed at the pit head is composed of tough, hard silica and softer carbonated quartz dolerite, and the precious metal is irregularly disseminated through it as 'free' gold, as gold telluride, or locked up in the pyrites.

The management proposes to treat 7,000 tons of this material per month.

The ore will be delivered over a grizzly the oversize to a 30 × 42 Big-elow rock breaker and will then be conveyed to a bin; from the bin it will be fed into three No. 8 Krupp ball mills and ground to pass a 100-mesh screen; thence by conveyor to three 100 ft. long centre and end fired Edwards' roasters. From the roasters a wide push conveyor will take the roasted ore to a mixer. In this mixer the dry ore meets, and is mixed with, overflow water from mechanical thickeners and spent cyanide solution.

from the zinc boxes. From the mixer the pulp is elevated by means of an air lift to eleven grinding pans of the ordinary type—seven 5 ft. in diameter and four 8 ft. in diameter. It is intended to "slime" (grind fine enough to pass a 200 mesh sieve) the whole of the ore in these pans. Amalgamation will be used at this stage to separate any coarse gold from the pulp.

A second air lift will elevate the slime from the pans to mechanical thickeners, where excess water will be removed. The thickened slime will then be taken by a third air lift and raised to agitation vats. To handle the proposed tonnage it is intended to employ two of these mechanical thickeners, each 25 ft. in diameter. In the agitating vats the cyanide solution will be brought up to its full strength—0·05 per 0·03%—and here the chief extraction of the gold will take place.

It is confidently anticipated that practically the whole of the fine gold will be dissolved in the agitation vats, and it will only then remain to separate the gold bearing solution from the valueless slime, and this will be effected by means of a vacuum filter of the Ridgway type, capable of treating 250 tons of dry slime per day.

Dry crushing mills are proverbially dusty, and this is disadvantageous from two points of view:—(a) The dust may become a nuisance and prove detrimental to the health of employees; (b) telluride of gold is brittle and has a tendency to be reduced to powder in the ball mills, and as such may be carried away with the dust. Special apparatus will, therefore, be provided to cope with this difficulty, and the dust will be drawn from the neighbourhood of the ball mills, by means of a fan, to a settling chamber.

The power for the whole plant will be supplied, partly by a 250 B.H.P. steam engine, and partly by a 340 B.H.P. gas engine, working on gas from a down-draught suction gas generator, burning firewood in five-foot lengths, direct."—J. D. CONNOR, *The Mining and Engineering Review*, Aug., 1914, Vol. 6, No. 71, p. 337. (J. E. T.)

CONTINUOUS DECANTATION AT THE PORCUPINE CROWN MINE.—"The Porcupine Crown Mine is situated on the T. & N. O. Ry., 250 miles from North Bay in the Porcupine mining district, Ontario, Canada and one-half mile from the town of Timmins. In February, 1913, a small amalgamating mill was built and put into commission primarily for the purpose of testing the mine sampling and also for producing revenue.

The small mill was operated until November 15th of the same year, during which time experiments made (both in the laboratory with cyanide tests and the mill on a larger working scale) indicated that the continuous decantation process would be the most economical. This was erected and running by November 15th without interfering in any way with the existing mill, which was made a unit of the new one, the amalgamating plates during the erection of the larger mill being placed in a temporary building which was later removed.

The factors determining the choice of the process were: (1) short time factor for solution of gold; (2) low solution strength required (1·25 lb.); (3) rapid settlement of pulp; (4) low costs of installation and operation; (5) small quantity of water required.

The ore is a hard quartz containing approximately 1% of iron pyrite. The grade of mill-run varies from $15 to $30 per ton. Some of the gold is coarse, but generally it is fine and freely disseminated in the quartz gangue, often associated with the pyrite. With the ore proper from 10 to 20% of country rock is sent to the mill.

Crushing and Elevating.—The ore is delivered in cars from the shaft house to the mill, where it is weighed and dumped upon a grizzly spaced 1½ in. The oversize falls into an ore-pocket of 60 tons capacity and feeds a No. 3 Chalmers & Williams bronze ball gyratory crusher set at 1½ in. The discharge from the crusher joins the undersize from the grizzly and feeds a vertical belt elevator carrying 120 buckets 9 by 9 in. travelling 200 ft. per minute; This delivers the ore to a short conveyor belt 16 in. wide travelling 200 ft. per minute, which distributes the ore in the bin equally to each battery of ten stamps. The ore bin has a capacity of 265 tons.

The ore is fed to twenty 1050 lb. gravity stamps by standard Challenge feeders. The stamps drop 102 times per minute. The height of drop is 7½ in. and height of discharge 4 in. A stamp duty is obtained of 8·8 tons for the 2½ mesh screens. The battery posts are 12 by 26 in. for the side posts and 20 by 26 in. for the king posts. The top girt for the 20 stamps is framed in one piece giving the 20 stamp structure considerable extra support. Fairlie belt tighteners are used. The mortars are standard narrow type for quick discharge and coarse crushing.

Two and one-half and 6-mesh screens are used on the batteries and varied to suit the mill requirements. The 2½ mesh has openings 0·25 in., and the 6 mesh has openings 0·111 in.

Classifier and Tube Mill Closed Circuit.—The stamps discharge to two Dorr classifiers, a duplex in circuit with a 5 by 16 ft. tube mill and a simplex in circuit with the 4 by 20 ft. tube mill. The 5 by 16 ft. makes 29 r.p.m. and the 4 by 20 ft. 31 r.p.m. The elevation of the classifiers is such that the discharge from the tube mills will flow by gravity back to the classifiers. In the case of the 5 by 16 ft. tube mill, this was accomplished by adding 4 ft. to the standard length of classifier, the oversize being fed to the scoop with a solution jet regulated to give 38% moisture. With the 4 by 20 ft. tube mill the desired result was obtained by increasing the radius of the scoop, the one used having a radius of 52 ins.

Twenty-four feet above the tube mill floor are placed two amalgamating plates, 9 by 6 ft., receiving the slime discharged from the tube mills, which is elevated by a triplex plunger pump and suitably diluted. The slope of the plates is 1½ in. per foot. The plunger pump has ball valves, the seats of which in time wear elliptical in shape, necessitating the use of a small amount of air to assist the pump, and later the valve seats must be taken out and reseated.

The plates have now been in use six months without material damage. The amalgam forms on the plates as a hard skin, and is removed with a scraper, care being taken not to get down to the copper. It is impossible to keep the plates soft and in good condition for catching fine gold, as practised in modern amalgamating mills, and this is not attempted. It does, however, pick up the heavy coarse gold, for which purpose only it is required. About 30% of the total recovery is made from these plates.

Cyaniding.—The cyanide plant essentially consists of five 30 by 12 ft. Dorr thickeners, working in series, with an agitator, 16 by 16 ft., between thickeners No. 1 and No. 2, together with the various pumping and precipitating apparatus. The pulp is delivered to a No. 1 thickener at a ratio of 5·25 tons solution to 1 ton solids. The overflow from this vat is the 'pregnant solution.' All of this solution is not necessarily precipitated. If the

overflow is of lower grade than required, a portion is allowed to by-pass to the pump returning the solution from No. 2 thickener to the battery storage, which is allowed to build up in value to a limited amount (see tabulation following).

The underflow or thickened pulp is directly connected by 4 in. piping to the suction end of two 4 in. diaphragm pumps. These are set 3½ ft. above the level of the solution in the vat. These pumps will handle pulp coataining as low as 25% moisture over short periods of time. Each pump has a sufficient capacity to handle the maximum output of the mill, thus leaving one pump in reserve. There are two speeds and three lengths of stroke to vary the volume pumped, giving ample variation to accommodate the mill tonnage.

The thick pulp (35% moisture) from No. 1 flows from the diaphram pump by gravity to the agitator. Here the moisture is raised by adding barren solution until the same has reached 60%. One-third of the cyanide is added at this point, the balance at the tube mill feed. The pulp is discharged from the agitator continuously to thickener No. 2. Then from this point to thickeners No. 3 and No. 4 the decantation is continuous, each vat having a 2 ft. elevation above the next preceding, giving the overflows a gravity flow from No. 5 to No. 2.

Cyanide Mill Averages for April, 1914 (120 tons per day).

	Thick 1.	Agitator.	Thick. 2.	Thick. 3.	Thick. 4.	Thick 5.	Battery.	Storage.	Solution pptd.
Dissolved gold ...	83.17	81.74	80.80	80.26	80.14	80.10	80.46	81.37	487 tons daily
Undissolved gold ..	1.82	1.06	0.82	0.60	0.53	0.36			Av. heads, 82.32
Ratio pulp to solution underflow	1 to 0.562	1 to 1.37	1 to 0.54	1 to 0.54	1 to 0.54	1 to 0.43	1 to 5.25		Av. tails, 80.03
Ratio pulp to solution feed...	1 to 5.250	1 to 1.40	1 to 5.25	1 to 4.41	1 to 4.41	1 to 0.97			
Percentage pulp to percentage solution underflow	64 to 36	42 to 58	65 to 35	65 to 35	65 to 35	69 to 31	16 to 84		

The barren solution from the precipitating presses is delivered to No. 4, a small portion going to the agitator. Water only is added to No. 5 equal in amount to the solution leaving with the thick pulp as tailing, thus maintaining a constant volume in the circuit. No. 5 is discharged through a spigot, with an average moisture content of 30 only. The diaphragm pumps in each case raise the thick pulp at 35% moisture from the various vats as already described for No 1. The solution overflows pass by gravity from No. 5 to No. 2, while the pulp flows in the opposite direction from No. 2 to No. 5. The value of the barren solution from the precipitating presses varies from 1c. to 3c. per ton solution. The average assay values of the pulp and solutions in the various vats during the months of March and April are shown in the tables.

There has been milled and cyanided 18,460 tons of ore with the plant described. The extraction has been between 96% and 97%. The loss in dissolved metal content in the tailing for March and April was as follows:—

	March.	April.
Value of solution per ton ...	80.120	80.100
Value of solution loss per ton of ore milled	0.052	0.043

The above result compares most favorably with any class of filter which might be used on a pulp such as that here treated. The pulp as discharged from No. 5 thickener, containing an average moisture content of only 30%, is an ideal condition for sampling, and while this has been done by hand up to the present time, an automatic sampler will be installed in the near future.

Miscellaneous Items.

	Per ton Mill.
Consumption of KCN ...	0.74
Consumption of Zn	0.98
Consumption of CaO	3.50

— M. SUMMERHAYES. — *Mining and Scientific Press*, July 18, 1914, p. 88. (G. H. S.)

PROCESS FOR SEPARATING PLATINUM FROM IRIDIUM AND OTHER METALS. — "The mixture of metals is treated with chlorine above 585° C. until the iridium is completely converted into chloride. A portion of the platinum is converted into chloride and volatilised, but most of it is unattacked and is separated from the non-volatile iridium trichloride by elutriation or by dissolving in *aqua regia*." — *Verein Chemischer Fabriken in Mannheim*, Ger. Pat., 273,178, March 28, 1913. — *Journal of the Society of Chemical Industry*, xxxiii., 13, p. 699, July 15, 1914. (J. A. W.)

SIMPLIFICATION OF GOLD ORE TREATMENT. — "Probably the most refractory 'versus' in controversial metallurgy is 'crushing in water and amalgamating *v.* crushing in solution without amalgamation.' The Dome and Hollinger mills are working side by side on ore differing essentially only in grade, the Dome crushing in water and amalgamating, while the Hollinger is crushing in solution without amalgamation. Both have been dropping 40 stamps on about the same tonnage, and both started during the summer of 1912. The only figures which are of any possible value for comparisons are those from the working of the mills during the latter part of 1913 and during the present year. Accurate figures from Hollinger operations for 1913 and 1914 to date are given in the following table, and also those approximating closely to the Dome present practice.

Treatment.	Hollinger Mill: (Exact figures from records.) 1913.	Hollinger Mill: (Exact figures from records.) 1914 to date.	Dome Mill: (Approximate present practice.)
Solution precipitated per ton of ore ..	3·16	2·67	1·70
Zinc consumption per ton of ore ...	0·66	0·59	0·34
Cyanide consumption per ton of ore ..	0·45	0·46	0·75
Cyanide added	...Tube mill feed		Before agitation
Total recovery ..	96·10	96·00	95·00

It is proved that in this instance crushing in solution without amalgamation is superior to crushing in water and amalgamating.

Tons of solution precipitated per ton of ore ('precipitation ratio') is a factor of the grade of ore. Now at the Hollinger 2·7 to 1 is being precipitated, against 1·7 to 1 at the Dome, while the grade of the Hollinger ore milled is about three times that of the Dome. Assuming that we were amalgamating at

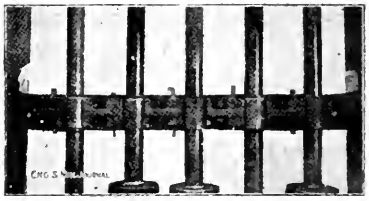
the Hollinger and that we could make the same extraction by amalgamation that the Dome makes, our cyanide 'heads' would still be three times those of the Dome, and a precipitation ratio of 5 to 1 (3 times 1·7) would be justifiable. In other words, we would be precipitating about twice what we do now. Again assuming that the Dome ore was crushed in solution and not amalgamated, the cyanide 'heads' would be one third of ours, so that the precipitation ratio should be one third that of the Hollinger or 0·9 to 1. Thus the Dome is precipitating twice what would be necessary were they crushing in solution and not amalgamating.

At first sight it would appear that if the gold going into solution can be reduced, the precipitation ratio can be reduced proportionately, but actually the reverse is the case. On account of crushing in water, an excess of solution is made and a corresponding amount must be precipitated and thrown away. This precipitation of 'waste' solution added to the high grade solution precipitation, makes the precipitation ratio greater than it would be were cyanide solution instead of water used in crushing—at the Dome twice as great. Discarding any solution from the mill other than with the tailing is poor practice if it can be avoided. One never knows whether a careless operator on night shift is not habitually opening the wrong valve and allowing good solution to get away; especially with zinc-dust a sudden loss of precipitation may run up the grade of the 'waste' solution; the cyanide loss is greatly increased and it may be necessary at any time to have to charge up a herd of sheep or a dozen cows to operation. The only point which can be made in favour of 'waste' solution is that fouling is reduced, and I must say that to precipitate and throw away solution is a crude method of avoiding this difficulty. There is always a loss of solution in the tailing, so that new solution is being made all the time sufficient to offset fouling with careful operation on normally clean ore. I say most emphatically that the slogan of a good mill superintendent should be 'Not a pound of water into the mill without getting a wash with it, and not a pound of solution out of the mill except with the tailing.' If a recovery of about 80% is possible by crushing and amalgamating in water, as at the Homestake, I admit that this principle does not apply; but with only 60% as at the Dome, it does apply, and I should say that with about 63%, which is the Rand recovery by amalgamation, I would even risk a charge as heresy in hinting that it is mighty close." NOEL CUNNINGHAM.—*Mining and Scientific Press*, July 4, 1914, p. 19. (G. H. S.)

NEW STAMP STEM GUIDE.—"The accompanying photograph illustrates the 'Little Wonder' battery stem guide, first introduced in the Yellow Aster mill, at Randsburg, Calif. This guide was designed by J. A. Little, mill superintendent for the Yellow Aster.

The guide frame, which may be either of special-grade iron or of cast steel, is a permanent fixture. There are only eight nuts used in the frame for the battery of five stem guides. These are used to hold the frame to the girt. The only wearing part in the entire structure is the bushing, which may be made of cast iron or cast steel, and can be renewed at small cost.

The bushings are made in single pieces or in halves, as desirable. They are 6 in. long and cylindrical in shape, and cannot stick in the frame, as is the case with tapered bushings. They are made with a shoulder, which prevents them from dropping through the frame. The guide frames are made with pockets in which gibs are placed. The bushing is placed



Stamp-Stem Guides at the Yellow Aster Mill.

with the shoulder resting on the frame and the key is driven lightly behind the gib. This holds the bushing with a tight grip, but causes no extra strain on the frame. The back of the keyway is a curved surface on the frame, and requires no machining.

The key is 1/4 in. distant from the stem, giving ample room for the use of a light hammer when necessary to remove the key. This arrangement prevents the stem being battered with the hammer, as is the case in the removal of many other stem guides. The rigidity of the guide frame insures perfect alignment of the stems.

These guides have been in use on two 5-stamp batteries in the Yellow Aster 100-stamp mill for 8 months. In that time there have been no alterations, repairs, or losses of time. As other stamp guides in use break or wear out, the 'Little Wonder' is substituted. They have been given a thorough comparative test with other designs and found to be really superior in performance and economy of cost." —J. A. LITTLE. *Engineering and Mining Journal*, July 25, 1914, p. 169. (G. H. S.)

NEW TUBE-MILL LINER. "It was evolved at Cobalt, Ontario, by Messrs. Globe and Mueller after continued trouble and excessive expense on account of the use of the parallel ribbed type of liner usually supplied by the manufacturers of tube-mills. These could not stand the severe abrasion to which the coarse-grinding mills of the district were subjected.

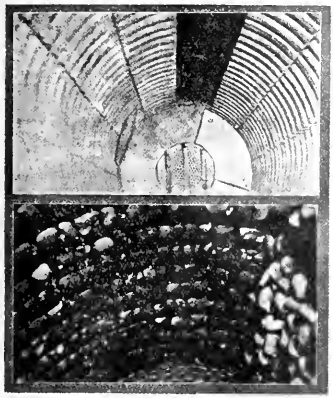
From the illustration the construction should be readily understood; the narrow chilled iron ribs are placed circumferential to the shell of the tube, thereby forming a series of tapered and dovetailed pebble pockets alternately disposed between the ribs on a soft cast iron base of the usual form. The pocket or channel is widest in the direction of rotation of the mill, and the ribs tapered inwardly toward the base, forming a tapered groove so disposed in relation to the direction of the rotation that the pebbles are automatically and quickly picked up and jammed securely into the grooves. This action takes place quickly and thoroughly.

It is absolutely essential that the initial load be a good mixture of all sizes of pebbles from marbles to number fours. The smallest pebbles find a secure hold in the crevices and voids of the larger ones, and the resulting surface, after two weeks, looks neat and is efficient; its roughness and irregularity tends to a better mixing and cascading action, and therefore better results in grinding.

So far there have been a dozen or more mills in Cobalt and Porcupine equipped with the liner, some coarse grinding, others on sliming, and in all cases they were installed after having been tested

against the parallel ribbed liner half and half in the same tube, or they may have been recommended by operators who where satisfied with the results, showing a longer life.

It should be remembered that in coarse-grinding machines the liners are deprived of that cushioning effect in sliming mills, on account of the more rapid water circulation, which causes a more rapid discharge of much coarse particles. In other words, the voids in the pebbles of a sliming mill are filled with a thicker and more sluggish mass than in a coarse-grinding mill, therefore the abrasive action on the liners and pebbles of such machines is not so severe.

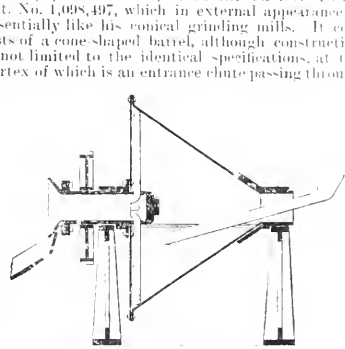


Globe and Marcy Tube-Mill Liners.

The cost of re-lining a 5 ft. × 20 ft. tube-mill, with a full set of 72 pieces, including 2 doors, is as follows:—15,550 lb. at $3.75 per 100 lb., $583; a set of 130 bolts for same at 1c. per lb., $19.59; felt washers at 1c. each, $1.50; one mechanic, 12 hours at $3.50, $5.25; two helpers inside the mill, 16 hours at $2.50, $10; two helpers outside the mill, 16 hours at $2.50, $10; one man putting on nuts and washers, 16 hours at $2.50, $5; one-half of the mill foreman's time, $2.25; making a total of $636. The actual time, including the removal of the old liners, with six men up to time of refilling with pebbles, was 12 hours, and no difficulties were encountered."—J. G. Mueller.—*Mining and Scientific Press*, Aug. 15, 1914, p. 255. (G. H. S.)

The Treatment of Complex Ores by the Ammonia CO_2 Process.—"The process for the treatment of the complex zinc ores is briefly as follows: Grind the ore, roast, regrind, leach out zinc with ammonia and carbon dioxide, precipitate from the solution any copper which has been dissolved with the zinc, and put back in the residue. Smelt the residue for the recovery of copper or lead, with the gold and silver. The ammonia and excess carbon dioxide are distilled from the solution containing the zinc and recovered for further use. As the ammonia is being distilled, the zinc is precipitated as a pure white basic zinc carbonate, filtered out, and then calcined to zinc oxide, or the basic zinc carbonate can be reduced direct to either zinc chloride or spelter."—S. E. Bretherton, *American Institute of Mining Engineers*, July, 1914, p. 1773. (J. A. W.)

Hardinge Classifier.—"To provide a new classifying machine for handling wet pulps, H. W. Hardinge, of New York, has invented a device, U.S. pat. No. 1,098,497, which in external appearance is essentially like his conical grinding mills. It consists of a cone-shaped barrel, although construction is not limited to the identical specifications, at the vortex of which is an entrance chute passing through



The Hardinge Classifier.

an overflow exit. The unclassified pulp enters through this chute, the light slime overflowing through the trunnion opening, while the sand settles in the angle between the cone sides and the head. There they are elevated by pipes and discharged into the other trunnion, the pipes acting like the spiral in a Frenier pump. The illustration accompanying shows the arrangement of the machine." *Engineering and Mining Journal*, Aug. 15, 1914, p. 308. (G. H. S.)

World's Production of Eight Useful Metals.—"According to a statistical summary and report compiled by the 'Metallbank und Metallurgische Gesellschaft,' Frankfort-on-Maine, the total production of these metals during the last three years has been:—

	1911.	1912.	1913.
	Metric Tons	Metric Tons	Metric Tons
Lead	1,136,000	1,181,800	1,186,700
Copper	893,800	1,018,600	1,005,900
Zinc	902,100	977,900	997,900
Tin	117,700	124,700	128,900
Aluminium ...	45,000	62,600	68,200
Nickel	24,500	28,500	30,000*
Mercury... ...	4,100	4,300	4,200
Silver, fine ...	7,685	7,715	

*Estimated.

—*Board of Trade Journal*, Aug. 6, 1914. *Journal of the Society of Chemical Industry*, Aug. 31, 1914, p. 833. (J. W.)

Standardization of Thermometers and Pyrometers.—"The question of standardization is of great importance to all users of thermometers, whether they are employed to measure high or low temperatures.

The boiling-points of:—

Aniline	...	184·1 C.
Naphthalene	...	218·0 C.
Benzophenone	...	306·0 C.

may all be used for standardization points. Aniline, however, oxidizes readily. Naphthalene is satisfactory, as it is cheap and readily obtained of sufficient purity. This is best tested by taking its freezing point, which should be 80·0 C. A special, but simple, boiling-point apparatus must be used with both naphthalene and benzophenone.

For higher temperatures, it is advisable to use the melting-points of pure metals or eutectic alloys and the freezing-points of pure salts.

The following will be found useful and satisfactory points:—

Freezing point of Tin	...	231·92 C.
,, ,, ,, Lead	...	327·43 C.
,, ,, ,, Zinc	...	419·37 C.*
Boiling ,, ,, ,, Sulphur	...	444·70 C.
Freezing ,, ,, ,, Antimony	...	630·7 C.
,, ,, ,, Sodium Chloride	...	800 C.
,, ,, ,, Silver	...	960·8S C.
,, ,, ,, Copper	...	1083·0 C.

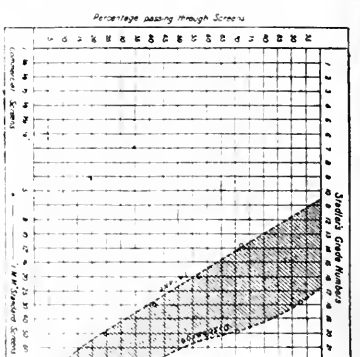
In the case of radiation and optical pyrometers, the best way is to sight upon a piece of firebrick or porcelain placed in either a small tube electric furnace or gas-fired muffle. A standardized thermo-electric couple is mounted either on the face of the firebrick or through a hole in its centre, the pyrometer being focussed on the couple. The furnace is then heated and the readings of the pyrometer compared at various temperatures. If a thermo-couple is not available, a single melting-point will be a valuable check on the pyrometer. A triangle is cut out of a piece of thin sheet silver, and one side partly folded over so that the sample will stand up when placed in the furnace. The pyrometer is then focussed on the tip of the triangle, and the temperature at which it melts (960·8S C.) observed.

When standardizing radiation and optical pyrometers, care must be taken to ensure that the conditions satisfy black body requirements."—R.G. WHIPPLE. *Proceedings Institute of Mechanical Engineers*, 1913, p. 759. (G. H. S.)

GRAPHIC RECORD OF SCREEN ANALYSIS. "In order to convey quickly to the mind the results of a screen analysis of crushed ore, Mr. S. J. Speak gives a method for graphically recording the results. Fig. 1 shows the graphic record of the figures given below:

Screen Mesh.	Aperture Inches.	Percentage in Tube Mill Feed	Percentage in Tube Mill Discharge
+ 12	0·03937	27·6	...
+ 26	0·02480	15·7	...
+ 30	0·01562	13·7	7·4
50	0·00984	19·9	22·5
80	0·00620	15·5	29·0
+ 120	0·00391	2·0	6·0
+ 200	0·00246	1·6	7·7
200	...	1·0	27·4

Referring to Fig. 1, the left-hand lower portion of the diagram represents the energy already expended on the ore, and the right-hand upper corner shows the work remaining to be done to reduce the ore to the finest grade. The shaded portion represents the useful work done by the tube mill, which can be measured more readily and satisfactorily than by



the mathematical method used by Stadler."—S. J. SPEAK, *Bulletin No. 116, Institution of Mining and Metallurgy; and Metallurgical and Chemical Engineering*, July, 1914, p. 482. (G. H. S.)

NITRIC ACID AND ITS USE FOR LIXIVIATING ORES.—Kingsley and Rankin treat chiefly galena and other sulphide ores with nitric acid. Kingsley obtains a lye consisting of nitrate of lead, whilst by Rankin's process lead sulphate is formed, the latter result being achieved by using concentrated HNO_3 and heating, stirring and employing a pressure of two atmospheres. The Norsk Hydro-Elektrisk Kvaelstofaktiehkab (a powerful Norwegian concern) uses a process by means of which ores containing bismuth as sulphide or oxide in small quantities are lixiviated with concentrated HNO_3, whilst the author recommends diluted HNO_3 when the treatment of oxides or carbonates of copper is required, ores which should, however, be rich in lime and free of sulphur. The recovery of the lixiviating acid is, of course, the chief point of the four processes, which are otherwise alike based upon the following formulæ and processes taking place:—

Kingsley Process.

Treatment of Ore.—

$$3PbS + 8HNO_3 = 3Pb(NO_3)_2 + 2NO + 4H_2O + 3S.$$

Recovery of Acid.—

$$Pb(NO_3)_2 + Na_2CO_3 = PbCO_3 + 2NaNO_3.$$
$$2NaNO_3 + H_2SO_4 = Na_2SO_4 + 2HNO_3.$$

Rankin Process.

Treatment of Ore.—

$$3PbS + 8HNO_3 = 3PbSO_4 + 8NO + 4H_2O.$$

Recovery of Acid.—

$$2NO + 3O + H_2O = 2HNO_3.$$

Process of the Norsk Hydro-Elektrisk Kvaelstofaktiehkab.

Treatment of Ore.—

$$Bi_2O_3 + 6HNO_3 = 2Bi(NO_3)_3 + 3H_2O.$$

Recovery of Acid.—

$$Bi(NO_3)_3 + H_2O = BiONO_3 + 2HNO_3.$$

Bueler de Florin Process.

The freezing point of zinc is a particularly useful point, as determinations made on a sample continuously for six years in one crucible gave strictly consistent results.

Treatment of Ore—

$$CuCO_3 + 2HNO_3 = Cu(NO_3)_2 + H_2O + CO_2.$$

Recovery of Acid—

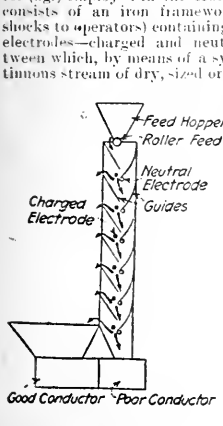
$$Cu(NO_3)_2 = CuO + N_2O_5.$$

$$N_2O_5 + H_2O = 2HNO_3.$$

With the aid of photographs, drawings, tables and practical calculations, the author gives a very instructive treatise of all technical and economical points, *i.e.*, supply of the lixiviating acid, action of acid upon different kinds of ores, rational recovery of acid, choice of building material for vats, pipes, concentration apparatus for the lye, calcining furnaces, acid condensing apparatus and absorption plants. As the working of the mine requires for the use of rock drills, ventilation, pumping, hoisting, milling, illuminating, etc., a central power plant it is advisable to design the latter large enough to allow for treating the ores by nitric acid which could be obtained from atmospheric nitrogen. It is, however, obvious that lixiviation by means of nitric acid is only economical in such places where the local conditions do not permit installing of smelters, wet concentration, erection of a sulphuric acid plant or cheap supply of the latter. Thus the HNO_3 lixiviation process does not represent an independent ore treatment, but lends itself only in particular cases where any other process fails to meet the requirements of decomposing and concentrating certain refractory ores."—B. DE FLORIN.—*Austral. Mining Standard*, Sept. 17, 1914, p. 261. (J. A. W.)

BRITTLENESS OF LEAD.—"Investigation has been carried out regarding brittleness which develops in lead under treatment. Lead plates were subjected to bending tests at varying temperatures, and it was found that brittleness increased from ordinary temperature up to 150° C. From 150° up to 260° a decrease in brittleness was observed. Metallographic examinations showed that the brittleness is connected with an increase in the size of the grain of the lead plates employed."—BAUER.—*The Engineering and Mining Review*, Sept. 5, 1914, Vol. VI., No. 72, p. 377. (J. E. T.)

ELECTROSTATIC ORE SEPARATION. "The separator (fig.) employed in the Huff electrostatic process consists of an iron framework (earth to prevent shocks to operators) containing two vertical series of electrodes—charged and neutral, respectively—between which, by means of a system of guides, a continuous stream of dry, sized ore is caused to fall from a hopper above, the best conditions as to rate of flow and spacing of electrodes being ascertained by preliminary tests. The charged electrodes are maintained at a potential of 20,000 volts and insulated from the iron framework, the neutral ones being in electrical contact with the latter. By the action of the electrostatic field upon the ore particles, the good conductors among the latter become immediately charged with electricity of the same polarity as that of the electrodes, and are repelled to the left side of the apparatus as indicated; while the poor conductors fall through the machine by gravity before any appreciable repulsion occurs. The two products thus obtained are re-treated in separate 'finishing' machines of the same kind, the 'middlings' produced being conducted to the dryer with the untreated ore. The electrical equipment consists of a generator for producing an alternating current of high potential but low amperage; a rectifier for changing the negative alternating-current waves into positive, pulsating waves; and a resistance box for regulating the speed of the generator. The process has been successfully employed for the separation of mica from graphite flake; slate from coal; and metalliferous minerals from various gangues." H. C. CLARK, *Eng. and Min. J.*, 1914, *98*, 264-267. *Journal of the Society of Chemical Industry*, Sept. 30, 1914, p. 924. (J. A. W.)



MINING.

DRIVING WESTON COLLIERY TUNNEL.—"Although presenting a problem not dissimilar to any number of other such tunnels, the 'entry tunnel' which was recently driven for the Locust Mountain Coal Co., of Shenandoah, Pa., and the methods by which the work was prosecuted, are of interest to coal operators and tunnel contractors by reason of the remarkably consistent and fast progress which prevailed.

The size of tunnel was 7 ft. 3 12 ft. with a drainage ditch 4 ft. 2 2 ft. at the right-hand side. It was driven through a hard conglomerate, with the exception of 450 ft. through red shale carrying a small quantity of water. The total length was 3,689 ft., 3 ft. less than the estimated distance.

Two Imperial air compressors belted to General Electric motors supplied the air for the drills. These machines compress 550 cub. ft. of air to 110 lb. at the receiver, giving 105 lb. at the drills; only one machine was in use at a time. An attempt was made to lower the pressure in order to reduce the strain on the drills, but this was abandoned owing to complaints from the drill runners who noticed the change immediately and claimed, that owing to the hardness of the rock, the drilling speed was reduced.

Two Leyner-Ingersoll drills were used on a 7 ft. horizontal bar and a third kept in reserve. One 'Butterfly' stoper and one Jackhamer were used for trimming and ditch work.

Water was supplied to the drills by a direct acting Cameron pump operated by air and shown in Fig. 1.

Fig. 1. *Weston Colliery Tunnel, looking towards Portal.*

This pumped the water from the drainage ditch into the vat shown on the right and air was turned on to keep it at an even pressure at the drills.

All steel was sharpened by an air-driven Leyner sharpener. The blacksmith shop equipment included a coke forge.

Ventilation was furnished by a pressure fan driven by a General Electric motor. This was placed at the entrance to the tunnel and the air forced through a 20 in. galvanised iron pipe. Air for the drills was carried through a 3 in. pipe ending in a manifold.

For handling the muck, six 1 yd. side-dump cars were used, drawn by an electric storage-battery locomotive. Track of 24 in. gauge was used and a false track kept well up to the face.

Three shifts per day were employed. Each crew averaged 6 hours at the face, from the time the crew entered until the round was completely drilled.

Each shift consisted of 1 shift boss, 4 machine men, 6 muckers, 1 motor man, 1 brakeman, 1 ditch and floor walker, and 1 man to pump the muck.

The three-shift method of driving was found feasible by employing high-speed drills of the self-rotating hammer type, and by the adoption of a comparatively shallow cut, approximately 5½ ft. per shot, although the drill holes averaged 6 ft. deep. This is in line with the most modern practice in tunnel driving, it being found that a slight decrease in depth of cut with a larger number of comparatively small holes, carefully placed, permits employing more shifts in a given time and a considerable increase in speed of driving. Though no attempt was made at record breaking, a remarkably uniform yet fast speed was maintained over the whole period of driving. The average rate accomplished was 15·66 ft. per day.

Considering the extreme hardness of the rock, this compares very favourably with such tunnels as the Laramie Poudre, the Montreal, and the Loetschberg (the two last named were driven through soft limestone).

The best month's progress was during October, 1913, when 513 ft. were driven in 28 working days.

The drilling round varied considerably, being changed according to the nature of the ground. In general, the system followed was that known as the 'Leyner cut,' shown in Fig. 2, in which 18 to 24

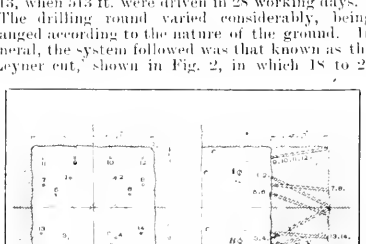


Fig. 2.—*The Leyner Cut.*

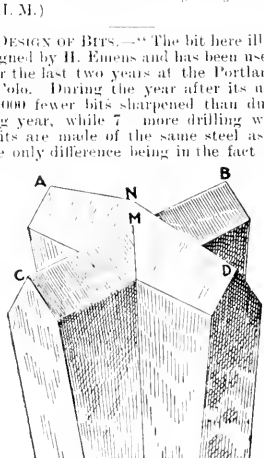
holes were drilled. The average time for drilling the round was 5 hours, Hercules blasting gelatine of which 77,975 lb. of 60 % and 3,150 lb. of 50 %, with a small quantity of 40 % and 100 %, was used. The shots were fired by fuse, cut to such lengths that the holes would fire in the order desired, the cut holes being the first to break. A total of 164,250 ft. of Crescent fuse, and 20,023 No. 8 caps was used.

In general, the muckers were able to enter the tunnel 15 minutes after shooting; this was made possible by the efficient system of ventilation assisted by turning on full air pressure from the compressors immediately after shooting. In from 15 to 30 minutes the muck pile was sufficiently reduced to allow operating the drills from the top set-up. Here the advantages of the tunnel bar over the usual column is clearly shown. All holes but the three lifters were drilled from the upper set-up of the bar and as a rule the heading was completely mucked out by the time the drilling crew were ready to put in the lifters. The bar was then changed to the lower set-up and the three lift holes drilled, this operation taking between 15 and 30 minutes. The right-hand lift hole was drilled deep and low to form the ditch. The commendatory feature about the system was the prosecution of the drilling and mucking operatings with the minimum loss of time for both drill runners and muckers.

The drills used embodied in their design a water feed attachment, which entirely eliminated the dust, the heading being remarkably clear except for a slight fog from the drill exhaust.

The tunnel was finished with ditch taken up and road to line and grade as the work progressed."—STEPHEN W. SYMONS.—*Colliery Engineer*, Aug., 1914. (H. M.)

NEW DESIGN OF BITS.—"The bit here illustrated was designed by H. Emens and has been used exclusively for the last two years at the Portland mine, Victor, Colo. During the year after its use, there were 34,000 fewer bits sharpened than during the preceding year, while 7 more drilling was done. These bits are made of the same steel as the old ones, the only difference being in the fact that one



Bit with One Edge High Centre and One Edge Straight.

set of wings is raised in the centre. The raised part acts as a guide for the rest of the bit and causes the cuttings to come out coarser. The wear on the bits comes at the edges, the centres not being worn appreciably.

The points *A*, *B*, *C*, *D*, are in the same plane. On 1½ in. bit *N* is ⅞ in. above *M*."—F. W. FOOTE.—*Engineering and Mining Journal*, Sept. 12, 1914, p. 484. (G. H. S.)

Results of Tests of Compressive Strength of Various Forms of Roof Support.*

No. of Test.	Construction Tested	Net Tons per Square Foot Required to Produce Compression of—						Compression and Load at End of Test.		Remarks.
		1 per cent.	3 per cent.	5 per cent.	10 per cent.	20 per cent.	30 per cent.	Compression.	Load.	
								Per cent.	Tons per sq. ft.	
1	Rectangular piers of mine rock ...	...	0·8	1·4	2·7	9·5	23	...	...	Average construction; voids not filled.
2	Circular pier of mine rock	...	3·5	5·67	11	22	38·5	31	42·5	Well constructed; voids filled with small and shovel stuff.
3	Timber crib filled with mine rock ..	..	0·6	1·57	5·11	20·3	31·5	..	..	Average construction.
4	Pile of broken sandstone; small pieces...	..	...	...	4	9·3	22·4	46	101·5	
5	Pile of small-size broken sandstone and sand	...	...	1·6	4	13·5	35	45	144	In these tests the materials was not confined, but was free to expand laterally.
6	Pile of broken sandstone; large pieces...	...	3·6	5	9·1	26·4	37	41	75	
7	Pile of Coal-Measures sandstone similar to No. 6	0·8	2·1	3·5	9	33·4	..	27	56	
8	Pile of river sand	...	...	...	...	...	...	63	108	
9	Broken sandstone in cylinder	..	3·33	5·55	13·32	46·6	98·6	35	666	
10	Broken sandstone and sand in cylinder	..	3·5	5·77	24·42	308·5	....	23	686	
11	{ Dry coal ashes in cylinder ... ... { Coal ashes flushed in with water ..	...	1	1·86	5·32	10·8	25	51	666	
		.		...	.	5·50	22			
12	Wet culm flushed in cylinder; partly dried (average of two tests) ...	2·44	8·9	14·28	35·52	138·7	444	33	668	In these tests the material was confined and could not expand laterally.
13	Dry sand in cylinder	0·88	3	5·27	33·3	129	499	32·2	666	
14	Wet sand flushed in and partly dried ...	8·4	39·3	67	173·8	555·4	...	20·75	666	
15	Concrete made of cement, sand and gravel; four months old; one barrel Portland cement to each cubic yard of concrete (about one part cement to seven parts sand and gravel); piers 3 in. × 2·81 in. × 3·85 in. high.	9 Cracked.	84	Gradually crushed to pieces under continuous load of 45 tons.						This test was made at the Dickson Works of the Allis Chalmers Co., in Scranton, by William Griffith.

Extract from Bulletin 60, on "Hydraulic Mine Filling, p. 22, U.S.A. Dept. of Interior, Bureau of Mines, 1913. (W. A. C.)

* See this *Journal*, Vol. XIV., Sept., 1913, p. 120.

ROPE IDLERS IN THE RAVEN SHAFT. "The shaft of the Raven mine, at Butte, Mont., is an incline 1,700 ft. in length and dipping at various angles. At the top the dip is 70° from the horizontal, but this is gradually flattened until at the 300 ft. level the inclination is only 47°. This angle continues to the 1,100 ft. level, below which it curves with a 125 ft. radius to 78°. In addition to these variations in dip, the shaft does not lie in the same vertical plane, with the result that the hoisting rope not only rubs at intervals on both the hanging and foot walls, but presses strongly against the west dividers near the collar, while 300 ft. below it runs close to the east end plates.

The early operators used no idlers, and wall plates cut 6 in. deep by the rope resulted. Later operators first attempted to overcome the excessive friction, and the wear of rope and wall plates, by introducing solid cast iron idlers, 3 in. in diameter. To allow for the travel of the rope from side to side some of these had to be 3 ft. long and were very heavy. Judging from the appearance of the old idlers of this type found at the mine, they often failed to turn in the bearings, which is not surprising when it is considered that they would have to make 1,000 rev. a minute under ordinary hoisting conditions.

two half circles together formed a bearing. Oil holes were provided, and in some cases holes were bored through the two pieces so that they could be screwed or spiked to the wall plates. The later practice was to forge the bearings from 1 in. × 3 in. steel, and to drill two holes at each end for ⅜ in. lag screws, by which the bearings were fastened to the timbers. These bearings were finally used on both foot and hanging wall. Similar idlers were so placed as to protect the dividers and end plates. The difficulty of proper oiling presented the greatest obstacle to satisfactory results from this type of idler. As the clearance between the skip and the hanging wall plates was sometimes less than an inch, there was not room for large oil or grease cups. In addition, the bearings were liable to get full of grit, especially when wet ore was being hoisted. Grease cups were generally unsatisfactory, although several kinds of grease were tried and especial attention was paid to having that which was suited to the temperature of the shaft. In any event it was necessary that the rolls be examined and the oil cups filled every two days, which meant the cessation of hoisting for two hours. The bearings wore rapidly and the rollers tended to get out of line. The full skip weighed over 3 tons, and where the shaft flattened

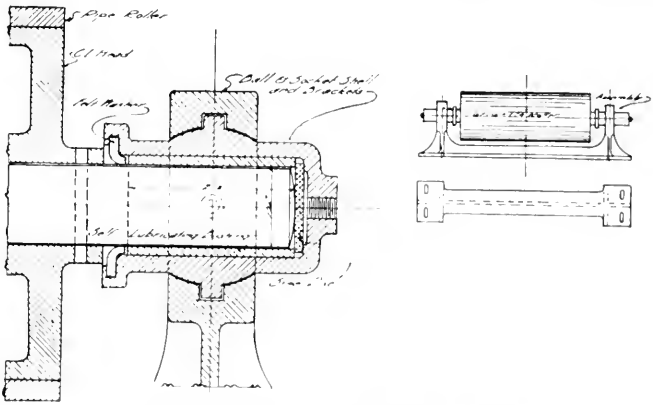


FIG. 1. *Rope Idler used in the Raven Mine, Butte, Mont.*

The next rolls were made of wood, 6 in. in diameter, with an iron band about each end, and a pintle of 1 in. round steel driven in at the ends to serve as a shaft. These wore rapidly, and were soon replaced by rolls made from water pipe, 5 in. or 6 in. in diameter, cut to the desired length and fitted with a wooden cylinder into which the pintle was driven. Where the idlers were used on the hanging wall of the shaft the original bearing was simply a piece of ½ in. × 4½ in. strap iron, 10 in. long, turned up at the end in a circle of 4½ in. in diameter to receive the shaft. A small hole served for oiling, and common black oil was generally used, although filtered oil from engine bearing and compressor bearings was also used. When the rolls were to be placed on the foot wall the bearings were made from two pieces of 1 in. × 3 in. steel, 6 in. long. A half circle was cut on the flat side of each piece and the

near the surface the pressure against the idlers was heavy. It was only by distributing this weight over idlers placed but 5 ft. apart that anything approaching satisfactory service could be obtained at this point.

To obviate the necessity for so much attention, an idler was devised, following my suggestions, by the Robins Conveying Belt Co. This idler is shown in Fig. 1. The roller is extra heavy 6 in. pipe, ⅜ in. thick, 20 in. long, in each end of which is pressed a cast-iron head, and through which passes a 1¼ in. steel shaft. This turns in a self lubricating bearing carried by a bracket in a ball-and-socket shell, which prevents cramping. As a preliminary to adopting these bearings two types of graphite and bronze self-lubricating bushings were tried side by side in the incline for three months. The one proving most satisfactory had cylindrical bodies of graphite ¾ in.

in diameter set in the bronze, or 'metalline,' bushing at about ¼ in. centres. One end of the bearing is entirely closed, the end thrust being taken by a steel disk, which also serves for forcing out the bushing when it is worn. The other end of the bearing is protected from grit by a felt washer. This, however, also retains the fine particles of metal and graphite, and in time this gummy matter causes the bearings to bind. Occasional cleaning of the bushings with kerosene obviates this trouble. The cap is hinged at one side and fastened at the other with a hinged bolt so the roller and bushing can be easily removed. The bearings can be turned through 90°, and the roll turned end for end, permitting the advantage of full wear. The whole is carried in a casting which is fastened to the wall plates with ¼ in. lag screws. These idlers have been in use nearly a year and are very satisfactory. While with rapid and continuous hoisting the bearings become quite hot, they do not bind if they are cleaned occasionally.

For the lower part of the shaft, where the rope runs true and the inclination is 78°, the idlers are merely common sheave wheels, cast solid and keyed on a shaft of 1 in. cold-rolled steel. This has a total length of 13 in. These sheaves are 9 in. in diameter with a 3 in. face, having a groove 1½ in. deep, 1 in. wide at the bottom. The bearings are maple blocks 4 in. × 4 in. × 6 in., bored to receive the shaft, and provided with an oil hole. These are fastened to the wall plates with six spikes.

The wood-filled pipe idlers with forged bearings cost at Butte about $8 each, including bearings. The idler with the self-lubricating bearing costs $15, but the difference is quickly saved in decreased cost of attention. The solid cast sheaves weigh 26 lb. They cost, when fitted with a shaft, but excluding the maple bearings, $2.75 each."—Geo. A. Packard. —Trans. American Institute of Mining Engineers. (J. W.)

Platinum Ore in Southern Nevada.—"The presence of platinum and metals of the platinum group in ores from south-eastern Nevada was noted by members of the U.S. Geological Survey as early as 1909, the platinum metals occurring with copper, nickel and cobalt in ores from the Key West and Great Eastern mines, near Bunkerville, in Clark County. In these properties, the ore bodies are associated with pegmatites and very basic intrusions, carrying pyrrhotite and chalcopyrite, the content of platinum metals in the ore varying from 0·1 oz. to 0·2 oz. per ton, so far as data are obtainable.

A deposit of ore, containing the platinum metals in greater quantity and entirely dissimilar in occurrence, has recently been discovered in the western portion of Clark County at the property of the Boss Gold Mining Co. This deposit is 10 miles west of Goodsprings, in the Yellow Pine mining district.

The property of the Boss Gold Mining Co., formerly known as the Boss mine, was originally explored in the early '90s on account of the copper content of the ore, there having been a large outcrop of copper-carbonate ore at the surface. A leaching plant was constructed at Goodsprings to treat the ore from this and the Columbia mine. The leaching plant proved a failure. Not recognising the more valuable constituents of the ore, the property was permitted to remain idle until the spring of 1914, when the Boss Gold Mining Co. was organised for its further development.

The country rock in the vicinity of the Boss property consists of thick-bedded limestones, probably of upper Mississippian age, dipping about S 60° W at an angle of 10° to 20° from the horizontal. These limestones are cut by a large batholithic intrusion of acid porphyry, which has been identified as a quartz monzonite, showing large phenocrysts of orthoclase feldspar and occasional quartz crystals in a fine-grained ground mass, the whole presenting a facies almost identical with the numerous porphyry intrusions throughout the Yellow Pine district.

The ore bodies so far developed occur entirely in limestone, in a crushed fault zone, striking N 30° E, which traverses both limestone and porphyry. A tunnel has been driven into this fault zone from the point where the largest outcrop occurred; at lower levels, two other tunnels have been driven, the first of which, about 30 ft. lower than the upper tunnel, has encountered the main ore zone. The lowest tunnel is being driven at an elevation several hundred feet lower than the upper workings, and has not yet reached the main ore zone.

Ore Richer in Platinum than First Discoveries.—The main or upper tunnel followed the ore for a distance of 200 ft. along its strike, in a north-easterly direction. The ore varies in thickness from 2 ft. to 10 ft., the average being about 6 ft., and in appearance is a brownish granular mass, showing occasional streaks of hard quartzose material. Both walls of the ore are shattered limestone, and along these walls occurs an encasement of malachite, varying in thickness from a few inches to several feet. The main body of the ore zone carries little copper. An average of the ore, sampled at 5 ft. intervals for 200 ft. along the upper tunnel gave the following results: Gold, 1·13 oz. per ton; silver, 5·20 oz.; copper, 0·20%. These samples were not analysed for metals of the platinum group, the chemist for the Yellow Pine Mining Co., who analysed the samples, reported that metals of the platinum group were present, this being the first report of the presence of these metals. It has since been learned that this ore contains 0·5 oz. to 1 oz of platinum per ton.

In June, 1914, a winze was started from the upper tunnel following the ore on its dip, and high-grade ore was encountered within a few feet. The appearance of the ore did not change materially except for the occurrence of small masses of a greenish talc within the vein. Some of these masses were sufficiently large to be mined separately, and two small shipments, aggregating about 1 ton, were forwarded to the American Smelting and Refining Co., at Murray, Utah. Their settlement analysis was as follows, platinum not being determined or paid for: Gold, 124·79 oz. per ton; silver, 23·9 oz.; lead, 1·05%; copper, 0·65%; insoluble, 73·2%; sulphur, 3·1%; iron, 6·7%.

On the same control sample of this shipment, New York assayers report the following analysis for metals of the platinum group: Platinum, 99·08 oz. per ton; palladium, 16 oz.; iridium, trace.

The winze has been sunk to a depth of about 35 ft., and drifts driven on the ore in both directions, developing a considerable tonnage of ore. A carload shipment of material from this development, without sorting, returned as follows: Gold, 8·752 oz. per ton; silver, 5·02 oz.; copper, 0·97%; insoluble, 80·7%.

Similar material from this location assays: Gold 11·55 oz. per ton; platinum, 7·38 oz.

Ratio of Platinum to Gold.—It has been fully determined that metals of the platinum group are present in all of the ore thus far developed. Analysts state as their opinion that the metals are in the free state, 'being apparently alloys of gold and platinum metals.' It would also appear from numerous assays

that the platinum metals bear a fixed ratio to the gold content, being in the proportion of about ¾ oz. platinum to 1 oz. gold. The ore is thoroughly oxidised, no sulphides having as yet been recognised, and the gold alloys have a 'rusty' appearance, showing no colours after panning until they have been thoroughly scrubbed or treated with acid.

The occurrence of the ore is especially peculiar in that it occurs entirely in the limestone, the nearest known porphyry contact being about 600 ft. distant. The ore apparently follows the stratification of the limestone more or less regularly, indicating replacement, but appears to be confined in a fault zone about 60 ft. in width, following the fault planes along their strike. A gouge occurs along some of these fault planes, which is strongly indicative of porphyry, and metamorphism is noticeable in the adjacent limestone, indicating that an igneous intrusion may be closer to the ore body than is apparent from the present development. It is probable that the porphyry was the original source of the ore, the precious metals having been deposited in the crushed fault zone, with copper and iron sulphides, which since have been thoroughly oxidised. It is a noticeable fact that the malachite above described carries little gold or silver and no platinum metals are evident. A carload shipment of copper ore from the same workings contained 23·4 copper, but only 1·4 oz. silver and 0·16 oz. gold per ton.

From present development it would appear that the district bids fair to become an important producer of platinum." F. A. Hall. *Engineering and Mining Journal*, Oct. 10, 1914, p. 641. (G. H. S.)

Calumet and Hecla Ventilation and Temperatures. —"The temperature in the deeper workings of the conglomerates of the Calumet and Hecla and of the Tamarack is higher than in any of the other mines. The temperature varies with the depth and with the season, but the temperature of the amygdaloid mines is universally low. The average temperature of the air in the bottom part of the Calumet and Hecla conglomerate is about 80°. The highest temperature ever found in that formation was in the rock and was 87°. Rock temperature was obtained by drilling a long hole in the wall and putting in a low registering thermometer, sealing the hole and leaving the instrument for a period of weeks or months. The temperature of the amygdaloids will vary from 55° to 70°, being higher in the lower levels of the Tamarack than at any other mine in the district. A long series of tests of air temperature at the Tamarack gave a maximum of 89° in October, 1909, on the 39th level. The average air temperature in the bottom level is from 82° to 84°.

The mines all have natural ventilation, which is so effective as to require control in winter, doors being placed over the shaft to retard the descending current of cold air. There is no artificial ventilation other than the compressed air used in drilling. Mr. MacNaughton gave as his opinion that no advantage could be had by artificial ventilation, since the only real difficulty encountered was that of gas clinging in moist muck after blasting, for which tan ventilation would of course be no remedy." *Engineering and Mining Journal*, July 25, 1914, p. 176. (G. H. S.)

Sectional Concrete Shaft Lining. —"A new and ingenious method of shaft lining is described in detail by Marcel Gillieaux, in the 1914 number of *Annales des Mines de Belgique*.

It is applied to circular shafts and consists of panels of reinforced concrete keyed together both horizontally and vertically and capable of being suspended one from another by hooks or goosenecks cast at top and bottom of the blocks. The shape and proportions of the blocks are evident from the illustrations. Fig. 1 shows a lining in the shaft, sup-

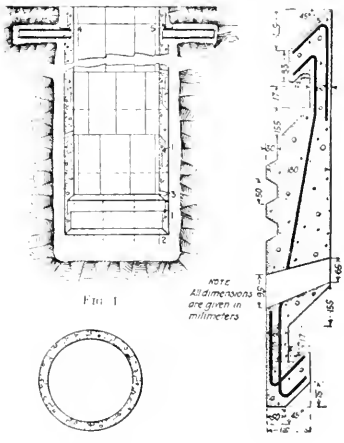


Fig. 1 Fig. 2 Fig. 3

A Suspended Reinforced Concrete Shaft Lining.

ported by beaters in concrete. The plan, Fig. 2, illustrates the vertical jointing. The dimensions of Fig. 3 are in millimeters. The opening at 3, Fig. 1, serves the double purpose of allowing the blocks and lining to be centred by iron bars to the shaft wall and of affording an opening for running in cement to fill between the lining and the rock.

This appears to be the only concrete lining for a circular shaft which can be carried on down as sinking progresses." *Engineering and Mining Journal*, Aug. 1, 1914, p. 240. (G. H. S.)

Changes of Address.

Alston, R. A., *to* Ravenswood, Kilmacolm, Scotland.

Crewpton, H., *to* Agincourt Mine, Belingwe, Rhodesia.

Gilchrist, D., *l/o* Scotland; Brakpan Mines, Ltd., P. O. Box 3, Brakpan.

Hingle, L. D., *l/o* America; Crown Mines, Ltd., P. O. Box 102, Crown Mines.

Jennings, S. J., *to* 42, Broadway, New York City, U.S.A.

Jolly, T. S., *to* Robinson G. M. Co., P. O. Box 1024, Johannesburg.

McLennan, J., *l/o* Rhodesia; c/o Union Club of South Africa, Johannesburg.

Willows Munro, S. E., *to* New Heriot G. M. Co., P. O. Box 10, Cleveland.

Woodburn, J. A., *l/o* Messina; P. O. Box 5258, Johannesburg.

APR 15 1920
UNIVERSITY OF TORONTO

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OF SOUTH AFRICA.

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Vol. XV. FEBRUARY 1915. No. 7.

Proceedings
AT
Ordinary General Meeting, February 20th, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, 20th February, Prof. G. H. Stanley (President) in the chair. There were also present:—

23 Members: Messrs. E. Pam, H. S. Meyer, C. Toombs, F. W. Watson, John Watson, H. A. White, W. R. Dowling, E. H. Johnson (Members of Council), R. A. Cooper, R. V. Crake, T. N. Dewar, J. M. Dixon, R. Dures, B. J. Hastings, E. J. Laschinger, G. Melvill, P. T. Morrisby, S. Newton, W. S. V. Price, C. E. Simpson, A. Thomas, G. A. Watermeyer, and E. M. Weston.

9 Associates and Students: Messrs. W. J. R. Hunter, J. Ingram, R. W. Irwin, A. King, G. Lundholm, G. F. Nisbeth, H. Olsson, F. J. Pooler, and W. Waters.

9 Visitors and Fred Rowland (Secretary).

MINUTES.

The minutes of the last meeting were confirmed.

NEW MEMBERS.

Messrs. R. Dures and P. T. Morrisby were appointed scrutineers, and the following members were declared unanimously elected:

HADFIELD, SIR ROBERT ABBOTT, F.R.S., 22, Carlton House Terrace, London, W.

WEINGREN, MAURICE, corner Staib and Beit Streets, Doornfontein, Johannesburg. Mining Engineer.

The Secretary notified the admittance of the following Associate:—

MILTON, CECIL HENRY, Planet-Arcturus Mine, P. O. Arcturus, Rhodesia. Mining Engineer.

The President: I should like to call attention to the fact that Sir Robert Hadfield is one of the foremost scientific men in Great Britain, and it is a distinct gain to the Society to have him amongst our members.

The next announcement I have to make is that His Excellency Lord Buxton has honoured us by taking office as Honorary President, and that Mr. Drummond Chaplin, who you know has left the Rand to take up the appointment of Administrator of Southern Rhodesia, has become an Honorary Vice-President.

I have also to congratulate our Vice-President, Mr. Pam, on his election as President of the Association of Mine Managers. I am sure you will all join with me in congratulating him.

I next have a sad duty to perform. You all know that Professor Christy, of California, has been one of our honorary members for a number of years past. He was well known for his scientific work in this district, more particularly because of his labours in connection with the chemistry of the cyanide process. He passed away on November 30th, at the comparatively early age of 61. I should like to propose that a vote of condolence with his widow be passed.

This was agreed to: all present standing.

GENERAL BUSINESS.

VISIT TO MODDER B.

The Secretary: I have to announce that by arrangement with the South African Institution of Engineers a visit to the Modder B. Mine has been arranged, which will take place on Saturday, March 6th. Members will be notified immediately the arrangements are fully completed.

SOUTH AFRICAN ENGINEER CORPS.

The President: There is just one little matter of interest to which my attention has been drawn during the meeting. There is an advertisement in the local Press with regard to recruits for the South African Engineer

Corps, giving a list of trades for which men are required. The list starts with analysts That shows a somewhat curious confusion of ideas on the part of the Defence Force authorities. Without wishing to say anything derogatory to the people who come under the following heads, I do think analysts always regard themselves as being members of a definite profession which requires different training to that for an ordinary trade. The remaining heads are blacksmiths, bricklayers, carpenters, joiners, and so on, in fact, the ordinary trade-men. However, as analysts are needed, we gladly draw attention to the fact, with the hope that the Defence Force authorities will reconsider the manner in which the requirements of the service are brought to the notice of those concerned.

SCALE OF SULPHATE OF LIME FROM WATER LAUNDER IN MAIN INCLINE SHAFT.

Mr. John Watson *(Member of Council):* The specimen of scale, herewith, may be of interest to some of our members. It is a little over ½ in. thick and was deposited in the City and Suburban Mine in the water launder in the main incline shaft. The water flowed from 18th to 30th levels, and had been treated by lime. The sample was handed to me by the General Manager (Mr. Chas. Glyn) and I found its composition to be as follows:

	Scale.	Cryst. Calcium Sulphate (Gypsum)
Lime (CaO)	28·27	32·56
Silica, etc.	5·76	
Alumina and Ferric Oxide	0·58	
Magnesia (MgO)	0·37	
Sulphur trioxide (SO_3)	13·32	16·51
Carbon dioxide	0·75	
Moisture (at dull red)	21·26	20·93
	100·31	100·00

It contained only a trace of gold.

It consists essentially of gypsum, plus a little silica, alumina, carbon dioxide, etc., which one might expect in any mine shaft, and was formed during a period of about three years.

NOTES ON TREATMENT OF ZINC-GOLD SLIME.

Mr. J. F. Pyles *(Member):* The separate treatment of white precipitate on the mine with which I am connected differs slightly from that recommended by Mr. H. V. White.* With us the boxes containing white precipitate are cleaned up two or three days prior to the ordinary clean-up. All white precipitate from the boxes and press is put into a vat used for this product only, where it is agitated for 24 hours in an excess of acid. It is then allowed to settle, when the supernatant liquor is drawn off through the press.

During the ordinary clean-up, when zinc is being dissolved and the action is most violent, the white precipitate product is dropped from its vat into a small sump, passing through a 61 mesh screen. It is lifted from there into the ordinary clean-up vat by means of an ejector, where it falls on to the dissolving zinc below. We have found that the greatest reduction in bulk takes place at this time. Whether there are four presses full of white precipitate besides that taken from the boxes in buckets, or whether there is only one, the final bulk of gold slime to be smelted varies little from half a press.

* This *Journal*, Vol. xv., Sept., 1914, pp. 50-51.

PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M.

This paper is submitted for discussion, both to this Society and to the Institution of Mining and Metallurgy, according to the arrangement existing between the two bodies.

The Secretary read the introduction to this paper,* submitted by Mr. Rickard to the Institution of mining and Metallurgy, which is reproduced here in a condensed form.

Mr. T. A. Rickard, in introducing his paper, said: By courtesy of Mr. C. F. W. Kup, the secretary of the St. John del Rey Company, I am able to inform you that that mine is now 5,387 ft. below adit and 5,711 ft. in total vertical depth, so that the St. John del Rey is the deepest metal mine in the world.

I also have a little explanation to make. I have reason to believe that a phrase in my paper has been misunderstood, as conveying a reflection on the good faith of one of those whom I controverted. I refer to Mr. Marriott. One page 11 you will see that I have said that "it is fair to assume that he had not the facts to support his assertion." That, of course, was an expres-

* Printed in the December–January *Journal*.

sion of my opinion. Mr. Marriott has evidence of an interesting character, and he has been kind enough to let me see it, but while he thinks it supports his assertion, I cannot agree that it does so. For the purpose of this discussion, it does not exist until it is published.

The enrichment of ore in depth has been consigned to the limbo of discarded fallacies; the generalization that ore persists indefinitely in depth must join it. Geologists may prove to their satisfaction that the deepest mine workings are relatively shallow and that the question of depth in itself is rendered supererogatory by the miles of erosion to which the ore-bearing rocks have been subjected. But such academic arguments are only confusing.

To assert the indefinite persistence of ore is to assume the inexhaustibility of ore deposits. Is history to be disregarded? Greece obtained her silver from Laurium as long ago as the Peloponnesian war; Hannibal drew money for his campaigns from the gold mines of Iberia; the Roman emperors took tribute from the gold mines of Dacia; the ancient world derived its copper from the Sinai peninsula; and so forth. Of these mines, in most cases, only the memory remains.

Is that an unpractical test? Then I turn to the share-list of mines quoted on the London Stock Exchange at the date when my first article on this subject was published, namely, January 21, 1893, nearly 22 years ago. In the *Financial Times* of that day I find a list of the mines that were then the subject of joint-stock speculation. Out of 250 companies then quoted, only a quarter have survived. Out of 22 British mines, only 7 have been continuously operated, and each one of these has gone through phases of reconstruction and recapitalization; 15 are dead as mutton. Out of the 76 South African mines, 30 have succumbed, chiefly diamond ventures and gold enterprises on the outskirts of the Rand. As the Rand is practically one lode, the discontinuance of an individual mine is inconclusive. On the Miscellaneous list there were 160, of which number only 23 are now doing business, and of those 5 have suffered reconstruction. Among the casualties are half-a-dozen Mysore 'pups,' for in 1893 Indian mining was enjoying a boom. The list also includes a number of mines once famous. Of these mines, the majority yielded fortunes to their happy owners, and the minority that entailed loss on their shareholders did so chiefly through such unjustifiable exaggeration of their productivity as was due to an erroneous assumption of inexhaustibility based on the idea of indefinite persistence of ore in depth.

It will be said by some worthy people that the insistence upon unpleasant realities is hurtful to mining speculation and of no particular advantage to the industry. Leaving the proud platform of the scientific man, scorning all intellectual dishonesty, let me reply on the sandy floor of workaday commercialism, by insisting that the capital available for mining is limited, and that the more of it that is squandered over wild-goose chases or will-o'-the-wisps, the less of it will there be for intelligent enterprise. When money is furnished for foolish ventures, the industry suffers by disgusting or disappointing those who would otherwise participate in reasonable projects.

It is high time that an unsubstantiated generalization should cease to provide material for flamboyant prospectuses and irresponsible promoters. It is high time that mining engineers should dissociate themselves from pseudo-scientific fallacies masquerading as scientific truths. That is why I have taken the pains to collect data and to marshal arguments in the hope of disproving once and for all that the indefinite persistence of ore in depth is neither a fact nor a theory in economic geology.

The President: This paper, following our new procedure, has been printed and circulated in the *Journal*, and discussion is invited at this meeting. I think you will agree that the author's introduction, although shorter, is as interesting as the paper. The paper is certainly extremely well written, and the arguments are marshalled very carefully, and to most people will be convincing.

I do not feel competent to express an opinion one way or the other, but whether we agree with the writer or not we must all agree it is an excellent paper, and pass a hearty vote of thanks to him for it.

Mr. E. M. Weston (*Member*): I have much pleasure in seconding this vote of thanks. We all know that Mr. Rickard is a mining man who has had the widest experience in mining subjects generally. One has to be very careful, in discussing anything with him, to use the latest 1915 expressions, and I am very chary in offering any criticism at all.

This paper summarises views which Mr. Rickard has expressed for many years, and views which are no doubt on the whole

sound. One cannot, however, expect Mr. Rickard to be quite *au fait* with the condition of every mining field all over the world. Some of us may have had peculiar and interesting experiences of the behaviour of ore-bodies, and in the course of practical work may have raised for ourselves questions which Mr. Rickard has perhaps not considered; but I am sure we all owe a debt to him for bringing forward a subject like this, and for giving us the benefit of his mature and wide experience in the matter.

With the general contentions of the author that experience has shown that mining becomes less profitable in depth and that, taking an average of all ore deposits mined, richer ores are more common near the surface, I am in entire agreement; in this he has the facts behind him. Mr. Rickard is, however, wrong I believe when he would lead us to suppose that rich ore-bodies cannot be expected in depth. It is true that in his paper he does not exactly define what he means by depth; but apparently he means anything between 3,000 ft. and 6,000 ft. vertical. If the ore goes down 3,000 ft. in his opinion it will not continue in depth at, say, 6,000 ft., and if it disappears at 2,000 ft. he points this out as an evident failure in depth. Mr. Rickard thus has it both ways.

He has himself reminded us that rich ore-bodies have been proved to exist at nearly 6,000 ft. vertical depth. Obviously, then, there can be no chemical or physical reason depending on depth *per se* to prevent their formation elsewhere, even as we shall see later, at even greater depths. It is no doubt true that precipitation depends greatly on temperature and pressure, and that precipitating agents, and in some cases fracturing also, may be more common nearer the surface; but I hold that experience has shown that in favourable circumstances at this depth there need be no bar to the formation of rich ore. Why, then, are ore-bodies so rarely mined at such depths? Here the author has advanced many sound reasons applicable in certain instances; but I think I can prove that he has laid undue stress on the effect of *depth per se*.

My criticism is that the author neglects entirely or most unduly minimizes the effects of denudation in truncating ore-bodies. In his very concise and partial discussion of this aspect of the question he endeavours to prove that ore-bodies are mostly young and have suffered little. He seems to be obsessed with his greater familiarity with the younger ore-bodies of the West of America, and he neglects almost entirely a very large and important class occurring in the older rocks of the globe. In dealing with his own instances he neglects many palpable evidences of denudation. My own experience has lain rather among the wonderful complex of gold, silver, lead, zinc, tin, and copper deposits of Tasmania, in the silver, lead, zinc deposits of Broken Hill, N.S.W., and the gold deposits of Victoria, New South Wales, West Australia, Rhodesia, and the Transvaal. I have long been of the opinion that Mr. Rickard and other geologists have laid far too much stress on the action of purely ascending magmatic waters in the formation of some ore-bodies. When as at Coolac Gundagi, N.S. Wales, I found apparently unaltered hornblende rock containing gold, with dykes containing highly auriferous pyrite, I naturally inferred that the small quartz lodes and gash veins traversing the rock owed their gold contents, to some extent at least, to the country rock itself. It appeared to me that the great irregular quartz makes of Ballarat East, Sebastopol Plateau, Clunes, and elsewhere, and that some at least of the ore-bodies interbedded in the schists of W. Australia and Rhodesia, largely owed their origin to the forces which produced the vast tectonic changes in their enclosing rocks and induced schistose structure, etc., while at the same time they caused metasomatic change in the rocks themselves, silicifying and solidifying them. I believed that such ore-bodies are often largely (shall we say) the waste products of the rearrangements of the elements contained in the minerals making up the rocks themselves, and that their mineral contents might have the same origin. These views are old-fashioned, no doubt. These changes can be expected to be apparent at the greatest depths. At Bendigo and Ballarat in the deepest levels there is no sign that depth has hindered them in any way. We know no reason why, after lateral exploration of thousands of feet, say, on the quartz reefs of the Ballarat Plateau, only a few hundred feet should show payable ore, and we have less reason to infer that the limited exploration in depth, if persevered with, should not show similar results. Most failures in depth in these regions are due to the bottoming of the quartz make, not to the impoverishment of an ore-body.

Much of the misunderstanding of the problem is due to the use of the term "ore

shoot" as applied to ore deposits. When a miner in Rhodesia or Australia discovers an outcrop of pay ore he apparently expects it to extend down a channel towards the earth's centre. Such ore-shoots are exceptional. Ore generally occurs in patches in the plane of the vein at various distances apart. Many mines have been abandoned in depth not because the fissure zone contained no more ore, but because it is rarely that sufficient capital and sufficient optimism are found in conjunction to ensure a thorough search for it in depth and laterally. If one could suddenly denude many abandoned fields 500 ft. to 1,500 ft. they would take on a new lease of life. This occurrence of gold ore in patches, commonly (in Rhodesia) about 70 ft. in width and depth, is the real explanation for much of the absurd nonsense written regarding secondary enrichment in solid quartz veins and the disappearance of gold at water-level, which lies about 70 ft. down. These are, however, all minor matters. I have already stated that my observation leads me to lay much stress on denudation. I should first like to remind Mr. Rickard that even his younger ore-bodies of the Sierras and Cordilleras have certainly suffered far more from denudation than he cares to admit. This denudation on many well-known American camps has amounted to hundreds and thousands of feet. I have not forgotten reading the author's eloquent and interesting description of these forces at work in the San Juan region of Colorado. These mountain chains, geologically young though they may be, have been deeply carved by the hand of time. One question alone proves my contention. From whence came the vast deposits of detrital minerals that mark the course of the valleys traversing these regions? How much denudation has the Mother Lode region suffered where ore even now persists to nearly 4,000 ft.? When we come to consider Mr. Rickard's own "frightful" examples of the affects of depth as he conceives it, we find him on even less firm ground. I know the Tasmania mine and district very well—having worked there. The rich ore-shoot indeed shortened at 1,500 ft., but was neither bottomed nor became poor. There was at least the chance of it opening out again, and no engineer would have advised its abandonment had there not been three million gallons or more of water per day to contend with. Admitting, however, that depth caused failure, we know that mining was started on the very roots of the vein system. The scattered and smaller ore veins in the "Cabbage Tree" conglomerate to the west and its mineralization show the vein to be a very old one. The rocks enclosing the vein are the basement ones of the island, and have undergone denudation which, as we shall see later, amounts to thousands of feet. The lode itself is cut by a gold-bearing deep-lead some 400 ft. deep, filled with detritus, from the formation in which the lode exists. Denudation in other parts of the island can be measured at 3,000 ft. or more. If we state the case thus, "Man started mining the Tasmania lode at the 3,000 ft. level and left rich ore at 4,500 ft." we are, I believe, nearer the truth than Mr. Rickard.

Let us turn to Bendigo. I cannot understand how Mr. Rickard could have been so blind as not to see that the crucial facts are the antiquity of the ore-bodies and their denudation. Was he not presented when visiting this region with impressive evidences of denudation—by glacier, river, and sea, also of continual oscillations of the ground level, and, above all, by the immense wealth of detrital gold torn from the ore-bodies by the hand of time? These were deposits traced back, if I remember rightly, to Miocene times, when large areas were covered with a probably marine deposit much resembling the Rand banket, which rivalled it in area, and whose remnants were afterwards mined from the hill tops as cement deposits and crushed in mills? How many hundreds or thousands of feet of gold-bearing strata were denuded to fill the old river channels with their unexampled wealth? I do not remember having read any reliable estimate from Victorian geologists of the amount. Denuded anticlines give an inadequate measure; but if I state that probably 1,500 to 2,000 ft. of strata and the ore deposits they contained have been removed, I believe I am in reason. In the greatest depth on these fields there is no evidence that the forces of segregation or precipitation have become less active, and payable ore-bodies have been proved to occur at 4,600 ft. Modern equipment alone is required to render them so. Apparently some slate beds are most favourable for the existence of rich ore, and these might again be entered in depth. Let me put the facts in this form: "Man began mining at Bendigo at the 2,000 ft. level. He mined rich ore to 5,000 ft. and at nearly 7,000 ft. lack of organisation and scientific mining called a halt with pay ore in the

lowest level reached." Similar epitaphs might be written over most of Victoria. I know of one zone of magnificent reefs with a strike of miles abandoned at 700 ft. owing to failure in depth!! The world, apparently, badly needs a geologist who will collect data of the relation of denudation to ore deposits. I can only here give a few instances, and suggest lines of enquiry. I note that the ancient Kewatin Peninsula copper deposits have been mined to over 5,000 ft. in depth on a formation that has been ground away under the attacks of the glacial period to an extent that defies measurement. If the ore-bodies of Porcupine persist in depth it will be only because they were in truth deep-seated in origin beyond anything Mr. Rickard likes to think about. Let me suggest that some of the most persistent known ore-bodies occupy regions where denudation has been less active. Consider Butte, Mysore, Kalgoorlie, Virginia City, Minas Geraes, Aldahbut and Charters Towers. Even in much denuded Cornwall tin ore is mined from 3,000 ft. in depth. Economic geologists and engineers reporting on mines have almost entirely neglected this important matter. Let me give one more instance drawn again from my native island. We know from the Linares district, Frias Silver Mine, and many other districts that silver-lead ores may extend to 2,000 or 3,000 ft. in depth as we know it. The silver-lead lodes of the Zeehan-Dundas field occurred in some of the oldest rocks in Tasmania in slates, quartzites, limestone, and ash beds. On a section from W. to E. we have first the sea, then the granite of Mt. Heemskirk rising about 2,000 ft., then the rocks containing the ore veins of the field about 800 ft. above sea-level in a belt about 10 or 15 miles wide, of which Zeehan itself was the richest part. The harder schists and conglomerates then rise 3,000 ft. or more to Mt. Dundas, sink again, and rise to form the main west coast range of mountains. Mt. Dundas is capped by an outlier of the diabase sheet which covers the centre of the island, and this gives an obvious and reliable indication of the extent of the denudation that the surrounding district has suffered, which is between 2,000 and 3,000 ft. Unfortunately these rich silver-lead veins became almost unproductive at a depth of about 500 ft., carbonate of iron gangue replacing the galena, and the prosperity of the field suffered. Messrs. Twelvetrees and Ward, Government Geologists, were ordered to report on the field and the probability of ore being found in depth again. In this report one is presented with many ingenious theories; but the writers have been pitifully blind to the vast panorama of stupendous denudation spread before their eyes. There is no doubt, as they admit, that the mineralization is of early origin, yet the connection so obvious is ignored. I believe that here we have a case where mining started at the 2,000 ft. or 3,000 ft. level, which itself accounts for disappointment in depth. On this field occurred an interesting case of secondary enrichment of ore by the removal of the soluble gangue mineral (siderite) by meteoric water, and there was some evidence of solution and redeposition of the galena itself.

Turning to South Africa, any visitor to Rhodesia must note that the country has suffered enormously from denudation. I think it easily proved that most of the mines are working on the very roots of the veins in the remnants of denuded schist belts. This fact alone accounts for many failures in depth. In the lower lying districts denudation has been heavier, and the results in depth still worse. Most of the ore-bodies are old. The low country districts of the Transvaal come under the same head.

Turning now to the problem of the Witwatersrand, Mr. Rickard, owing to lack of local knowledge, deals with the question of ore in depth in a somewhat one-sided manner. He apparently considers it proved that these deposits owe their origin to ascending magmatic waters. If the placer theory be the true one, and I note that the learned Government Geologist of the Transvaal inclines to that belief, Mr. Rickard is out of court at once; for in that case unless the line of the deposition of the detritus was from N. to S., and there is nothing to indicate this, depth *per se* can have no influence on the gold contents of the "reefs". There are difficulties in holding the placer theory; but underground experience, especially in the Far East Rand, forces one to consider some modification of it. As an example, one finds there lying on the slate an isolated patch or pocket of pebbles, with no sign that anything leading to it formed a channel for mineralizing solutions. Yet there is the rich ore. Of two things I am certain (even if the theory that these beds were mineralized by ascending vadose waters be true), the beds were not mineralized in their present position, and they were not mineralized after the enclosing rocks were solidified to quartzites and the

pebble beds to conglomerates. In other words, the silicification of the Witwatersrand formation may have accompanied but did not precede mineralization of its ore-bodies. It seems to me that, in considering problems such as these, most extraordinary ideas are held by people who should know better of the penetrating powers of mineralizing solutions in more or less inert unfractured and solid rocks. It seems to me that a study of ore-bodies formed from ascending waters shows there must be violent fracturing of the rock, or the constituents of the rock must be readily attacked by the solutions. It seems also that these conditions did not exist on this field after the rocks were solidified and in their present position. I know portions of the Rietfontein Mine intimately, and Mr. Horwood's theory that the banket beds were mineralized *in situ* through the agency of diabase dykes simply amazes me. It is plainly written there, and all over the Rand, just what mineralization the dykes were able to effect. Pyrite in the dykes and from the dykes is coarsely cubical and is easily recognised. The vapours and solutions from the dykes have only succeeded in introducing barren quartz veins, with perhaps some copper and iron pyrite, into the very local fractures made by the intrusion of the dyke or the faulting that preceded it, and also along bedding planes generally under the "reefs". Into the banket beds for a few feet they have introduced cubical pyrite, with galena and zinc blende occasionally, but for a very limited distance. Occasionally there has been some resolution of gold contents, and in exceptional instances replacements of pebbles, but this is local and limited. They have mineralized and darkened the quartzites and impregnated them with pyrite for a distance of at most 10 or 15 ft. from the contact. It is most plain that the dykes have utterly failed to bring about any general or extensive mineralization of the enclosing rocks, and this of course is exactly what one would expect in these hard dense formations. The relation of dykes to faults and faulted isolated areas of reef, especially noticeable in Rietfontein, renders the impossible still more impossible. If ascending solutions brought the gold into the banket and quartzites, it was when one was of the consistency of the Dwyka conglomerate (*i.e.*, pebbles cemented by sandstone and capable of being broken apart by the hand), and the other was a sandstone such as that enclosing the coal measures. If my contention is sound, the question of depth as it appears at present assumes another aspect.

Other questions press for discussion. Admitting that in certain areas there is a marked fall in productivity in "depth," we need to ask to what extent has the size and distance apart of the reefs themselves or a change in pebble contents affected their economic exploitation. Mr. Rickard instances the Village Deep Mine. A section of the banket bodies from the surface shows large reef bodies in all reefs from the Jubilee to the lower portion of the Village Main Reef Mine, with the Main Reef close under the Leader, thus forming large stopes and allowing cheap mining. Below this the South Reef nearly dies out, the Main Reef falls away from the Leader, thus becoming unworkable, and the Village Deep had to work mainly one reef only. In the deepest levels of the Turf section at 4,000 ft. it is reported that the conditions on the outcrop tend to repeat themselves, and the published value of the ore is good. The ore-body from which the Cinderella Deep mined one million pounds worth of gold at about 4,000 ft. in depth is certainly a poor instance of impoverishment. The value of the ore was equal to anything on the outcrop, and very rich ore was often met with carrying visible coarse gold. The causes of the shut down were not due to the grade of ore. At nearly 4,000 ft. in the Brakpan Mine, round No. 2 shaft, occurred a magnificent ore-body of a value of about 60s. over 8 ft. to 16 ft. in height. Few finer stopes have been worked on the Rand. In the Rand Collieries at 2,900 ft. visible gold was more common than in many outcrop mines. The Geduld Mine has a phenomenally rich ore-body in one stope at about 2,500 ft., from which over 5,000 oz. were added to reserve in a short time. Rich ore with coarse gold is common in all these mines. I have seen nothing to suggest that depth has had any influence on the deposition of ore here. Mr. Rickard will, of course, be tempted to reply that he does not deny that rich ore may occur at such depths, but that it is well known that of the area of these mines only about 50 of the ground is payable, and that this alone shows the effect of depth in patchiness of ore. To anyone knowing the conditions, such an argument is without weight, as it is obvious that the poverty of such areas is due solely to irregularities in deposition of the pebbles and their size. This

cannot, of course, be due to depth. Wherever the normal ore-body of large and medium-sized pebbles exist there the gold is to be found. Poor areas mean pinched reef, or small pebbles, a bare contact, or "bastard" reef. The ore-shoots of the Nigel are simply runs of larger pebbles. It, as seems reasonable in the circumstances (assuming these beds were mineralized by ascending solutions) the falling off in value should be rather gradual as pressure and temperature increase with depth, then all I can state is that in some districts of the Rand the curve is so flat as to be unappreciable at 4,000 ft. Carbon, as described by Mr. Horwood occurring at Rietfontein, is very common at these levels in the Far East Rand, and far from dykes.

One other consideration remains to be dealt with. Assuming the "ascensional" theory to be correct and that the "reefs" are as a whole tailing in value in depth along a curve which may be steeper in some districts than in others, we might ask: What denudation have the ore-bodies suffered, and at what original depth below the surface was the rich ore at Brakpan, Cinderella Deep and Turf Mines formed? Again, I do not remember that any competent geologist has given an estimate of this. Denudation, we know, has been great in this watershed, as the younger formations prove. No one believes that these ore deposits are not ancient. Reverse faulting no doubt gives us some inadequate measure of this denudation. The so-called N Reef of the East Rand Proprietary Mines which cuts out on the fault at about 600 ft. vertical was once no doubt superimposed on the outcrop of the Main Reef, and there are other reverse faults. A fair estimate from this data would be 800 ft. to 1,000 ft. If, as seems certain, the Rietfontein series is a down-thrown portion of otherwise denuded Main Reef series, denudation must have removed far more than 1,000 ft. If we estimate it at 2,000 ft. we will not be guilty of the use of too much geological imagination, and we are able to present Mr. Rickard with another instance of rich ore being found at 6,000 ft. vertical from the surface.

More evidence could, I am sure, easily be presented to show that Mr. Rickard has laid far too much stress on the influence of depth *per se* in hindering the formation of ore, and has paid far too little attention to the action of denudation in removing it.

In conclusion I should like to ask is there any known chemical reason why the ores of gold, silver, lead, and copper should be more difficult to precipitate from hot solutions under great pressure than, say, those of iron, zinc, and arsenic.

Mr. Edgar Pam *(Vice-President):* I did not really intend to make any remarks on this paper when I came here to-night, because I felt, after reading it, that it would not be fair to discuss a most carefully written scientific paper without having given a considerable amount of time and study to the subject, and my only object in rising is to support the vote of thanks to Mr. Rickard for giving us the opportunity of printing this paper in our *Journal*, and of listening to the introduction which was read to-night. It is probably one of the most scientifically written papers which has been published in the mining world for a considerable period. I believe a plan was made by Mr. Marriott with a view to showing the values of the Witwatersrand mines at various depths and various points along the strike, and it would be very interesting if we could get a copy, as the compiler had facilities and information at his disposal which we as individuals have not. I am not in any way attempting to burke discussion, but for the reasons I have mentioned I regret that I cannot contribute to the discussion.

Mr. H. Meyer *(Member of Council):* We can regard it as a great privilege that we are able to produce in our *Journal* Mr. Rickard's paper on the "Persistence of Ore in Depth." The paper must be of the greatest interest to mining engineers all over the world, as I venture to say that few would care to deal with so contentious a subject, especially as it deals with every field of economic importance. We here, who have for the most part spent the whole of our useful careers on the Witwatersrand, must naturally turn to that part of the paper which deals with the fortunes of these fields. Mr. Rickard, I think you will agree, dooms us to comparatively early extinction, and illustrates his views by calling attention to the results obtained from the deepest mines, viz., the Cinderella Deep, Simmer Deep, Jupiter, and Village Deep. He also quotes H. H. Webb on the experience gained from the mines of which he was Consulting Engineer, *i.e.*, quoting the words "our experience of our mines." This appears to me to be generalizing on a basis of particular areas. We have on these fields a strike of fifty-five miles, with mines of varying values, and it

must be obvious that a thoroughly scientific statement on the subject under discussion would entail a most elaborate investigation and the collection of a mass of figures and data. I am, therefore, afraid that at this meeting no one will be able to contribute anything which will be of real value as a contribution to the discussion of the paper. Further, no individual member can properly deal with this subject, as it involves so many different interests. Mr. Rickard calls on Mr. Marriott to "make good." I am afraid it would be necessary to appoint a Commission which would have access to all data before any definite result could be obtained.

It is quite possible to select definite areas on the Rand where values have decreased in depth, perhaps not in a regular manner, but nevertheless on the whole. The reef bodies are quite "persistent," but they do not yield the same profit as nearer the outcrop. On the other hand, the opposite has happened. I will take a few instances to emphasize my point. At the Durban Roodepoort, for many years, the South Reef only was worked. This was quite rich, and there is always a tendency to measure the productivity of a mine by its palmiest days. Of late years, the Main Reef has been consistently mined, and its value is lower. The eventual grade of this mine will be the average of the two. The Durban Deep mines both reefs at the same time, and therefore has a lower grade. Coming eastward, I believe that the deeper portion of the Consolidated Langlaagte is as good as the outcrop; the same applies to the Consolidated Main Reef. Coming to the Central Rand, there is a rather wider variation, and in many cases it appears that the deeper mines of the Far East Rand are poorer. This is based on definite results, but there are good reasons which perhaps are not quite apparent at first sight.

In the early history of these fields, the expenses of mining were very high; it was the object of every manager to produce the highest yield possible, and the period could be described as one of "ultra selective mining." Most mines, even in those days, were two reef mines and sometimes three, but the best reef was invariably selected for exploitation, and the others, after a small amount of prospecting, were left to the successors. We are the successors. The outcrop mines are, therefore, always considered on the basis of their best days, but all of you know that the reefs left behind by our predecessors have been successfully dealt with, and the productivity per unit of area has been largely increased. The average grade, however, is naturally much lower.

In the case of the deep levels, the procedure adopted in the outcrops has been impossible and uneconomical. We have been fortunate in reducing the operating costs by better and more economical planning, the properties are larger, and in most cases it has been found the best policy to mine systematically. I do not wish at this stage to discuss the question of selective as against wholesale mining; it is so entirely dependent on the particular conditions. It is scarcely necessary for me to remark that no man, with his eyes open, deals with ore that cannot show a profit under the conditions of working. But the final result is that we are gradually obtaining an average yield which has not yet been determined except for a few of the outcrop mines.

Generally speaking, from my own experience, I find that the rich sections of our outcrop show some persistence in depth; if they do not indicate the same richness they are still quite profitable, and the engineer here is faced more with the difficulty of dealing with high temperature and heavy pressure than with the persistence of values. In these brief remarks I am unable to deal with any specific figures, but merely wish to point out a few peculiarities of these fields as compared with others where perhaps the facts are more obvious. When fuller development has taken place on the Far East Rand, it may be a matter of surprise to the author of the paper that better results are obtained at depth than from the outcrop. So far, the tendency appears to be in this direction. In closing these remarks, I would call the author's attention to the fact that the Village Deep is not the deeper portion of the Ferreira Deep nor of the Ferreira—it is the deep of the Salisbury and Jubilee and Village Main Reef Companies.

Mr. L. W. Macer *(Associate):* The question of the persistence of ore in depth is of peculiar interest to South Africa, and a study of South African ore deposits provides useful data for the general investigation of this important subject. I refer more particularly to the bedded gold ore occurrences of the dolomites of Pilgrims Rest and to the conglomerate ores of the Rand, the similarity of origin of which seems capable of demonstration.

Primary source of metallic constituents of ores.—In assuming the earth to have been once a globe of molten matter, the

density of which increased towards its centre, mining engineers have also taken for granted that this increase in density was due to a concentration of the heavy metallic elements towards the earth's centre, and that from such profound depths the ore deposits were derived. Hence, it was assumed that increased values ought to be met with in depth.

Mr. Rickard's paper gives an honourable burial to a theory which may be safely pronounced dead, but the question still remains as to what extent mineralizing agencies have been able to draw upon the metals of the central core, or, indeed, if any connection has ever existed between this central storehouse and the ore bodies as we exploit them to-day.

In the Pilgrims Rest district, although in many cases enrichment occurs in the vicinity of igneous intrusions, either in the form of rich vertical leaders or enrichment of flat reefs near dykes and intrusive sills, I have never been able to find evidence of gold having been segregated from such igneous bodies. On the contrary, I have noticed that the gold seems to reach its greatest concentration at distances of from a few feet to 100 ft. from such dykes. I have further observed in the case of ore deposits near sills, that deposits of magnetite often occur more or less in contact with igneous rock, but where the magnetite is unaltered it is devoid of gold. Where altered partially or entirely into limonite, I have found it to contain moderate values, and where similar iron compounds occur a few feet above such sills the gold content increases, copper compounds occur and mineralization is generally more pronounced. These observations appear to indicate that the origin of the gold was not in the igneous rock, but, nevertheless, that conditions were produced by such intrusion, favourable to the concentration of ore.

I therefore assume that all such deposits owe their origin to circulating aqueous solutions, whose movements have been stimulated by these igneous intrusions, and that the degree of concentration of ore in such deposits depended upon the *amount* of solution which passed through the precipitating zone, this in turn depended upon the violence of the circulation produced by the proximity of the eruptive.

That ore deposition has occurred in the absence of eruptive rocks, notably in the Otago Fields of New Zealand, is well known, but it is conceivable that powerful downward circulation might be induced in an area of sedimentary rocks, by a corresponding upward circulation in some distant district of igneous activity, the deposition of ores having depended upon the relative nature of the country rocks in the two districts. The necessary energy might also be derived from deep-seated igneous rock similar to basement granite of the Transvaal, which, although cooled at the time the younger sedimentary strata were deposited, still had the energy to erupt its more basic segregations in the form of diorite or diabasic intrusions.

The remarkable similarity existing between the Pilgrims Rest district and the Witwatersrand in regard to the proximity of igneous rocks is shown on the sections attached hereto.

In considering the origin of the metallic contents of circulating solutions, the possibility of derivation from profound depth, I think, can be dismissed, for if not, we would be resuscitating a theory already defunct. I therefore assume that such metallic contents were primary constituents of the outer crust of the globe, and were extremely finely disseminated, if not molecularly distributed, throughout this solidified igneous shell. Subsequent denudation of the igneous shell would not necessarily have affected the distribution of metals, although it is possible they may have followed the basic silicates which appear to have an affinity for gold and silver. The ore depositing solutions have resulted from the interaction of an aqueous atmosphere and the heated interior of the globe. The principles of the action of the resulting subterranean waters have been ably explained by Van Hise* and Posepny†, but I find no support amongst the ore occurrences in the Transvaal for the theory of these writers for the origin of the metals in solution, namely, that they have been introduced directly from the igneous rocks.

I do not dispute that the gaseous emanations from intrusives may have been a contributing and secondary influence in the general enrichment. This would naturally be so, seeing that intrusives are generally the basic portions of the magma. The extent to which they have contributed is, I think, relatively very small.

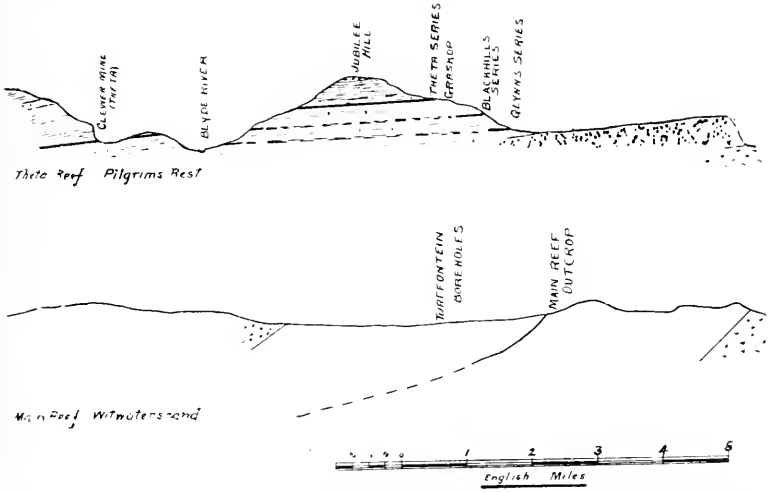
* "Some Principles Controlling the Deposition of Ores," C. R. van Hise, *Trans. A.I.M.E.*, February, 1900.

† "Genesis of Ore Deposits," F. Posepny, *Trans. A.I.M.E.*, Vol. xiii.

Depth to which ore concentration by aqueous solution is possible.—It is obvious that a limit must be reached at which temperature would prevent the existence of solutions. Although this point would not be reached until the water was dissociated, even such a depth would not be great compared with the radius of the globe. It is extremely improbable that capillary action would introduce water to such a depth that dissociation would occur. It is also probable that the lowest point to which water can penetrate is the zone of solution, seeing that at no great depth the solvent action of mineralized waters would be so accentuated by heat and pressure that the effect of any precipitant would be neutralized by an immediate re-solution. The conditions for effective precipitation therefore appear to exist relatively near the surface, and probably extend but little below the depth at which water would boil at atmospheric pressure. It is therefore apparent that ore bodies in steeply inclined fissures cannot continue to great depths, whilst no such restriction applies to ore-bodies which are replacements or mineralizations along the bedding of sedimentary rocks, such as the conglomerate reefs of the Rand and the bedded reefs of Pilgrims Rest, the origin of which is, in my opinion, identical. The mineralogical differences existing in these two occurrences appear to be due solely to the different natures of the original rocks which have been replaced.

In this connection it may be remarked that the conglomerate beds may not have originally consisted of pebbles of reef quartz, but that their present nature may be due to subsequent silicification of material of miscellaneous description. It is certainly difficult to imagine any deposit of crystalline quartz sufficiently extensive to have provided the material for all these beds.

The comparative extent of the conglomerates and of other Transvaal bedded ore deposits.—The extent to which the two reefs have been exploited towards their dip is shown in the diagram, and in the case of the Theta reef some of its richest chutes have been encountered at the extreme dip, namely, in the Clewer Mine, or at a distance of six miles from the original outcrop. In comparison, the depth to which the Main Reef series has been exploited is slight.



The relative position of the eruptive from which the mineralizing solutions derived their energy, and not depth, therefore appears to be the determining factor in the existence of payable ore on the Rand.

Seeing that ore deposition can be demonstrated to be a feature of the earth's extreme outer crust, Mr. Rickard's advocacy of lateral prospecting is obviously

sound as a generalization, but as far as African bedded deposits are concerned I think that the prospector following either the "lateralist" or "profundist" advice will waste much time and money unless he is also guided by a study of possible sources of the ore he seeks, and of the probable itinerary of the solutions to which such ore owes its present position.

The President (replying to a question by Mr. E. J. Laschinger): The paper was submitted, according to a long standing arrangement, before the two societies, but instead of being submitted simultaneously—as is the custom—it was, on account of the war, delayed in presentation here. Undoubtedly since Mr. Rickard has replied to some of the discussion which took place in London, he will also reply to any discussion before this Society. The author, as we all claim to be, is a seeker after scientific truth, and I think we can safely assume that anything that will tend to bring out facts will be welcomed by him, even should matters dealt with in his reply in London be touched upon here. The paper will be open for discussion before this Society at the next meeting, so there is time for any further facts and figures to be submitted, after which Mr. Rickard will doubtless reply in due course. We are including in the *Journal* in a condensed form, for the benefit of our members, some of the discussion submitted in London.

Discussion before the Institution of Mining and Metallurgy, London (condensed).

The President (Dr. F. H. Hatch) said that the subject of Mr. Rickard's paper was of the greatest importance, seeing that the life of most mines was chiefly determined by the downward limitation of the orebodies to the exploration and exploitation of which the mining man directed his energies. Strange as it might appear, it was not always easy to ascertain from the published returns of a mine whether the average metallic content of an orebody was diminishing or not as depth was attained.

Improvements in the methods of mining, of handling and of treating the ore were continually effecting a reduction in the working cost of a given mine; consequently, ore of lower and lower grade could be treated profitably. Thus it came about that rock which at an earlier period of the operations was rejected as 'waste' was at a later stage a valuable material for treatment. In consequence, a lower average yield did not necessarily mean a falling off in grade due to increased depth.

It was true that by systematic sampling and recording of results on a suitably devised assay plan, variations in the metallic content of an orebody from level to level were disclosed; and it was by those means that the cardinal fact as to whether there was or was not an impoverishment with increasing depth could always be established, provided of course that it really was desired to ascertain the true conditions, whether they proved palatable or not.

Mr. W. H. Trewartha-James said it was his very great pleasure to congratulate alike the President, the Institution, and the author upon the brilliant presentation of the paper, which was remarkable from two or three different points of view: the great ability and the great width of research shown in marshalling and collecting the facts; the rugged and outspoken honesty of purpose, which was self-obvious; and although the author pointed out that the question of the impoverishment of orebodies in depth had previously been lightly discussed, this was the first occasion on which a definite theory had been presented in such a form. Undoubtedly most of them had long been aware of this great truth, and had realised it in different degrees as the result of their experience, yet none of them had ever dared to put forward on such a strong basis the material fact which must now be accepted by the profession for the future.

He thought that the facts, that had been arrayed so logically, showed that in the great majority of cases the metalliferous 'ores' in lodes and veins do not and cannot be expected to persist to absolutely indefinite depths. But the word 'depth' was, as the author had pointed out, such a relative term that the speaker clearly saw a danger in placing a wrong interpretation on this word and in the acceptance of the theory of ultimate impoverishment at depth without reservation, for the word 'depth,' which was the gist of the paper, entirely begged the question of precise definition. The speaker hoped he might be able to make that point quite clear by an instance from Western Australia.

In the Black Flag district, which many of them knew, the orebodies had been practically worked out; the extreme depth to which 'ore' persisted would be about 150 ft. to 200 ft. from surface. Five miles

away to the south, at Broad Arrow, the depth would be about 300 ft. from surface. But about 20 miles further south, at Kalgoorlie, on the Golden Mile, after 20 years of continuous work on a large scale the ore persisted to a depth of 2,900 ft. from surface, and one did not know at what 'depth' 'ore' failed to persist. Thus, in one mineral district 200 ft. would be extremely 'deep,' yet in another, a few miles away, 2,900 ft. might be comparatively shallow, that is, not deep at all.

The result of that was that engineers called in, as happened to Dr. Maclaren recently, to advise people who were desirous of exploiting in the wisest possible way the famous orebodies at Kalgoorlie, could get no useful basis for calculation from this theory, and had to make up their minds on some other basis. That was perhaps the material point on which for the moment he must venture to disagree with the author's corollary. At the present stage depth could not be taken into account on the Golden Mile. They did not know what the ultimate depth was likely to be.

One of the Great Boulder lodes passed at depth into the Ivanhoe property, and here, in a zone varying between 1,800 ft. and 2,500 ft. from surface, this lode showed signs of becoming impoverished, and many believed the ultimate depth to which the ore would persist had been reached. The speaker understood that Dr. Maclaren, after most carefully examining the structural geology of that district, came to the conclusion that the impoverishment was due to what, for brevity, they might call an 'uncongenial rock formation' through which this lode had passed in the zone aforesaid. He also ascertained that the 'congenial rock' in which this lode had been so productive would reappear at or about 3,500 ft. from the surface, and therefore advised the owners that they would be warranted in exploring this lode at and below the depth mentioned, and the speaker understood that upon that theory this orebody would be so tested, with a reasonable expectation of finding normal values in that lode.

One had only to go a few hundred yards further south from that point, to the Golden Horseshoe Mine, to come across a most excellent instance confirming that conclusion. It did not for a moment refute the main contention of the author as to impoverishment at ultimate depth, and might indeed be held in a certain sense to support it, as they were all agreed that no orebody could be expected to persist to indefinite depths, but 'depth' might be 500 ft. or 5,000 ft., 10,000 ft. or, to carry it *ad absurdum*, 50,000 ft. That was the most important point raised on the paper; how was the engineer to deal with this basic problem?

At this place perhaps a still more interesting orebody was found, another of the Great Boulder lodes which had been exploited from surface down to a depth of 2,900 ft. with very great success, though it had passed through zones of varying richness, and there had been the same queries as to the depth to which that orebody might be expected to persist.

Mr. Sutherland and Mr. Larcombe had advised the Golden Horseshoe people, in whose property that important orebody appeared to persist, to go in for a policy of deep sinking. They had done so, and the latest news, a cable dated November 9th, stated: 'Main shaft 2,900 ft. level. No. 4 lode have driven north 122 ft., average value 208s. per ton; width 8 ft.' That should be put on record and added to the author's list, not as a contentious instance, but as an illustration of the question of relative depth.

The author had already met that argument in advance, and as he came from Missouri he would no doubt prefer to wait and see! That was quite right, but for the moment at least they had a remarkable case of persistence, a remarkable illustration of the possibility of an extraordinary enrichment occurring under impoverished zones at great depth, and, in his submission, apparently a good instance of justification for the policy of deep sinking.

This question of relative depth was of such importance that he would listen to the discussion with the greatest interest, because the whole thing turned on how depth is to be defined.

Another point. The author strongly expressed the opinion that orebodies did not become richer in depth. He could give one instance, out of several well within the author's knowledge, and ask if these were exceptions that proved the rule. He referred to the Great Flat lode in Cornwall. In the shallow levels in the Wheal Uny Mine, at and near the surface, the value and the productivity of that lode were very low indeed, so low that the mine was always a poor mine, and relatively soon ceased to work. The flat lode continued through North Bassett and West Bassett

into South Francis. In the latter mine he had acquired his first practical experience in mining, and it was one of the greatest achievements of his father to work out and to estimate the position and depth of that lode and to strike it in South Francis Mine at a depth of approximately 900 ft. from surface. From about 900 to 1,500 ft. in depth the lode became remarkably productive, enormous dividends were paid, when standard metallic tin stood at about £50 per ton (in fact, he remembered selling black tin at £33 per ton), therefore they might more easily judge for themselves of its great productivity, and he might add, that lode is still being worked at greater depths (over 2,000 ft. he understood) in South Francis and Wheal Grenville Mines, although with diminished productivity.

He thought that a good sound instance of remarkable enrichment at depth.

With regard to the 'forlorn hope' of which the author spoke, he could give a remarkable instance from his personal experience, the case of the Britannia Mines in Howe Sound, British Columbia.

The company had been exploiting those mines for a number of years, and had spent something like $3,000,000 without any profit to themselves; the material which they were mining was not ore, but a mineral aggregate. A very clever man, whom no doubt the author knew, Mr. Schley of New York, from long study of the subject in all its details as they affected the Britannia Mines, had arrived at the conclusion that before he abandoned the mine he would put in two or three adit levels at considerable depth beneath the great orebodies which they had not been able to work profitably above, and of which large quantities of available tonnage remained.

He desired to pay one of the highest tributes that could be paid to the engineer in charge of those mines, as he declined to be responsible for driving these deep tunnels, and resigned his post because he saw no prospect of attaining a useful object thereby.

The result of that exploitation at depth he thought they probably all knew. The Britannia Mine to-day was one of the most remarkable mines on the West Coast, one of the great profitable producers of copper, and that 'forlorn hope' had become of the utmost importance, not only to that company, not only to that district, but to the whole science and art of mining on the Pacific Coast.

He trusted he had made it perfectly clear that he accepted the author's main contention that metalliferous ores in lodes and veins do not continue or persist to indefinite depth, or, as defined in his earlier writings, that ores do not as a rule persist in depth, much less grow richer, and on the facts as stated it appears that mining engineers as a community must do the same.

With regard, however, to the various inferences, generalizations and corollaries deducible from the main contention, there was room for considerable modification.

Prof. William Frecheville said he would like to commence by complimenting the author on his very interesting and stimulating paper. It was a most important subject and probably many of the members would agree in the main with the author's conclusions. Also, he thought the public—and by the public he meant the people who invested in mining ventures—largely knew that mines were uncertain and that they were what accountants call a 'wasting asset.' The only people who did not acknowledge it were the Income Tax authorities, who would insist on charging the full rate on mining dividends.

In the main he thought many of the members would agree with the author, but at the same time he could not help thinking that in his keenness to prove his point the author had at times given a rather one-sided view. For instance, he quoted the Village Deep as an instance of a mine where the ground in depth was poorer than the ground immediately above it. That was quite true, but the author omitted to mention a neighbouring mine, the City Deep, where the exact opposite was the case.

Then again, the author gave a diagram of the underground workings of a number of mines on the Kolar goldfields, and amongst them the Champion Reef. Turning to that it would be seen that in the case of the Champion Reef the lode at the 2,000 ft. level had been more productive than it was at the 1,000 ft. level; more ore had evidently been stoped out. If that improvement in depth could take place at 2,000 ft., why not at 3,000 ft., or 4,000 ft.? Why should they select an arbitrary point and say there might be improvements from 1,000 ft. to 2,000 ft., but not from 2,000 ft. to 3,000 ft.

Again, the author mentioned the North Star Mine, and insisted that the mine had become poorer, although he acknowledged that the production per square foot had increased with depth.

Further, on p. 40, the author made a statement which might be criticised. He stated, 'But the miner can assert sadly that he has never failed yet to bottom any ore deposit he has found.' A number of mines were quoted which were still being worked very profitably at great depths, so that statement could hardly be taken to be strictly and literally in accordance with fact.

The task of comparing ore ground in depth with ore ground close to the surface was a very difficult one, because they all knew that ore deposits were patchy. Even the ore deposits on the Rand were patchy, and they were much more so in ordinary lodes. The prospector, to start with, walked over the surface of a mine and found a payable orebody. He did not go to much expense; he made a few trenches and found it, and it gave him a clue to follow in depth, and he followed it. But what was his position when he got down to a considerable depth and those orebodies began to fail, as they know orebodies always would fail? If the prospector wanted to prospect he had to work laboriously and slowly in the dark and at great expense; he was seriously handicapped. They knew there were blind shoots, that is, shoots coming in in depth which did not go to the surface, and the inference was justified that in many mines there were other orebodies, other patches, which were never brought to light.

Most people would probably agree with the author that too little lateral prospecting was done as a rule. In ground which was highly mineralised the chances of finding other parallel deposits, or other deposits on the extension, were probably not always sufficiently realised.

In conclusion, in spite of what the author had brought forward, he thought that miners would continue, when they had encouragement, to sink, in the hope that they might find a recurrence of good values, or other orebodies, such as those to which the author had drawn attention in the North Star and other mines.

Mr. E. T. McCarthy: In the few cases quoted by the author there were two or three very doubtful examples, namely, in the Indian and the St. John del Rey mines. Considering again the limited number of mines which had been carried to any appreciable depth, he maintained and repeated that one must be extremely cautious in arriving at anything like a definite conclusion.

Going back thirty years, he considered that the probability of most of the mines now quoted going down to the depth they had since attained was at that time improbable, not so much on account of any doubt then entertained as to the persistence of ore in depth as that the depths now reached were then looked upon as impracticable.

If he remembered correctly, the shaft at the Pzibram mine in Bohemia, which had then reached a depth of 3,650 ft., was looked upon as the extreme depth to which any mine could be sunk; so that when engineers in the past prophesied in somewhat loose terms that the ore would go down indefinitely, it was not intended by them to convey the idea of 'going down' in the exact meaning of that term, but rather as likely to reach down beyond the limits of a workable depth.

In that case the persistence of ore in depth beyond the then reach of man was largely supported by the fact that many mines which had been given up at comparatively shallow depths as worked out, had been found to continue down when they had been resumed.

As the author stated, the question was a comparative one. He believed that the depths of veins, or of a series of lens-like fissures, had some relation to the length, strength, and structure found along their strike, or lateral extension at or near surface. For example, if he found the outcrop of a vein continuing all round the globe, only then would he expect it to go down indefinitely in depth, or until it ended at the centre of gravity.

Passing from the structure of the fissures to that of the ore contained in them, or in other words to the mineralization of the filling of the fissures, one approached a separate problem, because the deposition of the ore was affected by a different series of physical and chemical phenomena from those causing the fissures, though that was not always so. Those were so complex and so many-sided, as was very ably expressed in the paper under the heading 'Theories,' that the chances were greatly against the ore continuing to the same depth as the fissures might themselves go down to.

Continuing, that was quite a different matter from laying down a law of non-persistence in depth, or, as he understood the author to mean, a gradual impoverishment in depth from surface. The impoverishment was often quite of a different character, namely, sharp and sudden, apparently

caused in certain cases by the vein passing through from one class of country-rock to another, as in the Thames, New Zealand. That was well known to miners, who, knowing the country-rock that was congenial to the deposition of ore, called it "kindly country"; and if that rock, after it had given out, should repeat itself, as had been the case in the upper parts of those mines, the vein might become ore-bearing again. To what limits that might go they had little or no knowledge.

Again, there was the question of veins now being worked at depths that must originally have been great. For example, let them take a series of lens-like orebodies becoming enriched as they approached an underlying contact rock, as in the Porcupine district of Canada; there they found orebodies existing which in all probability had been formed long before the Glacial Epoch had swept away their upper portions, and at a far greater depth than would be workable at present had the original surface been maintained. In Siberia, the Russian geologists considered that some of the present known ore deposits were formed at a time when many thousands of feet existed above their present level, which had since been denuded until the great Steppe Plains, in which they were found, were now reduced to a comparatively almost dead level.

If those were right assumptions, it would go to prove that ore fissures of one sort and another might exist at almost any horizon below the present surface of the earth, and that depth in itself had nothing to do with their persistence, but had rather to do with some relation between the lateral extent, strength, and structure of the fissuring, and was relative to that and not to the present ground surface. But when one came to the persistence of the ore itself in depth, relative to any particular fissure, after the upper or enrichment zone had been passed through (which latter, so far as they knew, occurred only in those veins which had been subject to decomposition where they had reached the surface), one was met with much more complex problems. Where, on the other hand, great homogeneity of the country-rock could be determined, through which the fissure passed, a greater continuity of mineralization in depth might be expected than where the reverse occurred.

In a sense, therefore, it would be seen that he was thoroughly in accord with the author's statement that there was a general tendency to non-persistence of ore in depth. But he made the distinction that that general deduction of non-persistence of ore in depth had nothing to do with the actual depths itself, but only apparently so, because of the known mines practically all had been sunk on veins and deposits followed down from or near surface.

Mr. H. W. Turner: With regard to ore giving out in depth, the author could recollect that in the early history of the Kennedy Mine, on the Mother Lode in California, on about two levels there was no profit. The mine was then closed down for some little time, and that would have been the end of it if no one had had any belief in deep mining. As those interested in that part of the country were aware, it was taken up again, and they had now gone down about 1,000 ft. vertically, and had found excellent ore in the bottom.

The Empire Mine, in Grass Valley, was another example. This property being in private hands, there were no public reports, but he was credibly informed that about the time the present chief owner, Mr. W. B. Bourn, took hold of the property a level was run that was nearly barren, and this was perhaps the thirteenth level on which the vein split up into branches. Nevertheless, below the ore came in again, and it is now one of the really good mines in the Grass Valley district, and said to be about 4,000 ft. deep on the incline.

It was a matter of common knowledge that most orebodies were superficial, as far as one could tell, but it must be recollected that orebodies were originally formed in most cases some distance from the surface. That point was also touched on by the author. So that they now had, in consequence of erosion all over the world of various formations, sections of veins at all possible distances from the original summit of the veins, and from the bottom of the vein, wherever that was. As Mr. McCarthy had brought out, probably the effect of great depth had more relation to the fissures than to the ore itself, because fissures could not exist under great pressure, and ordinary fissure vein ores are supposed to be deposited under conditions of lessening temperature and pressure, that is, comparatively near the surface.

The fact that in very many cases ores cease to be profitable at a comparatively shallow depth might lead to the conclusion that most of these workable ores are of

secondary origin. In ordinary sulphide ores, zones of enrichment are very evident as a rule, and even in the case of gold quartz veins there would be some transference of the metallic contents, as erosion proceeded, since gold is to some extent soluble in water, and other metals more so. Moreover, even in gold quartz veins there are usually sulphides present which on oxidizing would form acid, rendering the water a stronger solvent.

Such descending oxidizing waters would find the fissured zone a more favourable place to percolate downwards than the wall rocks, which are usually more compact. Thus the metallic contents of veins that are exposed at the surface are probably in all cases gradually being carried down along the vein, causing a gradual enrichment of the vein. It must not be forgotten that this process is a very slow one, and thus a workable ore deposit may be often the result of centuries of concentration.

That would seem to explain why it was that everything was on the surface and nothing went down; but, on the other hand, it must be recollected that the number of times that miners had gone down after the ore had given out were comparatively few. When the upper orebodies had been mined, and there had been no good indications farther down, occasionally a company sank a shaft deeper; but really there was a large percentage of orebodies which had been developed superficially only, and which had never been tested in depth, because the mining engineer did not want to risk it; he could not see what there was below and nobody could tell him.

Yet there are many places (and the author gives some examples) where fissure ores apex below the present surface. At Tonopah, Nevada, in recent years some important ore lenses have been found by sinking through barren rock and cross-cutting. This camp would have been nearly dead several years ago if only the veins outcropping had been mined. Thus the idea that nearly all the valuable fissure ores are to a greater or less extent the result of gradual concentration has its limit. It is true that the Tonopah ore lenses that do not outcrop belong to a relatively superficial area, but if some valuable fissure veins do not outcrop, why not others at greater depths?

Then the class of magmatic ores like the chalcopyrite-bornite ore of Namaqualand, in S.W. Africa, and the similar ore of Plumas county, California, being formed direct from an igneous magma, are certainly original ores, and may extend to considerable depths. Up to date no such deposits had been mined to over 1,000 ft. in depth, so far as he knew. These magmatic ores, however, presumably are not included in the scope of Mr. Rickard's most excellent paper.

Dr. C. G. Cullis said he had read the author's paper, and listened to the characteristic remarks with which he had introduced the subject, with pleasure and admiration. The matter was one, not only of great geological interest, but of still greater interest and importance from a practical mining point of view.

So far as he was aware, the trend of opinion amongst economic geologists during the last ten years had been emphatically in favour of the view that mineral deposits did not retain their values in depth. That was the author's contention, and at first it would appear that in this matter the conclusions of the miner and the views of the geologist were in agreement. But he did not think the agreement was quite as complete as at first might appear.

A discrepancy arose from the fact that the geologist and the miner did not use the term 'depth' in the same sense. The former employed it in the geological sense, the latter in the much more limited mining sense, and when the author contended that mineral deposits did not retain their values in depth, he practically meant that they did not do so at depths amounting to or greatly exceeding one mile. In other words, commercial ore deposits were limited to the outermost $\frac{1}{1000}$ part of the earth's radius, or in other words, to the uppermost $\frac{1}{13}$ of the earth's crust. The question which the geologist had to ask himself was whether he could accept the principle with such a narrow restriction of the term 'depth.'

There could be no doubt that the author's contention was valid for certain types of ore deposits which had been formed under surface conditions, such as alluvials, residual and sedimentary deposits, and those formed by the downward percolation of surface waters. From their mode of origin it followed that these were necessarily confined either to the surface or to comparatively slight depths beneath it.

But there were other important types which had been formed under deep-seated conditions, such as veins filled by ascending mineral solutions, and deposits genetic-

ally connected with plutonic igneous rocks —contact and pneumatolytic deposits, and magmatic segregations. With regard to these, while geology provided evidence from which it might be inferred that even they could not extend *indefinitely* in depth, it did not, he thought, favour the view that they were necessarily confined to the meagre depths of the author's contention. Judged by its mode of origin, he thought the magmatic segregation was a type of ore deposit which might be expected to extend, not indefinitely, but at least to depths exceeding the present limits of mining operations.

With regard to lodes of the type to which the author had confined his attention, a number of geological conditions would influence the depth to which their metalliferous minerals might continue of commercial value. One was the vertical range through which the precipitation of the ore minerals had taken place—the interval between the depth at which precipitation had begun and that at which it had ceased. In view of the slowness with which temperatures and pressures varied with ascent or descent in the earth's crust, this interval might very well be as much as 5,000 ft., or even twice that amount. Another condition was whether the existing outcrop exposed the lode near the upper limit of precipitation, or whether it had been long subject to erosion and consequently occurred deeper down, nearer the lower limit. There was also the possibility of the lode passing in depth from one formation to another, and the favourable or unfavourable effect of such a passage.

He ventured to express the opinion that while there must be a certain proportion of lodes in which these conditions of precipitation, exposure, and country, not to mention others, were such as to result in impoverishment or actual extinction of values at shallow depths, there must be others in which, the same conditions being favourable, values might be expected to continue to depths well below those at present imposed by mining practice.

They had to thank the author for having brought before them an extremely important subject. It was one which was certain to give rise to a most interesting and useful discussion.

Mr. C. Baring Horwood said there were only one or two points to which he would like to refer that evening. First, in commenting upon the author's very able paper it seemed to him that perhaps the most striking feature was that there should be any need for such a paper at all. However, he frankly admitted that, unfortunately, the need did exist, and he trusted that the paper would have the effect of calling forth much illuminating discussion.

It was true that there were certain classes of deposits in which no deterioration in the value of the mineral contents with depth need necessarily be expected, so long as the deposits themselves persisted and no change occurred in the enclosing rock. As examples of such might be mentioned, amongst others, replacement deposits such as were often represented by galena or hæmatite in limestone; and also those due to magmatic segregation, such as certain magnetite deposits.

However, with regard to those having an origin similar to that of lodes, which were he believed those with which the paper really dealt, he thought it was generally admitted that below the zone of secondary enrichment a decrease in value in depth was naturally to be expected. Such decrease in value was due to well-known chemical and physical laws which were easily understood and which had been lucidly expounded by various well-known authors. In this connection he would like especially to call attention to an able presentation of the subject by Mr. W. H. Weed,* in a discussion on "Secondary Enrichment at Cripple Creek." In that discussion Mr. Weed dealt with the point just now raised by Dr. Cullis, namely, the vertical range in which deposition might, or was likely to, occur, which seemed to him (the speaker) to be the question on which the explanation of the whole matter largely depended.

Referring now in detail to some of the points raised in the paper, the author might perhaps be surprised to learn that when the speaker visited Przibram in the early part of 1911, the silver-lead deposits were still being worked at the Adalbert Mine, and at a greater depth than that mentioned by Mr. Rickard. They had, however, become considerably poorer, the galena contained less silver and it was more intermingled with quartzite. He hope to refer to this again in greater detail at the next meeting.

It was with the author's remarks concerning the Rand that he especially wished to deal that evening. He thought it was necessary first of all to point out that,

* *Eng. and Mining Journal* (1903), p. 553.

speaking generally, the economic limits of a lode deposit could usually be represented in vertical longitudinal section, or in section along the plane of the lode, by a figure resembling the vertical section of an inverted cone, the base of which corresponded to the profitable portion of the outcrop along the strike. These cone-shaped figures were often somewhat squat, frequently they approached equilateral shapes. They were seldom greatly elongated in comparison with their inverted bases, which meant that the extent of the profitable ore along the dip was seldom more than (or even as much as) the length of the profitable extension along the outcrop. This being so it was clear that some relationship existed, as Mr. McCarthy had just maintained, between the profitable extension of outcrop and the persistence of economic possibilities in depth. Further, the existence of this relationship appeared to him to be easily explainable: cracks or fissures which might perhaps be well defined at the earth's circumference gradually became smaller and smaller, and finally tailed out and vanished as the rocks in depth became more compact.* Thus deep-seated mineralizers gained admission at great depths to fissures through minute cracks of slight lateral extension. As distance was gained towards the surface the fissures became less tight, and in consequence the solutions in ascending spread out fan-shape on either side along the fissures.

In the case of the Rand the function of fissures had been performed by big, well-defined beds of conglomerate which had necessarily differed from ordinary fissures, inasmuch as the walls had been kept apart by intervening pebble beds. Curves compiled to show the relationship between the widths of the banket at various levels showed clearly that the widths diminished gradually as depth was attained; that diminution in width doubtless being due to increasing pressure. These beds had thus formed porous channels throughout their entire length down to great depths. Consequently, instead of the lateral boundaries of the economic area tapering down to an apex, the average mineralization, whatever its value, at various horizons in depth should extend, at those respective horizons, the same distance, measured along the strike, as it did at the outcrop; and experience on that goldfield had shown that it did. In other words, the economic limits of this particular orebody would be eventually represented in vertical longitudinal section, not by a figure resembling the section of a cone, but by a rectangular figure having, needless to say, its longer axis parallel to the strike of the beds.

The author had alluded to the fact that the two deepest mines on the Rand were closed, but the conclusion suggested seemed quite unwarranted. It was true that those two mines were temporarily closed, but the author omitted to state that they were situated in what were admittedly poor areas of the Rand; and he also omitted to mention that in certain poor areas there were outcrop mines also which, although not worked out, had been closed. The vertical depth of the deepest workings on the Rand was less than one-fiftieth of the length of the profitable extension of the outcrop along the strike. The Village Deep, to which the author also referred, was being worked at an inclined depth of some 7,500 ft., and it was obvious that the economic limit of depth had not yet been nearly reached.

Mr. F. P. Mennell said that judging from what they had heard already he was afraid he would find himself somewhat in a minority, because, if anything, he really did not think that the author had gone far enough in the printed part of his communication. His general treatment of the subject was one which absolutely accorded with what he (Mr. Mennell) had put into writing himself; in fact, six or seven years ago he had ventured to assert that it was the universal experience in mining that all ores diminished in value, apart from certain fluctuations in various ways, as one followed them downward from the surface.

He thought that those who like himself had been engaged in looking at prospect after prospect—he had looked at hundreds—and tried to pick out those which were going to be profitable in the future, would have come to the conclusion that it was the rarest exception for any mineral deposit to go down to any considerable depth.

Of course it was a fact that mining would not be in the position that it was if it were

* The results of research work indicate that openings cannot exist at a greater depth than about 30,000 ft., and under certain conditions at a much less depth. *Vide* 'Ore Deposition and Vein Enrichment by Ascending Hot Waters,' by W. H. Weed. *Trans. Am. Inst. Min. Eng.*, Vol. xxxiii. (1903), p. 750.

Also 'On the Limiting Strength of rocks under Conditions of Stress Existing in the Earth's Interior,' by Frank D. Adams *Jour. Geol.*, Vol. xx., No. 2.

And 'Some Principles Controlling the Deposition of Ores,' by C. R. Van Hise in 'The Genesis of Ore Deposits,' 2nd Ed. (1902), pp. 286-288.

not for those exceptions, but they *were* exceptions, and he had been very much surprised to find that engineers whose reputation reached all over the world should have got up that evening and more or less attempted to controvert what the author had so ably put forward.

He had not had their experience, but he had certainly had very considerable experience. He had started as a pure geologist and worked as one for a number of years before having anything to do with mining, therefore he might have been expected to take what had been called the 'professorial' view of ore deposits as against the view of the ordinary professional man.

Mining, as someone had said, had rested under the imputation of being a very uncertain profession, and he thought one of the causes had been this fact, that the views which were put forward in the paper that evening had not been more generally accepted in the past by mining engineers.

In the country where he had been resident for ten or eleven years they were often visited by gentlemen who came from Johannesburg, like the last speaker. The Rand 'banket' happened to be a very exceptional deposit, but it was undoubtedly getting poorer as it went down. They had only to look at the returns to see that, they need not go any further. The Rand has been a great exception, and yet even that exception was proving the rule in the literal sense by undoubtedly becoming poorer as it went down. They could scarcely be expected to accept any such views as that of the previous speaker, who seemed to suggest that it might go down for about 40 miles.

Geologists had devoted themselves very largely in recent years to what might be considered a kind of transcendentalism. One heard a great deal about magmatic segregation and pneumatolysis and all sorts of things. When, however, one came to inspect deposits which were supposed to have been formed by magmatic segregation, or by pneumatolysis or by any such methods, one always found that they were among the most uncertain of all deposits; they died out quicker than almost any others. If one took Rhodesia, where there were the biggest chromite deposits in the world, he thought his friend Mr. Maufe would bear him out when he said that the orebodies went down to very small depths; they were simply lenses which occurred on the surface and were dug out on the hillsides. They were on top of the serpentine too, which was the wrong place if they originated from segregation.

Further he might say that he had examined a number of deposits which personally he should consider had every right to be regarded as having been formed at a very great depth below the present surface of the ground. In fact, in some cases one would expect that at least 20,000 ft. of denudation had taken place since the deposits were formed; but when one came to enquire into the working of the deposits, and to see what had been done when one went down on them, one found exactly the same tale as one did on all those little deposits which were due to secondary enrichment: as one got down they decreased in value; they petered out; they got shorter along their strike, and he was sorry to say that really no very definite deduction could be drawn from the geological conclusions which one came to.

He might say that he had always had most faith in deposits of that description. He had committed himself to recommending that they should be developed, and in several cases this had been done with great success. Two properties of this class were afterwards floated for sums aggregating a million, and one had already paid dividends to the extent of twice its capital. The author of the paper had lately, however, made very pessimistic references to this very mine in his magazine, and it certainly seemed to be on its last legs in spite of its previous success. What he would like to emphasize particularly was that even those large mines to which the author drew attention in his paper were, after all, the rare exceptions; that is to say, the great mines which had gone down thousands of feet. There were only a few of them all over the world. Where were all the thousands and thousands of mines which people had put money into, and put it in with every hope, which had only gone down a few hundred feet, even apart from the ones which were due to absolute surface enrichment? Of course the majority of deposits were simply due to surface enrichment; they died out at ground water-level, and that was the last that was heard of them, and he did not think anyone would be justified in going farther down.

Of course there were fluctuations of values in mines. In Rhodesia there was a striking example in the Globe and Phœnix mine, which was treating ore of treble the

value it did ten years ago after passing through a stage when the engineer said the directors would be well advised to look out for another mine with the money they had. But he did not think that was going on for ever, and when they were asked to believe that any ore deposits were going down indefinitely, he thought they were justified in taking a very strong stand against such ideas.

They had only one gold mine in the world—his knowledge was largely that of gold mining—which had gone down below 5,000 ft. vertical, although there were numbers of gold mines of the patchy class which had encouraged people to sink to great depths. For instance, at Bendigo, in Australia, it had been found that each successive level on the average got poorer than the one above, and when the greater depth, which was about 4,500 ft., had been reached, all the mines had become gradually unprofitable; it was only at higher levels that they had paid.

The question of what metal one was dealing with made a considerable difference. Copper behaved quite differently from gold and did not show the same surface enrichment, and often showed a very poor zone below water-level. One gentleman had quoted an example of a copper mine which had been given up and then was found very rich below a certain poor zone. Of course that was a common thing with copper and was easily explained by the fact that it became leached out at certain portions of the lode. He had been connected with a copper mine which, owing to leaching processes, did not average 0.3 of copper for a long distance, and then became quite rich lower down.

The question of lateral extension was about the only thing with regard to which he would be a little inclined to part company with the author. There were several mines where, on his own recommendation, work had been done along the lateral extension rather than in depth, because he considered that in depth the prospect of the ore being profitably worked was not very great. He was sorry to say that in no ordinary case had further valuable deposits been discovered. In two cases extensions of the lode were discovered, but they were of an exceptional character. In one case the lode was obviously faulted, and in the other case perhaps one ought not to call it lateral extension at all; it was the case of an orebody which pitched at a very small angle to the surface, so that when one drove along under the second or third level, one continued in the orebody for a much greater distance than on the first. To the ordinary eye it looked like a lateral extension, but the stuff never outcropped on the surface where it was discovered along the strike.

Mr. John S. Olver said his experience was almost entirely confined to the Rand, of which he thought he had rather an unusual knowledge.

They had heard a very emphatic opinion expressed and he was not going to oppose that opinion; he would simply say that he himself had not formed an opinion. He very much doubted if the Rand could be regarded in the same way as other ore deposits. He rather took exception to the author drawing illustrations from the Rand at all. Personally he was inclined to think that it should be excluded, and that one could scarcely deal with it on the same lines as one could deal with fissure veins.

What the source of the lode was he should not like to say. Personally he had seen no explanation that met all the facts as he had seen them, and he was inclined to doubt whether there was any real relationship there between value and depth. He was not giving a final opinion in any way, but so far he had not seen any definite proof of it.

Mr. C. Baring Horwood asked to be allowed to correct a misconception on the part of Mr. Mennell with reference to one point in his discussion. He had not made such an absurd suggestion as that the profitable ground on the Rand would extend to a depth of 40 miles: what he said was that the economic limit in depth had not yet been reached, and that the depth of the deepest workings on the Rand represented less than one-fiftieth of the length of the profitable extension along the strike, and that they might, therefore, reasonably expect an appreciable further extension in depth of the profitable ground. Mr. Mennell had entirely misunderstood what he had said with regard to the final shape of the vertical section of the profitable ground. What he meant was that in depth the gradual fall in grade, at various horizons, could be averaged over a lateral extension corresponding in length to the extension of the profitable outcrop, and that consequently the final figure representing the profitable economic area of these deposits would be represented in vertical section by a fig-

are approximating to a rectangle instead of to a section of a cone.

Mr. S. Vivian said that two of the speakers had referred to a point that occurred to him, viz., that the Witwatersrand reefs were not typical for the purpose of that discussion. To explain his reason for thinking so, he would like to put forward a theory of his which he had held for some time with reference to the gold values in those beds. He thought that, having regard to the comparative regularity in thickness of the reefs over such extensive areas, it was a fair conclusion to come to that they were deposited in a horizontal position.

He thought, as the gold values ran so generally throughout them, that it was also reasonable to conclude that the gold entered into the reefs while they were still horizontal.

If those deductions were correct, he did not see how their present depth could have anything to do with variation in value any more than the position of any area along the present strike had, because had the uptilting pressure come from a different direction, what was now the strike might have been the dip of the beds, and vice versa.

CONTRIBUTED REMARKS.

Dr. J. Malcolm Maclaren: I beg to enter a vigorous protest against Mr. Rickard's interpretation of my views on the persistence of ore in depth. I have never suggested that 'Mr. Rickard's effort to smash a fallacious generalization is likely to prove "exceedingly harmful both to the mining engineering profession and to mining capital,"' nor have I ever demurred from his general conclusion, which, as I understand it, is a denial of the truth of the theory of the persistence of ore to indefinite depths. I have, however, endeavoured to show that the '*universal* acceptance of Mr. Rickard's proposition would prove harmful,' and, stating the same idea elsewhere in other words, have said that I considered his negation of the possible persistence of ore in depth 'pernicious doctrine, *if universally applied*.' Briefly, we are in complete accord concerning the rule, but while Mr. Rickard appears to maintain that there can be no exception, I insist that there may be exceptions of economic importance. Indeed, it has been on account of our general accord that I have elsewhere discussed the minor point of divergence rather than the general ground of agreement. Nevertheless, I am bound to confess that while in theory I accept the rule, in practice I am always hopeful of finding the exception.

On Mr. Rickard's own showing such exceptions do exist, and are of economic importance, inasmuch as the deep levels of these mines yield a profit.

In discussing this subject elsewhere I have pointed out that the rare exceptions to the general rule are found in pre-Cambrian rocks or in the unique Mother Lode belt of California. The three principal cases cited by Mr. Rickard, viz., Kolar, St. John del Rey, and North Star, bear out my contention. The argument from the Tasmania does not appear to be helpful. The Tasmania has failed at shallow levels and not in depth, as we understand the term in the present discussion. Fifteen hundred feet is not deep as modern mines go, and numerous mines in the notoriously shallow Tertiary andesitic regions have carried ore to greater depths. Moreover, the water that has been the bane of the Tasmania has rendered and doubtless will always render it impossible to say whether the decrease in the value of the ore is permanent or whether it merely marks a poor horizontal zone between two productive shoots.

One of Mr. Rickard's conclusions demands critical examination, viz., that one that postulates that 'most orebodies . . . are geologically young.' So much may be admitted at once for all that ore dependent on Tertiary igneous activity, but I venture to assure Mr. Rickard of my belief in the pre-Cambrian age of most orebodies in the pre-Cambrian regions of India, Australia, South Africa, and North and South America (the two last only in the east of the continents and therefore away from the influence of the Cordillerean and Andean eruptives). Proofs are too lengthy to detail here, but the following are facts typical of one region, viz., Western Australia.

There the latest members of the pre-Cambrian complex are fresh unaltered diabase dykes. At Ravensthorpe (Phillips River) they cut through and absorb the copper lodes for lengths of 80 ft. At Meekatharra and Nannine they similarly absorb and intersect auriferous lodes. The best example is probably that of Robinson's mine, Nannine. Here the magma has not been sufficiently hot and fluid to absorb the rich gold quartz of the lode, and a drive across the diabase dyke in the track of the lode showed numerous blocks and frag-

ments of undigested quartz, of a value fully equal to that worked elsewhere in the lode. We have here, therefore, an example of a pre-Cambrian orebody that owes nothing to later enrichment.

I am informed by Mr. Montgomery, Chief Engineer of Mines, Western Australia, of similar cases of the intersection of gold lodes by pre-Cambrian acid igneous dykes. In similar fashion the geological age of many orebodies is definitely known and is not a matter of pure assumption, as Mr. Rickard apparently infers, nor is the geological evidence of their age affected by the discovery that another and far-removed deposite, as the Homestake may be much younger than had previously been believed.

Mr. Rickard also believes that orebodies may be the result of chemical reactions long post-dating the waters that precipitated the primary ore. It may be so; I certainly cannot produce any positive evidence of value to the contrary, but in my experience it has lately seemed to me that primary ore is normally deposited during a single continuous geological period.

Were it otherwise, it is then difficult to advance any satisfactory explanation of a common condition underground in which we have two or more fissures of the same age, in the same rock, and with the same gangue, one of which is metalliferous and the others barren, except on the assumption that only one fissure was open to metalliferous solutions at the geological moment of ore-deposition, and, further, had ore-impregnation been continuous over long periods, or had it been oft-repeated, then it is exceedingly improbable that other fissures would not have been accessible to impregnation.

I gladly seize this opportunity of expressing my indebtedness to Mr. Rickard for a thoughtful and stimulating exposition of his views on a subject of importance now and of still greater importance in years to come as mines grow deeper and outcrops become rarer.

Mr. W. F. A. Thomae: An important point to remember in discussing this paper is the definition of the term 'in depth,' given on p. 5. It would, however, have been better had the author stated this as his own definition for the purpose of his paper, and not that 'anyone' by the term means 'that the ore continues as far as it is likely to be mined.'

It is quite possible for an engineer, in expressing the opinion that a certain ore will improve in depth, to mean that it will be better at a greater depth than that already opened up, whether that depth be 50 ft. or 500 ft. or 1,000 ft. He certainly would not mean that the value of the ore would increase in arithmetic progression to the lowest depth capable of being mined. The author's definition is therefore essential to the paper, while I disagree that anyone would always use it in that sense.

Taking the definitions as given, there may be exceptions to the general rule of impoverishment, though it seems to me that even the St. John del Rey is not really one, as the drop from 54s. to 46s. in value seems to show. But as a general rule it surely cannot be denied.

There are still many points about ore formation that are imperfectly understood, but it is acknowledged that increased temperature retards deposition from solution, also that the greater density of rock and diminution of easily permeable channels would retard metasomatic action. These factors increase with depth, and, therefore, as far as their effect on ore deposition is concerned, increasing depth must make itself felt sooner or later.

The trenchant facts of experience seem to indicate that the depth defined in the paper is sufficient to show it. I must therefore hold that depth *per se*, producing as it does structural change in rocks, must cause impoverishment in ore, emphasizing with the author, that by impoverishment is meant a lowering not only of the grade, but also, or alternatively, of the quantity of the ore.

The discussion seemed to indicate a fear that this fact would militate against enterprise in mining. This I venture to deprecate. Impoverishment does not mean non-persistence. The fact that impoverishment in depth, as depth is defined, does not mean that, because a certain orebody has petered out at, say, 600 ft., further sinking will not reveal another lens or make of ore at greater depth, *vide* the Pahang Consols. It does not mean that, because ore has become poor at, say, 1,000 ft. owing to structural geology, it may not be rich again at 2,000 ft., if the disturbing factor is there removed. It does not mean that good profits may not continue to be made even with a falling grade, as at St. John del Rey. It does not roundly condemn deep sinking. Whether further development should take this line in any particular case must still be left to the judgment of experienced engineers,

after careful study of the local characteristics of the orebody in question.

It is only meant to warn against the fallacious assumption of *undiminished* grade and tonnage to the limit of possible mining.

Some speakers referred to the definition of 'ore,' and I think it should be pointed out that on p. 5 the author gives us an example of the inadequacy of his own definition. He says: 'The geologist would call 5-oz. gold *ore* at 3,000 ft. an evidence of richness in depth, but if the quantity of 5-oz. *ore* proved to be insufficient, it would not repay the miner for his trouble in finding and removing it.' Therefore it is *not* ore, according to his own definition, and yet he has no better term to give it.

Mr. T. Crook: In his paper on the question of the downward extension of ore deposits in the earth's crust, Mr. Rickard claims that ore deposits are restricted to relatively shallow depths. He uses the term depth in a practical mining sense, implying that depths exceeding 5,000 ft. are great depths; and he draws the important inference that deep development is undesirable and doomed to disappointment, since as a mine gets deeper the ore becomes more and more impoverished, until ultimately, and in comparatively shallow depths, it ceases to be of any economic significance.

This inference is such an important one, and it is so very pessimistic, that it cannot be allowed to go unchallenged. One is compelled to ask if Mr. Rickard has done justice to all the facts in making this sweeping assertion. It must of course be admitted that, in very many cases, mines have been abandoned after sinking through comparatively shallow depths. But is it really sufficient to explain this fact by saying that ore is non-persistent in depth? There are reasons why this question should be answered in the negative, at least for many mines.

The different parts of an extensive ore deposit are not always continuous. Where there is a break of continuity there is a possibility that development may proceed in the wrong direction, perhaps for want of geological guidance. Even where it proceeds in the right direction, failure may result from mere timidity or lack of capital. One may indeed quite fairly argue that many mines have been abandoned in which deeper development would have had good results; and no doubt many a valuable orebody has been missed through lack of the confidence, geological guidance and capital necessary to continue development.

It is not easy to admit that increase in ordinary working costs is a negligible factor at depth; but without pressing this point too far it may be claimed that, so far as the facts are concerned, persistence of ore in depth is not the only factor to be considered in explaining the failure of profits as mines get deeper. In special cases it is undoubtedly the vital factor. But Mr. Rickard has put the case generally; and as a universal proposition it is not true that the unprofitability of mining at depth proves non-persistence of ore. It is far too complicated a problem to be dismissed in this simple way.

To take ore deposits in the lump in dealing with this question makes confusion. Some ore deposits are by their very mode of origin limited to shallow depths, and are characterised by lateral rather than vertical extension. To these Mr. Rickard's dictum may be applied appropriately enough.

Other deposits, however, are characterised by great vertical extension, and change their character at different horizons, like the famous deposits at Butte. In dealing with this type of deposit, we have an important fact to assist us. We know that it is a type intimately associated with igneous intrusions of comparatively deep-seated origin; and we also know that the ore extends to the roots of the intrusions.

Not every geologist will admit that such ores have been sweated out of the igneous magmas during the later phases of intrusive activity. There are some who argue that descending waters have done the work. If it were to be admitted that ore deposits never arise as the direct result of intrusive action, then Mr. Rickard's case would be very strong indeed; but I gather from his paper that even Mr. Rickard does not subscribe to this antiquated notion. He admits that igneous intrusions probably exude metalliferous solutions which precipitate their metallic contents partly in the igneous rocks themselves, and partly in the overlying sediments, on their way to the surface.

If we admit this view, and it is probably the correct one for certain deposits, what is its bearing on the question of extension in depth? It implies the probability that certain ore deposits of magmatic origin had their birth at depths of not less than 10,000 ft.; and it is not pseudo-scientific to assume that some deposits have been born at depths exceeding this. But even where they arise at shallower depths than this, if

due allowance be made for the effects of all the agencies that operate to produce subsequent changes in depth, there is nothing in the facts to upset the belief that exploitable ore deposits exist plentifully at depths exceeding 5,000 ft. At any rate, there is no reason why the miner should abandon as hopeless the downward development of certain deposits well beyond the depths represented by the large majority of abandoned mines.

I confess much sympathy with the view that, since certain deposits of great vertical extent have probably been deposited from solutions that have traversed the earth's crust in an upward direction, the most logical thing to do in developing them is to follow them downward towards their sources, not forgetting, of course, that under certain conditions they may expand laterally.

The study of ore deposits has recently gone through an important phase in which great additions to knowledge have resulted from the investigation of the enriching effect of processes operating only in shallow zones. The glamour of this study, and its great practical importance, should not be allowed to blind one to the fact that there exist some important types of ore deposits that have not been affected by these processes, and that appear to owe their origin to processes operating deep-seatedly.

Mr. Rickard tells us that 'if the crust of the earth in a productive mining region were suddenly rendered transparent to the radio-active vision of a transcendental geologist, he would see many an orebody that had been missed in the hurry to follow the lure of deep sinking.' One might indeed go further than this and say that, if the practical miner were only sufficiently sagacious, he would see on looking into the earth's crust many orebodies, both in the shallow and deeper zones, that he has missed through want of geological guidance. It may be, of course, that with all the help the geologist could give he might have missed some of them; but he has far more to gain than to lose by allowing the geologist to co-operate with him in his subterranean gropings.

A great deal remains to be learned about ores in depth. There is plenty of room for hope, and in the existing state of knowledge a more optimistic attitude than Mr. Rickard's seems permissible. It is much too late in the day to pretend that the 'brutal facts' of mining are inconsistent with the 'theories' of geology. In science, even in mining science, facts and theories stand very much in need of each other; and if they are divorced the facts will suffer as much as the theories. Now that mining has got a branch of geology (mining geology) to minister to its special needs, the facts and theories can fraternize much more successfully than they could in former times, and it is highly desirable than this fraternization should go on.

In conclusion, I would add my thanks to Mr. Rickard for having opened the discussion on this important subject in such a very interesting and stimulating paper.

Mr. F. L. Terrell: Some time ago when in Rhodesia I found, in many instances, that while the reef was fairly rich near the surface, there was a falling off of values on reaching the sulphide zone, and a gradual falling off in depth.

While there I spent a short time on the tin properties in the Abercorn district. These tin 'lodes' are greisen and pegmatite dykes cutting through the formation, which is composed of beds of banded ironstone quartzite and schist, near the contact of the granite. On the surface large specimens can be found which will assay as much as 20 % of SnO_2. Sinking on these lodes, very little tin indeed was found, although surface indications go to prove that the portions of the lodes eroded have been rich in cassiterite. Much money has been spent on these lodes, but as far as I am aware not one of the lodes have proved persistent in depth.

There appears to be no doubt that Nature herself has bottomed the ore-shoots in ages gone by. I wonder if the author has ever noticed if the lateral extent of an ore-shoot provides any indication of its depth; that is, is there any proportion?

NOTES ON THE USE OF POTASSIUM PERMANGANATE AS CYANICIDE IN SAND-FILLING SOLUTIONS.

(Read at October Meeting, 1914.)

By R. A. Cooper (Member).

DISCUSSION.

Mr. Morris Green (*Associate*): Mr. Cooper, in his treatment of the subject, has omitted to consider the effect of the presence of the oxides of manganese that

are precipitated a portion of the precipitate is, for a time, held in suspension—on adding permanganate to a cyanide solution. This omission is of some consequence, since it leaves one or two points vague, the explanation of which would otherwise be clearer.

The writer pointed out some time ago (The Action of Oxidisers in Cyaniding, February, 1913) that the precipitated manganese oxides have a certain oxidising effect; also, a portion of these oxides is held in suspension. It was shown that the oxidising effect of a cyanide solution to which excess of permanganate had been added and the precipitate filtered off was the same in periods of time varying from 12 to 26 hours. The oxidising effect of a solution to which permanganate had been added, and from which the precipitate *immediately* had been filtered off would be different, owing to the suspension of a portion of the manganese oxides. This latter case closely resembles conditions obtaining in actual practice as outlined by Mr. Cooper, the oxidising effect in practice being somewhat greater owing to the presence of the precipitated manganese oxides. As Mr. White points out, the presence of the oxide of manganese in fine colloid suspension may bring about the subsequent destruction of cyanide in the stope.

Mr. Cooper's reference to the high cost of certain cyanicides is, in view of recent events, very pertinent.

NOTES ON THE DUST ALLAYING PLANT AT THE FERREIRA DEEP.

Read at November Meeting, 1914.

By S. Newton (*Member*).

DISCUSSION.

Mr. E. H. Johnson (*Past-President*): In seconding the vote of thanks to Mr. Newton at the November meeting for his interesting paper on dust allaying, Mr. Toombs referred to the work done in this connection by the East Rand Proprietary Mines, and I think some amplification of Mr. Toombs' remarks may be of general interest.

The investigation to find some economical means of allaying the dust from the dumps, which of late years has been accentuated by the finer grinding in tube-mills and cleaner separation of sand and slime by classification, was commenced in November, 1912, on the suggestion of Mr. Anderson, the Superintending Engineer of the East Rand Proprietary Mines. The first experiment was with molasses, followed by silicate of soda in the form of waste slag and other materials, forming a list very similar to that which Mr. Newton refers to as having been tried at the Ferreira 12 months subsequently. In September, 1913, none of the previous experiments having proved entirely satisfactory, the binding and adhesive qualities of the heavy black soil common to low-lying ground and vleis along the Witwatersrand were investigated. (Motorists who have strayed from the hardened road into this type of ground can testify to these two qualities.) Incidentally, I might refer to Mr. Toombs' mention of the Boksburg fire-clay, which, like "the flowers that bloom in the spring, has nothing to do with the case." This black clay, loam or bog is common throughout the Reef, plentiful in low-lying or vlei ground and absent on higher ground. It is easily miscible with water, and forms a slimy emulsion that remains in suspension almost indefinitely. It can be pumped with an ordinary plunger pump without water service to glands. On spraying, the loose sand combines with it to form a thin crust, which effectively prevents the wind lifting the sand.

Any form of mixer will serve for preparing the emulsion, the apparatus adopted here being an 8 ft. x 8 ft. cone having a vertical shaft passing through the bottom of the cone and supported by a cross beam at the top. To the shaft are attached agitating paddles, and the shaft is driven through a bevel gear beneath the cone by an extension of the pump shaft. Above the agitators and attached to the agitator shaft a wire netting basket is fixed into which the soil is fed. This basket is immersed about a foot deep, and serves to prevent lumps falling into the apex of the cone. The water is fed through a perforated pipe near the bottom of the cone, thereby assisting the agitators. The suction of the pump is taken from about two feet below the surface, and near the apex of the cone is a valve to allow the discharge of any grit that may come in with the clay, and injure the pump. The pump used is a Cameron feed pump, 3 in. x 5 in. stroke, driven by compressed air. The cost

for air is 3d. per hour. A 3in. pipe leads to the top and across the dump, with connections at suitable intervals for attachment of a 1 in. hose. The spraying nozzle is ½ in. diameter, so that the hose and spraying is easily handled by one boy. The ratio of water to solid used is approximately five to one.

For sixteen months this covering has proved an effective preventative of dust from the dump, and during that time it has been subjected to extremely heavy winds, and also very heavy rains without affecting its efficiency. The rain tends to improve the covering somewhat, by making a better combination between the soil and sand.

Reverting to Mr. Newton's paper, I find difficulty in reconciling the figures given, and perhaps he can explain. He states in the appendix:—

Duty of spraying pump, 50,000 sq. ft. per hour

Cost, including salt, power, labour, etc., 6d. per 1,000 sq. ft.

Average time run per week, 22 hours.

Total cost per month, £25.

Then $50 \times 6d. \times 22$ £27 10s. per week, or £110 per month.

The assumption that the treated slime from a decantation plant costs nothing is scarcely correct. A certain recovery is obtained from solution returned from the slime dams, which at a low estimate will amount to 3½d. per ton for every ton pumped on to the dump, where this solution would be irrecoverable.

For the purpose of comparison I repeat Mr. Newton's appendix, with the corresponding data from the East Rand Proprietary Mines:—

Ferreira Deep.

Duty of spraying pump per hour, 50,000 sq. ft.

Cost including salt, power, labour, etc., 6d. per 1,000 sq. ft.

Spraying pump has run on an average 22 hours per week.

Cost of erecting plant, £900.

Amount of salt used, 5 to dry slime.

Total cost per month during six months, £25.

East Rand Proprietary Mines.

Duty of spraying pump per hour, 18,000 sq. ft.

Cost, including salt, power, labour, etc., 2d. per 1,000 sq. ft.

Spraying pump has run on an average 15 hours per week.

Cost of erecting plant, £250.

Amount of salt used, nil.

Total cost per month during six months, £9.

The credit of first effectively solving the problem of allaying the dust from the dumps belongs primarily to Mr. W. T. Anderson, to whose initiative and encouragement the effort was due, and, secondly, to Mr. P. R. Nothersole, who carried out the experiments and discovered the properties of the black-clay and its adaptability to this purpose.

I have brought in a sample taken from the Driefontein Dump, and for comparison one sprayed with salt and slime at one of the other dumps, the Village Main Reef I think, and also a sample showing sand treated at the laboratory.

Mr. H. Meyer *Member of Council*: Mr. Johnson has not mentioned the cost of the original material, transport, and apparatus.

Mr. E. H. Johnson *Past-President*: I took that at 2s. a ton. The actual cost is 1s. 2d. a ton. We are favourably situated in having it close by.

Mr. S. Newton *Member*: I should like to say that a typographical error was made in reference to the time given in my paper. It is not 22 hours per week; but on the basis of 22 hours per month.

The meeting then terminated.

The following papers will be submitted for discussion at the meeting of the Society to be held on Saturday, the 20th March, 1915.

THE THEORY OF TUBE-MILLING.

By H. A. White, M.Inst.M.M. (Member of Council).

As experience has greatly increased our knowledge of the most economical methods of using the tube-mill, it has, at the same time, opened up fresh problems which still await both theoretical and practical solution.

The whole subject may conveniently be considered under the two heads of (I.) design and (II.) operation. These may be further subdivided as follows:

I. *Design.*—(a) Shape; (b) Dimensions; (c) Linings; (d) Discharge; (e) Feed; (f) Prime mover; (g) Measuring apparatus.

II. *Operation.*—(a) Speed of revolution; (b) Load of pebbles; (c) Working level of load; (d) Size of pebbles; (e) Coarseness of feed; (f) Amount of feed; (g) Moisture in feed; (h) Fluctuations in power and speed.

(Ia.) *Shape.*—From the fact that the operation of a tube-mill depends almost entirely upon centrifugal force arises the theoretical deduction that a long cylinder is the most suitable shape. The length of this cylinder will be determined (if trunnion bearings are used) by strength of material which it is economical to use for its walls, and generally by the amount of feed which it is possible to pass through the tube. The maximum length must obviously not exceed that at which the discharged product is as fine as the final pulp required, and in order that capacity may be fully utilised it is probably best that the pulp discharged should not contain much more than half its weight reduced to the desired fineness. Thus if the final product desired is of 90 material, the tube discharge should not have much more than 45 of this grade. This ensures a full supply of material to be crushed up to the very end of the mill, though the feed end will be working under the more favourable conditions. The practical limit will favour length (giving larger units), while the merely theoretical aspect would be in the other direction, which tends to remove from the circuit all material fine enough as soon as it is produced. The short tubes temporarily in favour in some parts of the world principally owe their adoption to some other feature of their design, whose effect is not fully appreciated, or to the fact that the tonnage of sand fed is too small for the most efficient work in longer tubes.

The only other shape suggested is a combination of the cylinder with a double cone, and this design is apparently based upon a misapprehension of what takes place in the tube of ordinary shape. (See Bulletin, Amer. Inst. of Min. Eng., July, 1913, p. 2702.) This mill aims to provide special arrangements for graduating the power to the work required, but having a varying diameter must be driven at a speed which, however carefully chosen, must be incorrect for a great portion of the load. The varying crushing power, in a direction parallel with the axis, so expensively attained in the Hardinge Mill, is inevitably present in the ordinary mill, in a direction at right angles to the axis, and the well known tendency of the smaller pebbles to travel toward the discharge end of the standard tube mill, gives the advantage of reducing the power consumed at the point where the feed has naturally become finer. It is obvious that the unit of capacity is more cheaply obtained in a cylinder than in a double cone, and no defence of any sort has been put up for the cone at the inlet end.

It might be suggested that a slightly tapered cylinder (say 2 in. wider at inlet than outlet) would be useful, as the lining, if 2 in. thicker at inlet than discharge, as is common with silex liners, would have a cylindrical surface when new, and while giving a uniform "life" along the tube, would improve the tendency to keep the larger pebbles near the feed end and provide a longer average drop at this point where the feed is coarsest.

This construction would not interfere with giving correct speed more than the wear of liners does at present, but it is doubtful if the increased capital required could not be better utilised in a different type of liner, especially as the extra diameter at inlet would have an unfavourable effect upon the adjustment of the "working level" subsequently referred to.

(Ib.) *Dimensions.*—Having concluded that the shape most suitable is a long cylinder, the relative and absolute dimensions of length and diameter require further consideration.

Diameter. The diameter of a tube mill will depend upon the size of the largest pieces in the feed and the largest pebbles which can conveniently be used. The weight of the pebble and the diameter of the tube

are to some considerable extent mutually compensating factors whose product will determine the largest pieces permissible in the feed.

The longest drop any pebble in a tube will get will be in the layer corresponding with an "angle of departure" of 45°, if the speed of revolution is sufficient to give any layer that angle; if not, the outer layer will have the longest drop. The drop cannot therefore exceed 0.8 x diameter of "circle of reference." (See Diagrams IVa. and IVb.) (The "circle of reference" is the locus of centres of pebbles touching the lining of the tube, and the "angle of departure" is the angular distance from horizontal diameter of tube at which a pebble begins its curve of flight.)

The ft. lb. of energy required to smash any piece of average banket will vary as the square of its diameter, and may be from 2 to 6 ft. lb. for a 1 in. diameter piece, in accordance with shape, etc.

Assuming a maximum of 6 ft. lb. for 1in. feed, pebbles approximately spherical with Sp. Gr. 2·75, and tube linings of 4 in. thickness, if the largest pieces of banket to be fed were of 1in. diameter the following would be diameter of tubes required:—

For a 4 in. pebble a tube diameter of 39 in.
For a 3 in. pebble a tube diameter of 76 in.
For a 2 in. pebble a tube diameter of 226 in.

In a similar manner it may be calculated that a standard tube of 5 ft. 6 in. x 22 ft. will not take a feed coarser than ½ in. unless a sufficient proportion of pebbles larger than 2 in. are present.

Similarly, if it were required to design a tube to take banket ore direct from the rock-breakers with a maximum size of 3in., the diameters corresponding with various pebble sizes, putting the blow required at 36 ft. lb., and allowing for a lining 4 in. thick would be:—

For 6 in. pebbles a tube of 5 ft. 2 in. dia.
For 5 in. pebbles a tube of 8 ft. dia.
For 4 in. pebbles a tube of 14 ft. 6 in. dia.

As, however, the blow is mitigated by resistance of semi-fluid pulp so that in practice it is found that feed larger than ½ in. is somewhat unsatisfactory with the 5 ft. 6 in. tube, though the pebbles present therein will average about 0·65 lb. (2¼ in.) with lumps of rock fed in at an average of about 2 lb., it is probable that a suitable tube for taking ore direct from the rock-breakers would have an inside diameter not less than 8 ft., and the coarse ore fed in to form pebbles would be lumps from 6 in. to 8 in. cubes. In such a case it would probably prove advantageous to use a second row of ordinary tubes for the final reduction.

Length of Tube.—The capital cost for the cheapest design will depend upon the relative cost per square foot of barrel and ends. If a square foot of end costs "n" times a square foot of barrel, then the length must be "n" times the breadth. In designing a tube all the costs which are affected by the diameter must be tabulated for the various sizes and a similar set of calculations made for the various lengths and upon the above principles a determination can then be made of the best relative dimensions. I am informed that the ratio so determined will be about 4 to 1, which implies that the ends, including driving pinion, trunnions, etc., cost four times as much as the cylinder per square foot. It has, of course, been assumed that the internal capacity is a reasonable measure of the work a tube mill can be made to perform, which is in practice found to be roughly accurate.

Other considerations are of at least equal importance with considerations of prime cost, and the principal one appears to be connected with the amount of feed it is practicable to pass through the tube so that the full length may be utilised. Four hundred tons per day is quite common on the Witwatersrand, and this seems to be sufficient for the ratio of 4·1 with a 5 ft. 6 in. diameter tube, but it would be difficult to set a limit for the possible amount if the feed and discharge inlets are suitably arranged.

Before proceeding with the other divisions of our subject, a reference is necessary to the experiments recently carried out by the Mines Trials Committee, which has been kind enough to give permission to make use of the results obtained.

This series of experiments was carried out in the ore-dressing laboratory of the South African School of Mines and Technology by Professor G. H. Stanley, assisted by Mr. Morris Green.

(1) *Objects of the Trials.*—These experiments were undertaken to observe and measure—

(a) The effect of varying speeds of revolution.
(c) The effect of varying kinds of tube lining.
(e) The effect of different "working levels" and the ascertaining of the most suitable discharge screens.
(d) The effect of varying sizes of pebbels and feed coarseness.
(b) The effect of varying loads of pebbles.

(f) Testing the practicability of using a continuous automatically replacing lining of the pebbles themselves.

During the conduct of the trials it was further decided to

(g) Measure the fluctuation in speed and power consumption during a revolution of the tube.

Description of apparatus employed.—The tube was a cylinder of ½ in. iron with a diameter of 77 in. inside and a length of 18 in. mounted on a heavy cast-iron spider and driven by gearing. The back was of 1 in. ribbed cast iron mounted on a shaft 7 in. diameter. The front was closed in by 1 in. iron screening, prevented from bulging out by two strong iron supporting plates. A man-hole was used for introducing or removing pebbles. The spur-wheel on the shaft was cut steel gearing with 150 teeth, pinion 20 teeth, counter shaft belt pulleys 30 in. and 20 in. diameter, motor pulley 8 in. (see Diagram facing this page).

Motor.—A 25 H.P. shunt wound motor was supplied, and the speed varied by using suitable resistances, and this gave no trouble. The calculations on power consumed do not take into account any variation in efficiency caused by changing speed, as the engineers did not consider this of importance.

Pebbles. Imported Danish pebbles from 2 in. to 1 in. diameter were used, and a very fine spray of water had to be directed upon them to keep down the dust. All the observations are upon the tube loaded with pebbles only.

Measuring instruments.—The voltmeter and ammeter employed were carefully calibrated from time to time.

The tachometer supplied was of little use as its readings were not sufficiently "dead-beat." A stop-watch was therefore used in counting number of revolutions. Various subsidiary apparatus was used to determine levels and "angles of departure" of pebbles, etc. As the light was insufficient, no cinematograph record could be taken, as intended, but numerous diagrams were drawn from eye observations to illustrate the appearance of the moving pebbles under the various conditions. Several oscillograph records of the current consumed, and tracings of an electrically vibrated tuning fork upon smoked paper wrapped round the shaft were also taken, the latter being used to determine the variation of speed during a complete revolution of the tube.

Special Note.—All the observations recorded were made with the tube mill containing pebbles only, thus securing simplicity and definiteness. Any practical application of the deductions when sand and water are present requires due allowance for these factors.

1(c). *Linings.*—The liner used for most of the experiments was of plain concrete, and this wore away very rapidly, especially at first while it was fairly new, and the wear was not very regular. A concrete surface does not appear to have much "grip" on the pebbles, and according to the calculations made on observed figures it allowed more "slip" than even the unlined tube.

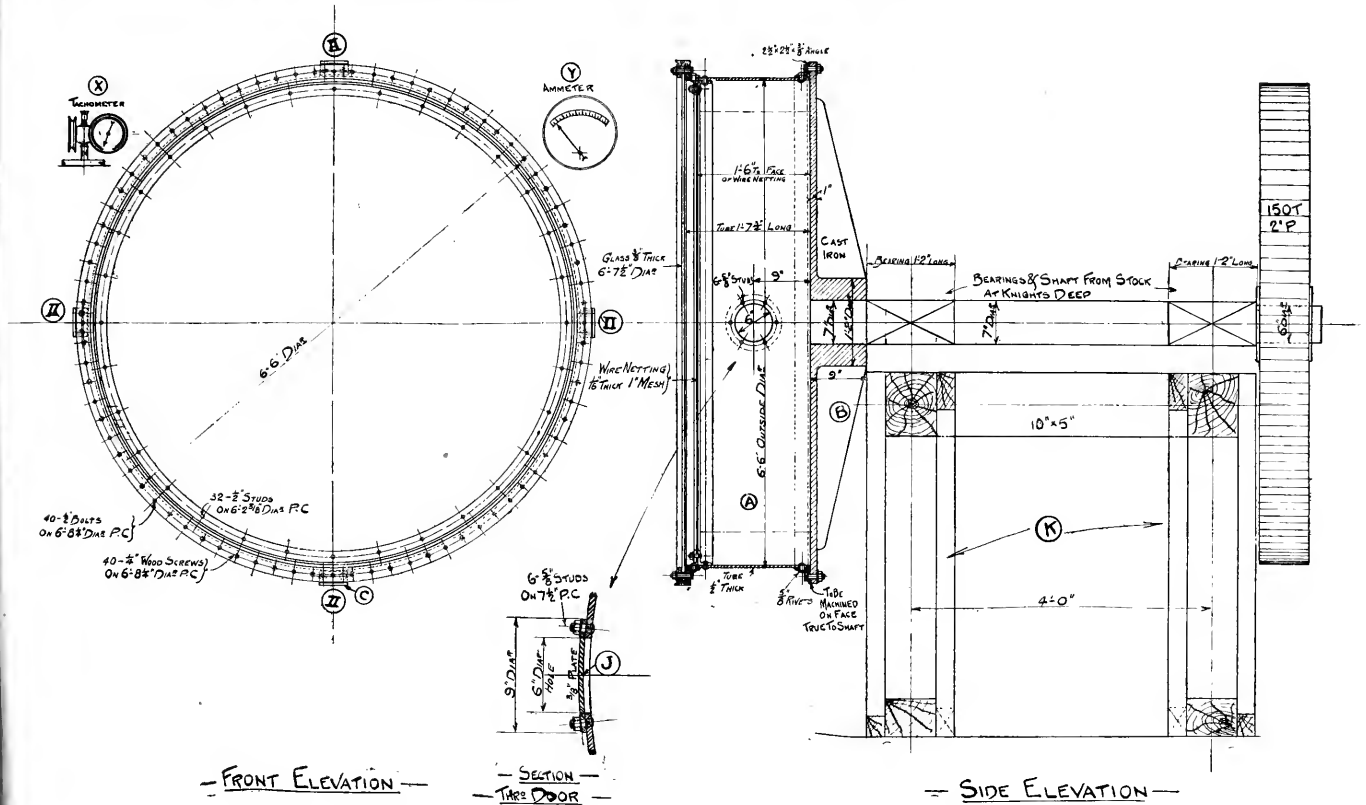
A steel bar liner, arranged alternately flat and upright in the usual manner, was found to take up pebbles well till the actual working surface consisted of pebbles. The wear in this liner was not measureable during the experiments made with it, and the working surface appeared to take up the pebbles well, giving a minimum amount of "slip."

No other kind of liner was available for experiments, but the experimental tube mill remains available for trials in this direction, if at any time such appear to be desirable.

An objection to the bar liner of the Osborne or similar types is the greater amount of amalgam held up thereby as compared with the silex liner. This may be to some extent minimised by using cement as a backing to fill up interstitial spaces, but even with great care the retention of amalgam is capricious, and may vary on the same mine from 500 to 2,500 oz. It will, however, do almost as efficient work at the first hour of starting as at the end of its life, and this compares favourably with the indifferent work done when a new thick silex liner is started up.

Possibility of using automatic liner of pebbles.—Upon driving a tube beyond a certain critical speed, one or more layers of pebbles may be made to adhere to the circumference, and thus form an automatic lining, which would replace itself as it wore out. In order to test the practicability of this, the experiments recorded in the following paragraphs and Table I. were undertaken.

In all cases the power consumed in lifting pebbles was much smaller than the full capacity of the tube at more ordinary speeds, and other drawbacks are detailed.


X
TACHOMETER
Y
AMMETER
II
II
II
II
C
6'-6" DIAR
32-½" STUDS ON 6'-2⅜" DIAR P.C
40-½" BOLTS ON 6'-8¼" DIAR P.C
40-¼" WOOD SCREWS ON 6'-8¼" DIAR P.C
GLASS ⅜" THICK 6'-7½" DIAR
WIRE NETTING 1/16" THICK 1" MESH
6'-5⅝" STUDS ON 7½" P.C
9" DIAR
6" DIAR HOLE
⅜" PLATE
J
1'-6" TO FACE OF WIRE NETTING
TUBE 1'-7½" LONG
CAST IRON
6'-5⅝" STUDS
9"
6'-6" OUTSIDE DIAR
A
B
TUBE ½" THICK
⅝" RIVETS
TUBE MACHINED ON FACE TRUE TO SHAFT
BEARING 1'-2" LONG
BEARINGS & SHAFT FROM STOCK AT KNIGHTS DEEP
7" DIAR
150T 2" P
10"×5"
K
4'-0"
FRONT ELEVATION
SECTION THRO DOOR
SIDE ELEVATION

In practical experiments on a working scale it may be anticipated that considerable saving in lining expenses would be made, that tubes of large diameter (7 ft. or more) should be employed with a corresponding decrease per cubic foot in capacity, but that there is no reason to anticipate any decrease in tons –90 produced per H.P. employed. Possibly an extra lock up of amalgam in the tube may be expected.

TABLE I.

Diameters and R.P.M. to Make Layers of Pebbles Continuous.

The diameters given refer to the "circle of reference" and show measurement inside tube lining less diameter of pebbles used. A layer of pebbles is said to be "continuous" when it adheres to the circumference of the tube or lining during a complete revolution.

Diameter.	1st layer continuous.		All Pebbles Continuous			
	R.P.M. Calculated.	R.P.M. Observed.	R.P.M. Calculated.	Load.	R.P.M. Observed.	Load.
in.				in.		in.
53	36·4	36·0	40·8	0	46·0	+ 5
54	36·1	—	40·5	0	—	—
55	35·7	—	40·1	0	—	—
56	35·4	34·7	39·8	0	46·2	+ 2
57	35·1	35·5	39·5	0	47·8	+ 4
58	34·8	35·0	39·1	0	48·5	+10
59	34·5	34·5	38·8	0	47·2	+12
60	34·2	(39·0)	38·5	0	47·2	– 7
61	33·9	—	38·1	0	—	—
62	33·6	(38·0)	37·8	0	49·0	0
63	33·4	—	37·5	0	—	—
64	33·1	33·2	37·2	0	46·0	+ 6
65	32·8	—	36·9	0	—	—
66	32·6	33·5	36·7	0	51·2	+12
67	32·3	—	36·4	0	—	—
68	32·1	33·7	36·1	0	51·2	+17
69	31·9	—	35·9	0	—	—
70	31·6	—	35·6	0	—	—
71	31·4	—	35·3	0	—	—
72	31·2	—	35·1	0	—	—

It will be observed that slightly more than the theoretical speeds are required to make the first layer continuous, and that though an increase of 3·9 to 4·1 R.P.M. is theoretically sufficient to take up all the pebbles to the circumference, in practice an increase of 10 or 12 R.P.M. is required (owing to excessive slip in inner layers); the greater speed is naturally required with very heavy loading as the inner circle formed by the last pebbles taken up is proportionately smaller. To take up two or more layers on the circumference, the R.P.M. would require an extra revolution per minute for each layer, more or less, in accordance with size of pebbles and diameter of layer formed. It is evident, therefore, that if the speed were set for two layers of pebbles to form an automatic lining, a variation in speed of 3 either way would result in increasing the layers to three or reducing to one. There is consequently no great difficulty in this direction, but the fact that an increase in speed is in these cases accompanied by a decrease in power consumption requires the provision of a constant speed prime mover.

In all cases where a continuous layer of pebbles is maintained on the liner, it was found that the smaller pebbles reached the circumference; experiments with a very small model tube mill with sand present showed that if the speed were gradually increased only sand was found on the circumference, but a rapid increase of speed up to the required point enabled the much larger beads used to form the automatic layer. It may be anticipated that in practice as the layers on the circumference become reduced in size by wear, the tendency will be for their place to be taken by sand, and especially by pyrite or amalgam, because the smaller particle can be held there by the centrifugal force where a larger particle would fall, as its centre would fall outside the limiting circle fixed by the speed of revolution.

The following experiments were made to illustrate the possibility of maintaining stable conditions under these conditions:—

(1) Height of pebbles above axis, 10½ in. (+10½ in.).

Liner thickness, 1·25 in. (at beginning).

Liner thickness, 1·25 in. (at end of 5 hours).

The tube was run at 34·8 R.P.M. with several continuous layers of pebbles for 5 hours without material variation, and although the pebbles wore themselves down to + 6 in., no measurable wear took place on the liners. The average rate of wear with R.P.M. 23 to 33 was from 0·1 in. to 0·2 in. per hour with the concrete lining used.

(2) Height of pebbles above axis, 7 in. (+7 in.).
Liner thickness, 7 in. (at beginning).
Liner thickness, 7 in. (at end of 3¼ hours.
The tube was run at 38·5 R.P.M. with several continuous layers of pebbles for 3¼ hours, but though the pebbles were worn down to a considerable extent no measurable wear took place on liner (concrete).

(3) In order to determine the effect when the first layer of pebbles was not quite on the circumference, a further experiment showed a wear of 0·2 in. in three hours, against a wear of 0·7 in. in three hours when no such protection was afforded.

I (d). *Discharge.*—While the inlet opening to the tube need only be large enough to pass the largest pebbles to be fed in, with an allowance for the wearing lining, the diameter of discharge aperture may either be large enough to bring down the level of the semi-fluid pulp to the required point, or the discharge screen may be provided with a lifting scoop delivering to a discharge outlet of smaller diameter. (See Section II (e).

This discharge scoop works in a chamber cut off from the end of the tube by a false end plate, provided with a screen having the diameter required; and iron balls or a few stem ends might be placed therein to smash up the small pebbles passing this screen, after the manner of the ingenious arrangement by Mr. Thurlow at the New Modderfontein.

I (e). *Feed.* The Schmitt Feeder is very satisfactory and flexible, and where this is used there is no reason why the inlet to tube should not be of sufficient diameter as to obviate the possibility of jamming of the pebbles fed and to allow larger rock to be used for this purpose where desired.

I (f). *Prime Mover.*—The peculiarity of the tube mill that an increase of speed may have the effect of reducing the power consumed renders it necessary that the prime mover should be of a constant speed type to avoid the possibility of a dangerous runaway. This should be more especially necessary if the automatic lining of pebbles were used, as in this case the normal speed is much nearer the danger limit.

I (g). *Measuring Apparatus.* The present usual method of controlling the pebble feed to tube mills by means of an ordinary ammeter in a three phase circuit has several disadvantages. In the first place it does not discriminate between loads above and below the maximum power consumption, and of course the reading is directly influenced by the variations in the voltage of the power supply which at times are very considerable. Where measurement of power consumed must be relied upon, the most advantageous instrument would be a graphic recording watt meter and a daily chart from this would be a useful check upon the regularity of maintenance of the required pebble load. If the same level of load is maintained in a silex liner it is obvious that the power consumed will vary with the wear of the liner, but experience will enable this variation to be properly allowed for and the chart may be marked as a guide.

It will be obvious that in comparative trials with various adjustments an integrating wattmeter, duly calibrated, is the only reliable instrument where motors are used and figures based upon comparative ampere readings are quite useless.

It is to be regretted that no reliable method of directly measuring the pebble load, while the tube is running has yet been devised, though some operators consider that the load may be roughly judged by pushing a thin iron rod through the centre hole of the discharge screen and thus judging the depth of the fallen pebbles.

In ordinary tube mill installations driven by three-phase motors, one integrating watt-meter would be sufficient for all the tube motors to enable power charges to be properly allocated, one ammeter and one voltmeter would likewise be sufficient for the whole circuit, while each motor switchboard should be provided with a recording watt-meter, and an ordinary indicating watt-meter should be placed in a secure position, clearly visible from the pebble feeding point.

II (a). *The effects of varying speeds of revolution.*—In all the experiments carried out under this head, the power consumed when the tube was running empty at the various speeds was first determined. The nett power after deducting these results from the final reading refers therefore to the power actually consumed in lifting pebbles, plus internal friction inside the tube. It may be assumed that the power actually consumed in lifting pebbles will be a fair measure of the crushing capacity of the tube, and that at anything like reasonable speeds and loads the amount of fine

material produced per H.P. will not vary to any great extent, in the same way that heavy stamps have a high crushing power per machine unit, but no greater product per H.P. than lighter stamps.

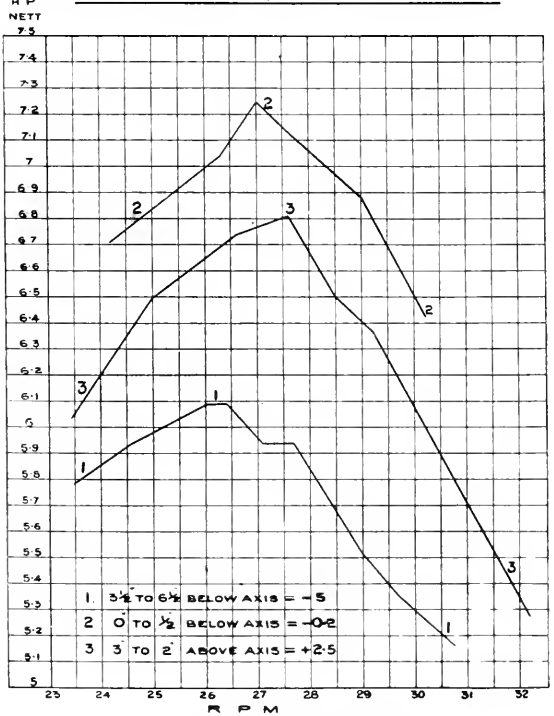
It was very obvious on watching the fall of the pebbles, that greater loads required higher speeds to develop full power consumption, the effect of the greater speed being to provide adequate paths of free fall for the greater load of pebbles. If the speed was too low or the tube too crowded at the speed used, the inner layers of pebbles had no space for free fall and were merely carried round and round with slight relative movement, which in most cases would mean wasted power. The low power shown with excessive loads is due to the further effect of the pebbles, after filling up this inner cavity, heaping up at the bottom of the tube and thus lessening the fall of the other pebbles present until the speed was great enough to carry up the excess pebbles on to the rim. A free fall was thus again secured, but of course the effective tube diameter was reduced by the pebbles adhering to the lining.

It was very difficult to observe the "angle of departure" (measured from a horizontal diameter) at which the various layers of pebbles left the rim, but as the speed measured for the "angle of departure" 90°, (*i.e.*, when the first layer is "continuous"), agrees well with that calculated

— DIAGRAM I —

— BAR LINER 5¾" THICK, INCLUDING PEBBLES —

— INSIDE LININGS = 66½", CIRCLE OF REFERENCE = 64" —



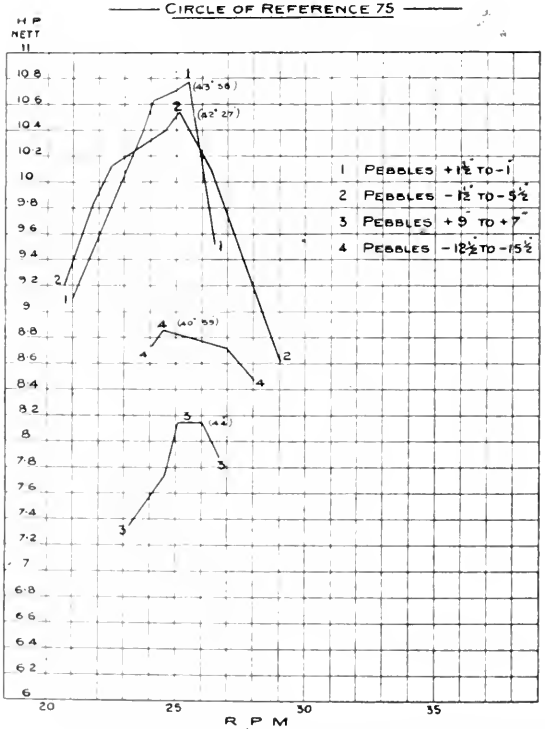
theoretically, the attached Diagram III is given to show the R.P.M. corresponding with this angle for the outer layer for a standard tube. As the angle of repose for the wet pebbles is about 33°, there is practically no lifting of the pebbles till the speed corresponding with that angle for the layer is exceeded (when the tube is half full or more).

It has not been found possible to devise any means of gauging this free space in a working tube, as it always contains a few jumping pebbles, though observations with a small model show it to be free from sand. At present there is no means of determining from power readings upon which side of the maximum loading effect a given tube is working. Experiments were made

— DIAGRAM II —

— UNLINED TUBE 77" —

— CIRCLE OF REFERENCE 75" —



It was observed in practicable cases that the angle of departure corresponding with the inner layer cannot be much less than 30° if free fall is secured, and this is the limit of loading at any speed beyond which there is a loss of efficiency. If more pebbles are added there is no free space between the rising layer and the falling pebbles.

through a wide range of speeds with a great variety of loads, but only a selection is given in the following tables and diagrams sufficient to fully cover the different phenomena observed, and to give the evidence upon which the conclusions are based.

It may be observed, in passing, that a very small model on a 6 in. scale differs insignificantly in visible effects from the

experimental tube employed. (See *Journal* Chem., Met. and Min. Soc., S.A., vol. 5, May, 1905.)

An interesting visual observation was that at very low speeds the smaller pebbles gathered towards the centre of the tube, at practicable speeds no segregation was noticeable, and at speeds beyonds the critical the smaller pebbles worked out to the periphery. This is also observable with sand and glass beads in the miniature model.

The noticeable points in the following Tables II. and III., and attached Diagrams I. and II., are the increased speed (necessary for maximum power) required for increased load: the slow rise in power up to the maximum and the quicker fall thereafter; especially from the point at which pebbles are picked up on the lining.

Further observations on the effect of speed variation will be found under the next heading, where they can be more suitably discussed.

DIAGRAM III

REVOLUTIONS PER MINUTE
CORRESPONDING WITH VARIOUS ANGLES OF DEPARTURE
OF OUTSIDE LAYERS
FOR STANDARD TUBE—57" CIRCLE OF REFERENCE
R.P.M = 35·1073 √SINE φ

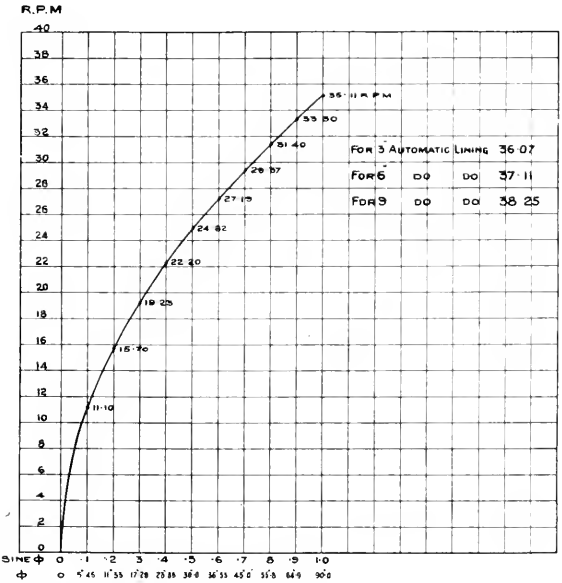


TABLE II.

Variation in Power with Load Constant at Various Speeds.

I.—Diameter inside concrete lining beginning 58 in., end 59 in.
Pebble load 2 in. above centre level—weight 1,400 lb.

II.—Diameter inside concrete lining beginning 61 in., end 62 in.

Pebble load 12½ in. above centre level —weight 2,520 lb.

III.—Diameter inside concrete lining beginning 62 in., end 62·6 in.

Pebble load 7 in. below centre level—weight 1,300 lb.

I.		II		III.	
R.P.M.	Nett H.P.	R.P.M.	Nett H.P.	R.P.M.	Nett H.P.
21·0	3·79	24·6	2·57	23·2	4·51
				24·6	4·67
25·0	4·23	25·7	2·67		
26·0	4·24			26·0	4·78
26·5	4·40	26·5	2·83		
27·0	4·46	27·4	3·40	27·4	4·93
27·7	4·56				
28·2	4·51	28·0	3·32	28·0	5·01
29·0	4·46	29·2	3·64	29·0	5·01
29·5	4·66				
30·0	4·66	30·0	3·80		
30·5	4·66				
31·2	4·98	31·2	3·97	31·0	4·55
32·0	4·98	32·2	3·59		
32·4	4·61				
33·7	4·41	33·0	3·32		
34·7	4·22	34·5	3·03	35·0	2·71
		35·5	2·63		
37·7	2·51	37·0	2·41	37·0	2·67
		38·2	2·23		
39·1	1·91	39·2	1·93	39·0	2·03
				39·7	0·58
40·5	1·56	41·0	1·81		
42·6	1·23	43·7	1·37	42·5	0·29
46·2	0·58	46·2	0·67	47·2	0·19
		47·2	0·39	49·2	0·09

The above are selected from the very numerous experiments in order to show the various effects at different speeds with normal, under, and overloading of tube. Diagrams I. and II. show graphically the results from seven other sets of experiments.

II. (b) *The effect of varying loads of pebbles.* The subjoined table III. gives the maximum H.P. observed with the different pebble loads tried, and the number of revolutions per minute at which that H.P. was required. To make this speed comparable for the different diameters of tube employed, the "angle of departure" of the outer layer of pebbles is also given, as this would be approximately the same, under similar conditions, whatever the diameter of the tube mill might be. The sine of this angle is also given, as this function depends directly upon the number of revolutions per minute.

It is convenient to express the height of the pebble load in the tube mill by the distance in inches of its surface when at rest above (+) or below (−) the axis of the tube mill.

It will be observed from the figures given that the most suitable speed (and this is conveniently ascertained from the pebble lifting H.P. quoted), depends to a certain small degree upon the load used when the diameter remains the same. If the load is increased the speed must also be slightly increased to get the best effect; this is accompanied by the spreading out of the layers across the tube, thus giving more room for the fall of a large quantity of pebbles, though a sacrifice in height of fall of the outer layers also occurs. It will be noted that the load 0 in., that is when the tube is half full, gives the maximum power consumption in each case. This is the effect of the compromise between forming more layers of pebbles and maintaining a high average fall for all pebbles present.

In practice, therefore, if the full load (0 in.) cannot be used (because of power available or other reasons), it would in some cases be advisable to reduce the speed, especially for coarse battery crushing where the gain in average fall per cycle of each pebble might be more advantageous than a greater number of pebbles with a correspondingly smaller fall. The problem resembles the adjustment in batteries by reducing the number of drops per minute and increasing the fall in accordance with the product to be crushed.

Upon comparing the angle of departure for the varying diameters of tube used, it will be seen that the smaller tubes require to be driven at a greater speed than that which would correspond with the speed required in the larger tubes; except in the case of the bar liner. This effect is most probably due to slip, which prevents the lighter loads in the smaller tubes being carried up to the theoretical angle corresponding with the speed. The much rougher working surface shown by the ribbed liner counteracts this slip in much the same way as the heavier loads in the larger tubes. In practice the percentage of moisture present may exercise similar effects besides affecting the amount of adhering sand carried up by the pebbles, but in any case it seems probable that a smooth liner would require

TABLE III.

R.P.M. Corresponding with Maximum H.P. for Various Loads.

Diameter	Load.	Nett H.P.	R.P.M.	Angle.	Sine.	Liner.
in.	in.			deg. ft.		
75	−16	5·87	25·4	43 32	·68878	Unlined
,,	− 8	8·14	25·6	44 24	·69966	,,
,,	0	10·78	25·5	43 58	·69419	,,
,,	−3·5	10·63	25·2	42 47	·67795	,,
,,	−14	8·85	24·5	39 50	·64078	,,
66	−12	5·23	28·5	49 44	·76304	Concrete
,,	+5·5	6·51	28·0	47 26	·73649	,,
,,	0	7·30	27·3	44 26	·70008	,,
,,	− 2	6·73	27·0	43 13	·68476	,,
65	−17	4·42	28·2	47 22	·73570	Concrete
64	+ 6	6·01	28·8	49 4	·75547	Concrete
,,	−2·5	6·81	27·6	43 37	·69882	Steel bar ribbed
,,	0	7·25	27·0	41 37	·66414	,,
,,	− 5	6·08	26·4	39 25	·63495	,,
62	0	6·39	28·0	43 47	·69193	Concrete
60	− 7	5·04	29·0	45 55	·71833	,,
59	−12	3·97	31·2	55 1	·81915	,,
58	−10	4·44	31·7	56 3	·82953	,,
57	+ 4	4·79	30·7	49 53	·76455	,,
56	− 2	4·98	31·6	52 45	·79600	,,
53	− 7	4·94	33·0	55 14	·82148	,,
,,	− 5	4·80	32·7	50 24	·77051	,,

a greater speed of revolution than is the case with the iron bar ribbed liner, all other things being equal. Subject to slight modifications in accordance with size of pebbles employed and coarseness of primary crushing, it may be deduced that the most advantageous way to run the standard tube (5½ ft. by 22 ft. shell) would be (using silex liners) 31 revolutions per minute with a pebble load up to the centre. If bar ribbed liners are used, the speed would be about 30 revolutions. It is obvious that if thick silex liners are employed, the tube should be driven two or three revolutions faster when a new liner is put in, and this would to some extent avoid the very poor crushing shown in this case, due to the three-fold disadvantages—smaller effectual tube, greater slip, and unsuitable speed.

The visual observations made lent no support to the view that any significant portion of the working effect is due to grinding between the pebbles, though the slight increase of −90 product obtained when the tube is overloaded is probably made in this manner at a great waste of power. It is obvious that the greater portion of the work is done by impact of the falling pebbles, either directly on the liner, or on pebbles lying on the liner. This being the case, it will obviously be an improvement if the trunnion outlet or discharge screen is made of sufficient diameter to keep down the level of water or fluid pulp to the point that practice may prove most efficient, and the observations also indicate that such level would depend to some extent upon the loading of pebbles employed, in the sense that the smaller loads would require the lower effective discharge levels.

The diameter given is the inside measurement of tube, less diameter of pebbles employed.

The angle refers to angular distance from horizontal diameter at which the outer layer of pebbles leaves the tube lining, and the sine of this angle is given in next column.

In order to further elucidate the facts observed, the subjoined Table IV. and Diagrams IV.a and IV.b were prepared, based entirely upon the theory of the tube mill. This theory is of course based upon the fact that in a moving tube the pebbles begin to fall when the radial component of their weight is equal to the centrifugal force. The observations made show no serious discrepancy with the calculations, except in so far as the wider deductions have to be controlled by the facts observed, and allowance made for the factors which cause complications beyond those it is possible to deal with mathematically.

When due allowance is made, it will be found that the table and diagrams appended agree fairly with the measurements, and in addition give information of actual drop of the pebbles, which it is hardly possible to obtain from the running tube, but which is important in designing tubes to meet any given conditions.

The importance of adjusting the speed to the load and *vice-versa* is clearly shown, and if similar calculations were made for larger pebbles, it would be shown that a small increase of speed should accompany increased pebble diameter in every case.

This table (IV.) illustrates the observed fact that between the loads of 1 in. below the centre and 4 in. above it, there is no very great change in power consumed and probably very little in tons of −90 produced per H.P., provided that the speed is varied accordingly. The given speed, 29·52, corresponds with a maximum of fall in the outer layer which strikes the lining direct (as shown in Diagram IV.a), and would therefore represent the speed at which the wear of the liner becomes greatest.

The point of maximum capacity thus calculated is not identical with the point of greatest efficiency, but it is probably the most important for practical use.

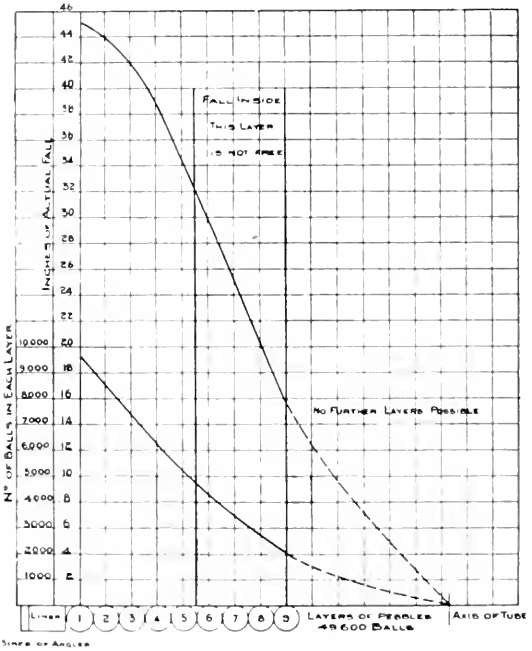
The diagrams (IV.a and IV.b) also illustrate the effect of the greater speed in producing space for an increased number of pebble layers, and also the partly compensating disadvantage of reduced fall in each layer thereby caused. The number of layers where an unpeded fall only is possible depends upon the ratio between pebble diameter and diameter of tube, thus showing advantage of larger tubes, especially when coarse crushing precludes the feeding of small pebbles.

It may be pointed out that the number of cycles per revolution for each layer of pebbles depends only upon the "angle of departure" for that layer, and is not directly affected by the diameter of the tube or the number of revolutions per minute. The pebbles of the outer layer in a standard tube mill perform their journey about 1·2 times, and those of the innermost layer about 1·8 times while the tube goes round once, owing to the shortened journey and their greater average speed while falling through the air.

It will be observed from Diagrams IV.a and IV.b that the height of fall is different for each layer of pebbles, and is less to-

DIAGRAM IV *a*

— HEIGHT OF FALL PER CYCLE CORRESPONDING WITH EACH LAYER —
— IN STANDARD TUBE 5·6 × 22·0 WITH 3½ LINERS, 2 PEBBLES $\frac{\text{R.P.M. } 29{\cdot}52}{\text{SIN } 45^\circ 707}$ —
— AVERAGE ACTUAL FALL PER BALL PER MINUTE = 145·9 FEET —
— Nº OF BALLS IN EACH LAYER SHEWN IN LOWER CURVE —



wards the centre of the tube, and that the number of layers which can be given a free fall is limited, depending partly upon the number of revolutions per minute.

The maximum referred to in Diagram IV.b differs from that calculated in Table IV., the former refers to total possible fall and the latter to free fall only.

All other conditions being equal, the power required to drive tubes of different sizes will depend upon the fifth power of the square root of the diameter multiplied into the length.

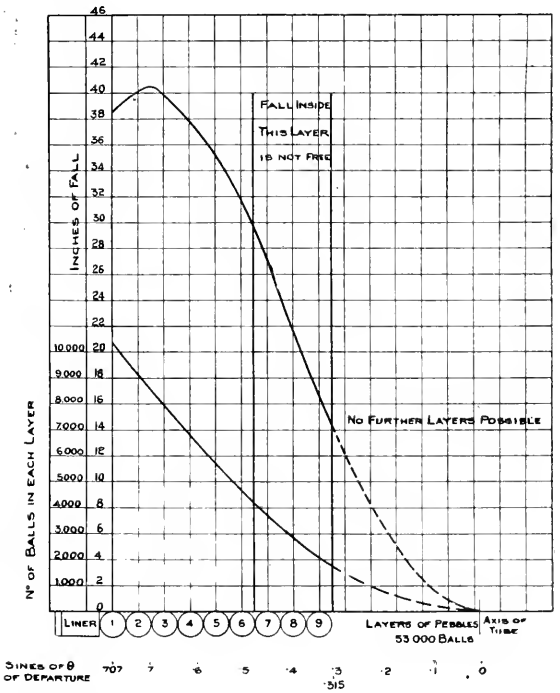
TABLE IV

Theoretical Maxima of Pebble Lifting Horse-power for Standard Tube Mill with Various Loads.

Tube mill diameter inside lining, 59 in

DIAGRAM IV*b*

HEIGHT OF FALL PER CYCLE FOR EACH LAYER OF STANDARD TUBE 5·6 x 22·0 WITH 3½ LINERS, 2 PEBBLES, R.P.M = 31·17 SIN θ 51° 55 = 787183, GIVING MAXIMUM TOTAL FALL x LOAD PER MINUTE



Pebbles assumed to be 2 in. equivalent diameter.

Load of pebbles when tube full, 21·92 tons.

Taking the speeds given, the total loads possible at that speed which will allow *free* fall to all the pebbles is calculated; next the free fall per cycle for the average peb-

bles is obtained: this multiplied by the average number of cycles per revolution gives the average fall per revolution: from this the H.P. is readily ascertained.

R.P.M.	Pebble Load.	Average Fall per Cycle.	Average Fall per Revolution.	Pebble-Lifting H.P.
	in.	ft.	ft.	
26·38	15	3·96	6·16	41·70
28·18	8	3·75	5·46	69·97
28·44	7	3·73	5·36	72·51
29·52	4	3·57	4·91	81·61
29·74	3	3·48	4·78	81·76
30·43	1	3·42	4·58	87·29
31·07	0	3·30	4·53	90·09
32·20	**- 3**	**3·03**	**3·82**	**91·43**
32·67	+ 1	2·90	3·54	89·22
33·42	+ 5	2·52	3·05	83·50
34·50	+ 7	2·32	2·73	80·88
35·11	- 8	2·07	2·39	75·93

In actual tube mill operation additional H.P. would be required to overcome friction outside the tube, inside pebble friction, and any power consumed in lifting up portion of the pulp feed (dependent principally upon moisture and feed tonnage). Any overloading with pebbles beyond the figures given for each speed will at first result in increase of H.P. (principally wasted in inner space where no free fall can be obtained), and subsequently the power will fall off owing to reduced fall allowed by the excess pebbles.

H. (c.) *Discussion on Pebble Working Level.* In practice the pebbles in a running tube are so arranged that the point of impact is everywhere below the axis of the tube. Observations with the experimental tube show that the level, below which all the pebbles have completed their fall, depends upon the load of pebbles and upon the relation between the number of revolutions per minute and the square of the tube diameter.

The level so defined is referred to as the working level; and it is clear that if water or fluid pulp remains in the tube above this level, a certain portion of the effect of the falling pebble will be lost and power thereby wasted. The formation of this level is the only effect that could be dealt with experimentally, but the application of the results will be modified in practice, partly in accordance with the percentage of moisture present and partly owing to the useful effect of the fluid in removing from the tube the particles sufficiently ground. Part compensation is caused by the reduction in effective weight of lifted pebbles owing to their submergence.

Observation shows that this level is not cleanly defined, owing to the fact that the pebbles do not fall dead but rebound, either from the lining direct or from other pebbles lying thereon, and that the theoretical curve joining points of impact is materially modified by this effect and by the tendency of the fallen pebbles to heap up to the limit of the angle of repose. In every case the effect of loading pebbles beyond the tube centre is to raise the working level, and this explains the customary 3 in. above the axis, which is the loading used with a high level pulp discharge, the gain being due to saved splash energy, as so large a load cannot be maintained in free-falling flight at the usual speeds.

The important figure to be measured is the effective diameter of the discharge screen which will keep the fluid down to this working level, but it is to be noted that in practice the diameter most suitable may be less than either the theoretical figures or those deduced from actual measurements when pebbles only are present.

Table V. shows theoretical diameters (1) for outside layers of pebbles, (2) for lowest falling pebbles. (This diameter is not directly dependent upon diameter of tube but only upon number of revolutions per minute, and is associated with that inner layer whose "angle of departure" is 37° 15'.) Table VI. gives actual measurements of observed working levels, but all the figures in these tables are expressed in terms of the diameter of discharge necessary to lower the fluid outflow to the working level observed.

TABLE V.

Theoretical Discharge Diameters.

Standard tube mill, 59½ in. inside lining, 2½ in. pebbles.

R.P.M.	(1). Discharge Diameter (outer pebbles)	R.P.M. (any tube).	(2). Discharge Diameter (lowest pebbles)
	in.		in.
24·9	56	25	62
		26	57
		27	53
28·2	48	28	49
		29	45
		30	43
31·0	27	31	39
32·4	16	32	37
32·6	0	33	35
		34	32

All the above figures assume pebble loading suitable to speeds quoted; any overloading will reduce the diameter of discharge screen required, but the exact relationship is too complicated for calculation.

It is also assumed that area of screen openings is sufficient to prevent backing up of the pulp inside the tube.

TABLE VI.

Observed Discharge Diameters.

Diam. of Tube (inside lining).	R.P.M.	Pebble Load.	Diameter of Discharge.
in.		in.	in.
62	23·2	− 7	42
,,	24·6	− 7	42
,,	27·4	− 7	44
,,	31·0	− 7	40
,,	35·0	− 7	36
66	23·5	− 3	56
,,	25·5	− 3	50
,,	28·5	− 3	44
66	25·5	0	39
,,	28·8	0	36
63	22·2	0	36
,,	26·5	0	33
66	25·6	+ 3	43
,,	28·5	+ 3	36
60	24·2	+ 10	14
,,	26·0	+ 10	15
,,	27·2	+ 10	15
,,	28·7	+ 10	17
,,	31·2	+ 10	20
,,	32·7	+ 10	21
61	24·6	+ 12	14
,,	26·5	+ 12	15
,,	29·2	+ 12	17
,,	32·2	+ 12	22
68	30·0	+ 14	26
,,	30·2	+ 19	20

The foregoing figures clearly demonstrate that the effect of higher speed of revolution is to decrease the diameter of discharge outlet required, unless loads are very excessive, but that the contrary effect is obtained by increasing the loading of the tube. In any practical experiments upon variations of screen discharge, by means of scoops or in any other manner, these facts must be carefully borne in mind or the results may only lead to confusion.

II. (d.) *Size of Tube Mill Pebbles.*—The experiments made included all sizes of tube mill pebbles between 1 in. and 4 in. As theory and observation alike indicate that the vertical height of fall in a standard tube-mill under normal conditions will vary for different layers between the limits of 10 in. and 14 in., it was decided to determine what size of banket particles could be broken with pebbles of various dimensions at ascertained heights of fall. Table VII. gives particulars of the results obtained from which the following conclusions may be drawn:—

1. The weight of the pebble and height of fall are mutually dependent factors, so that a given piece of banket may be broken by either increasing the weight of the pebble or the height of the fall thereof. It follows that tube mills of greater diameter are likely to be suitable with very coarse battery crushing, and that the only limit to successful tube milling of even products taken direct from the rock-breakers is the mechanical one of size of tube mills and the provision of large enough pieces of banket to operate in them. By calculation (from the observed figures) it may be concluded that in an 8 ft. diameter tube mill, 2 in. cubes of banket would easily be broken up by using 8 in. pebbles.
2. It may be deduced from the figures that ½ in. pebbles may be retained within the tube if no coarser screen is used in the battery than 9 mesh, but that when ½ in. or 1 in. screening is used in the battery the tube mill discharge should be arranged to eliminate all pebbles under 1 in., and might preferably be slotted so that flat pebbles with that diameter would be rejected.
3. It would seem that 2 in. or 2½ in. pebbles are large enough if the battery screening is no coarser than 9 mesh and that 3 in. to 3½ in. pebbles are required if very coarse crushing up to 1 in. screens in the battery is employed. No obvious advantage is apparent in using larger pebbles than those quoted, except to allow for wear by abrasion.

TABLE VII.

Height of Fall of Pebbles to Break Particles of Banket.

Weight of Pebble Used.	Diameter of Equivalent Sphere (Assuming sp. gr. 2·7).
28·2 grammes	1·0676 inches.
220 ,,	2·118 ,,
645 ,,	3·032 ,,
1157 ,,	3·682 ,,

concrete floor and (*b*) upon a steel plate 1¼ in. thick. The pebbles rebounded vigorously from the concrete floor, but none were broken. Upon the steel plate all tried were broken by the fall into two, three or more large pieces.

II. (c.) *Coarseness of Feed.*—With the standard 5 ft. 6 in. x 22 ft. tube at the Princess Mill, battery screen with 1 in. aperture have been successfully employed, though the pebble chips discharged increased very considerably. This difficulty was not insuperable. With battery screens of ½ in. apertures much less difficulty is met

Size of Particle to be Broken.	Height of Fall.			
	1·0676 in. pebble.	2·118 in. pebble.	3·032 in. pebble.	3·682 in. pebble.
− ½ in. + ⅜ in.	42 in. drop does *not* break particle.	19·5 in.	7·0 in.	4·0 in.
− ⅜ in. + ¼ in.	Do.	9·0 in.	3·2 in.	1·8 in.
− ¼ in. + 5 mesh (I.M.M.) *i.e.*, − 0·25 in. + 0·1 in.	11·5 in.	1·5 in.	0·5 in.	0·3 in.
5 mesh + 8 mesh *i.e.*, 0·1 in. + 0·0625 in.	1·5 in.	0·2 in.	Very small drop breaks particle.	
− 8 mesh + 10 mesh *i.e.*, − 0·0625 in. + 0·05 in.	0·4 in.	Very small drop breaks particle.		
− 10 mesh + 16 mesh *i.e.*, − 0·05 in. + 0·03125 in.	0·2 in.	Very small drop or continued rolling breaks particle.	Rolling breaks particle.	Particle cannot support weight of pebble.

The height of fall is the distance measured between bottom of falling pebble and top of particle to be broken. Chert is the most difficult constituent of the banket to break, mixed quartzite next, and pure quartz the least difficult.

The − ½ in. + ⅜ in. material was practically all quartzite; the − ⅜ in. + ¼ in. had less quartzite, and the − 0·25 in. + 0·1 in. was a mixture of quartzite and quartz in nearly equal amounts. In this case the drop given is that required to break the quartzite. In the remaining cases quartz predominates, and the drop given is that required to break the quartz. In view of the variation in material, no attempt is made to deduce any law of crushing from these figures. The flint pebbles used were subsequently dropped 70 ft. on upon a concrete floor and (*b*) upon a steel plate 1¼ in. thick. The pebbles rebounded vigorously from the concrete floor, but none were broken. Upon the steel plate all tried were broken by the fall into two, three or more large pieces.

II. (c.) *Coarseness of Feed.*—With the standard 5 ft. 6 in. x 22 ft. tube at the Princess Mill, battery screen with 1 in. aperture have been successfully employed, though the pebble chips discharged increased very considerably. This difficulty was not insuperable. With battery screens of ½ in. apertures much less difficulty is met with in this direction, as using ¾ in. holes inside tube discharge screen it is possible to do without the pebble eliminator entirely and to run the entire discharge over the plates where the rounded nature of the escaping pebble chips prevents much interference with amalgamation as the high extraction (75 %) sufficiently demonstrates. Other mines have used or are using ½ in. screens in the battery, but with larger diameter tubes; with the standard tube it must be admitted that the case of working is very considerably increased if the battery screens are reduced to ¾ in. aperture. The tube discharge then contains much less very coarse material, and the duty is not very materially decreased (about 3%). It thus appears that material coarser than ¾ in. cubes may to some extent escape frac-

ture, and is only gradually worn down to the point where it is certain to be broken and thus released from circuit. Of course, with such feed to the tubes the necessity of keeping up the size of rock fed for pebbles becomes of greater importance.

II. (f.) *Amount of Feed.*– Numerous experiments have demonstrated that the standard tube requires more than 400 tons of sand per diem to produce the greatest possible quantity of −90 product. The maximum limit has not been determined, and special arrangements will be required to enable much more than 500 tons per diem to be passed through, though the experiment is apparently worth trying. The best feed of course depends upon the length of the tube, as the pebbles right up to the outlet must have sufficient material to work on.

As improvement is shown by eliminating from the feed all material already fine enough, it is clear that the real limit to the length of the tube is reached when the effect of the already reduced material becomes excessive, and this effect is probably present when the discharged pulp shows more than a half to be of the required size. If therefore the most effective discharge level is ascertained, then the amount of feed it is possible to pass through satisfactorily determined, the correct length of the tube would probably be somewhere near the point at which one-half of the feed has been reduced to the desired size.

II. (g.) *Moisture in Feed.*—Results of experiments upon the effect of different amount of moisture in the feed with varying levels of discharge have not yet been published. Though it appears theoretically that more water might well be used as the effective diameter of discharge screen is increased, it is probable that the determination of this point will rest upon a compromise between the various effects produced. Taking 38 to 40 of water as the most favourable with practically central discharge, a decrease in moisture must increase power required, as the pebbles will lift up an increased amount of adhering pulp. This is partly set off as the increased gravity of the pulp will reduce the effective weight of the pebble being lifted through that portion of its path which is below the pulp level; on the other hand, increased friction would tend to increase the power consumed. With a low level discharge, increased water feed will facilitate the removal of sufficiently fine material without reducing the effect of the blow of the falling pebble by increasing the depth of mud it must splash through.

II. (h.) *Fluctuations in Power and Speed.*

A paper has recently been published showing great but regular variations in power consumed during each revolution of a standard tube, and it is a notorious fact that the ammeter indications with an ordinary tube mill are continually fluctuating to a material extent. It was therefore considered advisable that this matter should be thoroughly investigated, and the circumstances and causes of these variations ascertained. It had already been observed and noted that with certain loads and at certain speeds of revolution there was a marked visible fluctuation in the cascade of falling pebbles accompanied by a distinctly audible sound, so that the number of "drops" per minute could be easily counted.

The observations under this head are collected in the following tables:—

TABLE VIII.

Fluctuations in Fall of Pebbles, R.P.M. Steady.

Diameter inside lining.	R.P.M.	Load.	"Drops" per minute	
in.		in.		
55	38·0 —39·0	+ 5	50	(audible).
56	42·5—48·5	+ 7	53	(audible).
59	37·7—42·6	+ 2	44	(audible).
,,	38·0—46·2	+ 4	48	(audible).
60	41·7	+10	52	(just audible).
61	44·0	+12	48	(just audible).
62	23·2—35·0	- 7	47	(distinct at the higher speed).
64	31·2—38·0	0	40	(pronounced).
,,	26·5	0	50½	(very feeble).
66	31·5—34·0	0	42	(pronounced).
,,	29·0—30·2	0	42	(very feeble).
,,	30·7—32·2	+ 2·5	42	(feeble).
,,	29·0—34·2	- 3·5	39½	(distinct).
67	34·5—39·2	+ 6	42	(visible and audible).
68	40·0	+12	—	(pronounced & irregular).
77	27·0 31·8	−14	59	(very pronounced).
,,	26·6—35·0	- 3·5	38	(distinct).
,,	30·7—33·0	0	38½	(distinct).

The above table clearly shows that the load must be too light (below −5 in.), or the speed excessive for the tube diameter

before this phenomenon becomes of importance. *It is to be noted that the number of drops does not change at all with the speed* while the load and tube diameter are kept constant.

By the courtesy of the Electrical Department of the South African School of Mines a Duddell oscillograph was fitted up to show the variation in current consumed during a revolution of the tube. A load 8½ in. below the centre (−8½ in.), driven at 32·5 revolutions per minute, showed violent fluctuations, 39 cycles per minute corresponding with the drops audible and visible. Every fourth wave was of different character, showing less variation, but the other waves dropped below the zero line of current.

An oscillograph record with the tube running normally, showing no "drops," gave, however, a constantly fluctuating current line, but the extreme variation was small. A further record with the tube running empty showed an exactly similar line, so that there can be no doubt that the cause of fluctuation in this case was merely mechanical, due to unbalanced tube construction, variation or irregularities in gears, or in belt drive. An oscillograph with the motor running light showed some small fluctuation.

An electrically vibrated tuning fork marking smoked paper wrapped round the tube mill shaft was fitted up in order to show variations in speed, which might be expected to accompany considerable power fluctuations in such an apparatus; and experiments were made in this way with the tube running normally and when a well marked "drop" was experienced. The results, exhibited in the following Table IX. show no variation in speed, beyond experimental error, except in the case where marked "dropping" was observed. In each case the tracing for one revolution was divided into a number of equal parts, the number of waves counted, and the variation in speed was recorded in percentages of mean speed.

In this last case the maximum variation in speed was 7.5 , but the power variation was much greater. The conclusion arrived at is that the excessive fluctuation is either due to the absence of a sufficient weight of pebbles originally, or to the bulk being taken up on the circumference by driving beyond the critical speed (angle of departure 90° or more). Observation shows that a certain weight of pebbles must always be on the circumference, or merely slipping

TABLE IX.

Variation in Speed during one Revolution of Tube.

(1) Tube diameter inside (ribbed) lining, 66 in.

Load −3 in. R.P.M. 29.		Load Central R.P.M. 28·2.
Portion of Revolution.	Per Cent. of Mean Speed.	Per Cent. of Mean Speed.
1	100·19	99·7
2	100·19	101·2
3	98·60	99·7
4	101·70	101·2
5	100·19	99·7
6	100·19	99·7
7	100·19	99·7

(2) Tube diameter inside (ribbed) lining, 66 in.

Load −3 in. R.P.M. 28·7.		Load −3 in. R.P.M. 29·0.
Portion of Revolution.	Per Cent. of Mean Speed.	Per Cent. of Mean Speed.
1	100·83	100·82
2	99·17	99·18
3	99·17	99·18
4	99·17	99·18
5	100·83	100·82
6	100·83	100·82
7	99·17	100·82
8	100·83	99·18

(3) Tube diameter inside (concrete) lining, 69 in.

Load −8·5 in. R.P.M. 32·5.
"Drops" 39 per minute.

Portion of Revolution.	Per Cent. of Mean Speed.
1	98·8
2	104·8
3	101·8
4	101·2
5	98·8
6	97·3
7	97·3

takes place until enough pebbles have fallen from the air to give enough "grip" on the liner. This recurs periodically, and if insufficient pebbles are present to keep up the cycles in the layers formed, another period of slipping takes place till all the pebbles have again fallen, when they are again carried up and so on alternately. The period depends upon cycles per revolution and weight of pebbles present.

Besides the fluctuation in ammeter readings in ordinary practice which may be due to irregularities in belt and gear, two other sources have to be considered:—

1. In the first place the lining of the tube does not wear with perfect evenness, and this may be exaggerated by the not uncommonly observed lack of truth in the tube cylinder employed. Perfect running cannot be secured unless the tube, lining, and trunnions are practically co-axial.
2. The fluctuations in voltage of the electric power supply.

NOTE ON A METHOD OF VENTILATING SINKING SHAFTS, WINZES AND DRIVES

By S. Nettleton (Member).

Of a total of 223 deaths arising from the use of explosives in the mines of the Witwatersrand during the year 1913, there were 51 due to fumes, and in addition to the loss of life there must have been considerable loss of efficiency amongst workers who were effected by slight and unrecorded cases of gassing.

Apart from consideration of the health of white and native employees, which should be a matter of first importance, ventilation has a direct bearing on financial interests. In many instances development work is quite unremunerative until completion, and to reduce amount of interest on large capital expenditure it is desirable to reduce to a minimum the time wasted owing to operations not being actually in progress at the working faces.

The chief loss of time arises in the removal of dust and fumes from explosives, and various methods are employed to render shafts and development drives fit for the re-entrance of employees. The most simple and probably the most common arrangement is to introduce fresh air from the compressed air service, with the addition of a water blast to precipitate the dust. Compressed air is, however, a costly commodity, and though the pressure at which it is delivered may be high, the volume is rarely sufficient to entirely replace the foul atmosphere of mine workings in a short time. To introduce larger volumes of pure air there have been employed wooden and light metal pipes going direct to the working face.

The initial cost of these arrangements is high, and they are liable to frequent derangement by the shock of blasting operations. Even with constant attention to joints, leakage of air is apt to be excessive, and in mains 1,000 ft. in length not more than 30 or 40 of the air entering the pipe may be delivered at the place where it is required.

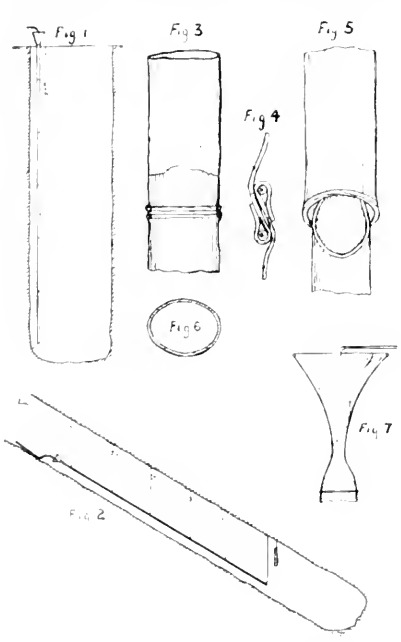
For the ventilation of a "dead end," such as exists in a sinking shaft or development drive, the author has devised an arrangement which obviates these objections, and which consists in the employment of conduits of canvas brattice cloth or other textile material, made up into convenient lengths and provided with attachments which permit of simple connection and disconnection. The nature of the conduit tends to ease of transport and erection underground. The flexibility of the canvas permits of slight adjustments in passing obstacles in the shaft or drive which is being ventilated, and its weight is such that it may be suspended in any position without the provision of heavy and costly attachments.

The initial cost of the arrangement is small compared with thin metal pipes or wooden boxing, and it is possible to use them on more than one occasion. In practice it is usually found that when metal pipes have been once used for ventilation purposes, they are almost useless for employment in another part of the mine. Even with great care in removal it is practically impossible to make them into an air-tight main on a second service.

Fig. 1 shows an application of the system during the sinking of a vertical shaft. The pipes are distended by the joint rings, their own weight, and the pressure of air in the interior. They are suspended from the surface and at convenient intervals by hangers fixed on the side of the shaft. Immediately

previous to blasting the lower portion of the conduit is drawn into the position shown in dotted lines by a light cord which extends to the surface. Removal from the zone of blasting obviates any risk of damage to the pipes. After blasting the pipes are lowered to the bottom of the shaft, which receives without delay a supply of fresh air from a fan or some form of injector placed at the top of the shaft.

For an inclined shaft the air conduit is suspended from the roof by wires or light rope as shown in Fig. 2. The pipes in this



instance are protected from injury by a batter board of rough timber swung from a couple of hooks fixed in the roof at a convenient distance from the face of the incline. When blasting takes place the conduit is temporarily drawn towards the roof of shaft as shown by dotted lines in sketch.

Figs. 3, 4, 5 and 6 explain the principles of attachment of the sections of conduit to each other. To the ends of each tube of canvas are attached oval rings all of equal size. The end of one pipe is placed in the interior of another as shown in Fig. 5, and one ring is rotated until it is exactly parallel to the adjacent ring. The pipes are then drawn out in a longitudinal direction, and it is obvious that not only is there produced an air-tight joint, but that increased tension increases the efficiency of the connection.

Fig. 7 shows an application of the principles of the Evasee chimney to conduits where the ventilating current is produced by a jet of compressed air. This increases very considerably the quantity of air introduced into the pipe, as by reduction of eddy currents it tends to approach the ideal condition in the production of a ventilating current of air, viz., "to receive without shock and discharge without velocity."

It may be mentioned that the principles of connecting air conduits and other novel features of arrangements here described form the subject of an application for patent rights made several months ago.

OBITUARY.*

SAMUEL BENEDICT CHRISTY

The death of Prof. Samuel Benedict Christy on the 30th of November, at the age of 61 years, cuts short a brilliant and influential professional career. He was born in San Francisco, August 8, 1853, and graduated as Bachelor of Philosophy at the University of California in 1874. For the five years that followed, he studied mining and metallurgy as a post graduate in the same institution, serving also as instructor in analytical chemistry, and becoming in 1879 instructor in mining and metallurgy—a position which he held until 1885. During this period he became known also by his contributions to technical literature.

But his great work was the creation of the school of mining and metallurgy at the University of California. It is not necessary here to recount the early difficulties and perils of the University, and its long struggle for academic independence of political control or interference. The battle is over, and the institution is now upheld by the intelligent loyalty of the citizens of California, without regard to party. But while the issue was

* Abstracted from the *Engineering and Mining Journal* for October 19, 1914.

uncertain, the development of special schools of applied science was scarcely practicable. Doctor Becker and William Ashburner had given up the school of mines as a hopeless task when Christy, younger and less experienced, took it up. As he wrote in February, 1901—

> "I took an empty building without equipment, and with half a dozen students. Now I have a fully equipped department, adapted to our local needs, better than any I have seen in the world. I have now nearly 250 students; and I am willing to compare the record of our graduates, now scattered all over the world, with that of any other institution of equal age."

The secret of this success lay in the enthusiasm, pertinacity, industry and personal charm of Christy himself. He was able not only to inspire his students, but to impress upon their parents and the public at large the value of the thorough training given them under his direction. His own example was contagious. Besides the practical labours of his office, he prosecuted laboratory researches of the most delicate and complicated character, in which it was a privilege for any student to be his assistant.

In 1900, as the result of patient and thorough experiment, he patented an improved process for the recovery of gold and silver from dilute cyanide solutions. And to the *Transactions* of the American Institute of Mining Engineers (of which he became a member in 1883, and was a vice-president in 1891 and 1892) he contributed a series of highly original and important papers on the metallurgy of gold, silver and mercury, and on the training of mining engineers and the relation of American mining schools to the mining industry, as well as a remarkable biographical notice of Joseph Le Conte, his friend and colleague, which has been recognized as the most appreciative and satisfactory of the tributes paid to that illustrious and beloved leader and teacher. But beyond this list, which is enough of itself to secure for Professor Christy a permanent place in the history of American scientific and technical progress, there remains his work in connection with the California Academy of Sciences (of which he was a life member, and for five years corresponding secretary), the Society for the Promotion of Engineering Education (of which he was three years vice-president), the California Miners' Association and other bodies.

Review.

THE CHEMISTRY OF CYANOGEN COMPOUNDS.

By Herbert E. Williams (*J. and A. Churchill, London.* Price 10s. 6d.)

The author, who will be favourably remembered here, has evidently intended to produce a work in English as compendious as Dammar is in German, so far as the compounds containing a cyanogen-group are concerned; and he has certainly succeeded in collating a mass of information—sometimes about the most out-of-the-way compounds—in a form exceedingly handy for reference, especially since many of these substances, though known to science, escape indexing in ordinary chemical literature.

Probably the most useful part of the book is Part III., dealing with the analytical chemistry of the subject in all its branches: this is exceptionally well done, probably because of the author's greater familiarity with the subject. On the other hand, Part I., dealing with the chemistry (descriptive and theoretical) of cyanogen compounds, is rather unsatisfactory. For one thing, it is full of misprints, as if proof-reading had been entirely omitted: it also contains statements which are surely due to ignorance or extreme carelessness, *e.g.*, the preparation of the compound C_4N_2 by heating a substance that does not contain nitrogen! (page 15, line 3 from below).

A great deal of space is occupied with the author's discovery of the presence of "combined" alkali in prussian-blue and other metallic ferrocyanides. Most chemists consider that this alkali is merely adsorbed, and that many of Mr. Williams' formulæ are much too complicated to be correct, even if they are supported by analyses. For example, on page 142 we meet with the strange compound $[Fe_{56}K_3Cy_{162}+210H_2O]$ which surpasses albumen for complexity. Surely the author might admit that the potassium in this and similar compounds is merely adventitious, just like the trace of unremovable ash in many organic substances.

The large section on manufacture is most comprehensive, and may later on prove of value to this country when it makes up its mind to be less dependent on importations than it is at present; for example, with

ammonia from coal, the process given on pp. 296-7 is quite feasible in South Africa near the coalfields. Another remark of the author's (p. 226) is pregnant: "The reason of this failure is the policy . . . . of placing difficult and intricate chemical problems in the hands of an engineer who though highly skilled . . . . is yet quite incapable of dealing with problems of this nature."

In case a second edition is called for, the following more serious mistakes should be noted for correction: (1) HCNS is not hydrothiocyanic acid but thiocyanic acid (its salts are thiocyanates, not thiocyanides: (2) the formulæ P_2Cl_3, P_2OCl_3 and Sb_2Cl_3 (all on p. 10) are wrong: (3) the word phosphorus (pp. 2 and 10) only contains two *o*'s: (1) the compounds of $(CN)_2$ and H_2S are not cyanogen sulphides (p. 12) but thio-oxamides: (5) on page 38, for aldehyd read formaldehyd, and for acetonitrite read acetonitrile: (6) on page 51 (centre), for aurocyanide, read auricyanide; and at bottom of page, for auricyanate read auricyanide; so on p. 79 read ferrocyanide for terrocyanate: (7) in three places read N_2H_4 for hydrazine, instead of N_4H_2: (8) at foot of p. 93 "the salt of iron and copper" should be "a salt of iron *or* copper". (9) on page 108, line 6, *ferric* should be ferrous: (10) on page 129, Y_2Fey should be Y_4Fey_3, and ThFey should be mentioned: (11) p. 195 for *hydroscopic* read *hygroscopic*: (12) for phthalopherone (p. 319) read *phenolphthaline* (di-oxyphenyl-o-toluic acid): (13) formula of mercury and silver acetates (p 325) are wrong: (14) the names Schleicher and Schnell are dreadfully misspelt on page 356: also Kjeldahl on p. 383: (15) the middle equation on p. 391 is ridiculous; liberation of carbon in a wet reaction!

Careless writing, bad grammar, and misspellings occur occasionally: thus on p. 208 we are told that "100 parts of water . . . . might be used as an analytical method" (!) and in several places the use of *tarred* filter-papers is recommended (*i.e.*, tarsch): the phenomenon of the *discolouration* of cyanide also occurs! On page 205, when dealing with ferric thiocyanate the author omits to mention the work of H. A. White and J. Moir in this country (whose results differed to some extent from those of Philip and Bramley quoted by the author).

Despite these criticisms the book is a good one, and will be in demand for reference libraries, serving as it does as a sort of enquire-within-upon-everything on this specialised subject.—J. M.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

WINKLER'S VOLUMETRIC DETERMINATION OF AMMONIA.—"Winkler recommends receiving the distillate containing the free NH_3 in H_3BO_3 solution and titrating back with HCl. Methyl-orange or Congo-red can serve as an indicator. According to Winkler, this method is particularly suited for Kjeldahl determinations. The method has been investigated by S. on filtered sewage. The advantages of the new method are as follows: The distilled NH_3 is titrated directly and no re-titration is required. Only one volumetric solution is employed. The standard HCl keeps for a long time, while the NaOH employed in the other method must be frequently controlled. Large quantities of NH_3 can be distilled without fear of having an insufficient amount of acid in the receiving flask."—A. SCHULZE, *Mitt. kgl. Landesanst. Wasserhyg., Heft 18*, 87-90.—*Chemical Abstracts*, Aug. 10, 1914, p. 2660. (J. G.)

STUDIES ON A NEW METHOD FOR THE QUANTITATIVE DETERMINATION OF POTASSIUM. — "After many experiments on the precipitation of K bitartrate and the contributing steps, M. has developed the following procedure: The solution must contain the alkalies as chlorides. Other bases, silicates and H_3PO_4 must be absent. An aliquot containing about 0·05 gm. K_2O is pipetted into a beaker and the solution evaporated to dryness on the water bath. Dissolve the residue in the smallest quantity of H_2O (not more than 2 or 3 drops). Keeping the solution cold, add an excess (20 cc.) of a 2% solution of tartaric acid in ethylalcohol, which has been shaken for several days with solid K bitartrate and then filtered. After precipitation add 10-20 cc. of a K bitartrate in 96% ethylalcohol and heat for 10 minutes on the water bath. Let stand with frequent shaking for 24 hours, filter through a weighed Gooch and wash with more of the ethylalcohol solution of K bitartrate. Finally wash with pure 96% ethylalcohol, dry at 80° and weigh as K bitartrate. This weight divided by 3·9992 gives the equivalent amount of K_2O. Tables of results show the method to be accurate." F. MARSHALL, *Chem. Ztg.*, *38*, 585-7, 615-6. *Chemical Abstracts*, Aug. 10, 1914, pp. 2660-1. (J. G.)

THE ESTIMATION OF CARBON DIOXIDE IN THE AIR. A SIMPLE AND EXPEDITIOUS METHOD.—"A series of 100 cc. flasks, stoppered and filled with distilled H_2O, are emptied at the place where the sample is required to be taken. The stoppers are inserted and the bottles taken to the laboratory. Into each flask a standard solution of Na_2CO_3 coloured with phenolphthalein is rapidly run, in arithmetically progressive quantities until a limit is reached, beyond which it is not necessary to go. For instance, in the first flask 1 cc. are placed, in the second 5 cc., etc. Two or three flasks should be reserved for final adjustment if necessary. The standard solution of Na_2CO_3 is of such a strength that 1 cc. = 0·01 cc. of CO_2 (N.T.P.). It is made by dissolving 0·4748 gm. of Na_2CO_3 in 100 cc. H_2O and after standardizing with HCl this solution is diluted 100 times. The flasks containing the air to be tested and the Na_2CO_3 solution are well and continuously shaken for 20 minutes. The amount of CO_2 expressed in percentage by volume is readily indicated by the number of

cubic centimetres of standard solution decolorized. A reserved 100 cc. sample can finally be tested with 0·5 cc. more of the standard solution than was required by the decolorized flask. A correction for the actual volume in the flask is made, subtracting the volume of the solution added."—W. M. DOHERTY, *Chem. News*, *109*, 281-2.—*Chemical Abstracts*, Sept. 10, 8914, p. 2991. (J. G.)

DETERMINATION OF HYDROCYANIC ACID AND ALKALI CYANIDES.—"A new method for the determination of hydrocyanic acid and the alkaline cyanides has been worked out by the authors. Briefly, it consists of titrating an ammoniacal cyanide solution containing a small quantity of dimethylglyoxime with a standard nickel-ammonium sulphate solution until a permanent red precipitate is produced. The reactions involved are:

$$NiSO_4 + 4KCN = K_2Ni(CN)_4 + K_2SO_4$$

$$NiSO_4 + 2C_4H_8N_2O_2 = Ni(C_4H_7N_2O_2)_2 + H_2SO_4$$

No permanent red precipitate is produced until all the cyanide has been used up. An ammoniacal solution is used since free sulphuric acid prevents the precipitation of nickel dimethylglyoxime.

The standard nickel solution consists of 15·3 gr. of nickel-ammonium sulphate dissolved in 1 litre of water containing 2 cc. strong sulphuric acid. The dimethylglyoxime solution consists of 8·9 grams dimethylglyoxime in 1 litre of 95% alcohol.

Standardize the nickel solution thus: 25 cc. portions are diluted with distilled water to 200 cc., treated with 0·2 gm. tartaric acid, and heated to boiling. Glyoxime solution sufficient to precipitate all the nickel is then added. If the glyoxime solution has been made as above directed, 30 cc. should be enough. After adding the glyoxime, make slightly alkaline with ammonia, boil for 2 minutes and then set aside to digest for half an hour. Filter on a tared gooch, wash with 200 cc. hot water, dry for 45 minutes at 120° C. and weigh. The weighed precipitate contains 20·31% Ni. From the equations given above the CN titer of the solutions can be calculated. If a chemically pure potassium cyanide is at hand, the titer can be determined directly by titrating weighed portions as directed below.

Five grams of the sample are dissolved in water and diluted to 500 cc. Pipetted 50 cc. portions of this solution are diluted with water to 100 cc. and treated with 1 cc. of ammonium hydrate and 0·5 cc. of dimethylglyoxime solution and then titrated with the standard nickel solution until a permanent red precipitate is produced.

The colour play toward the end of the reaction resembles the methyl orange end point observed in titrating an alkaline solution with an acid solution. The nickel-ammonium sulphate solution may be added rapidly at first, provided the cyanide solution is vigorously stirred; toward the end point the addition should be slower and the stirring more rapid. If more than 0·5 cc. of glyoxime is used, the end point shows a tendency to appear too soon unless the addition of the standard solution is slow and the agitation of the solution very brisk. The cyanide dilution may be varied without serious effect; however the method works better when the volume is approximately 100 cc.

A large excess of ammonium hydroxide delays the end point; the 1 cc. to 5 cc. in volume specified does no harm. A titration requiring 50 cc. of the standard solution costs one-fifth of a cent. In titrating solutions which contain hydrocyanic acid, a measured volume of solution is made alkaline with ammonium hydroxide and then treated as above.

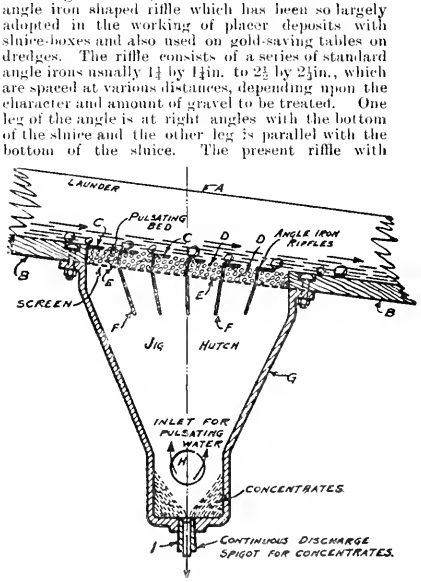
The method checks established methods well, and is remarkably free from interferences. KOH and $K_4Fe(CN)_6$ interfere when present in large amounts; the small quantities usually present in commercial cyanides have practically no effect at all.

It is accurate in the presence of the double cyanide commonly used in electroplating.

It is seen that the method gives only the 'free' cyanide in silver and copper cyanide solutions. Experiments show that more than the free cyanide is obtained in zinc-cyanide solutions, the end point occurring when the Zn : KCN ratio is approximately 1 : 0·6. In copper and silver cyanide solutions Liebig's method works the same as the new method. In zinc cyanide solutions Liebig's method gives the total cyanide, that is, the free cyanide plus the cyanogen in the zinc double cyanide. Hannay's method gives the total cyanide in all three cases. This is a disadvantage because only free cyanide is ordinarily desired."—G. E. F. LUNDELL and J. A. BRIDGMAN, *Journ. Ind. Eng. Chem.*, July, 1914. — *Engineering and Mining Journal*, Sept. 19, 1914, p. 529. (G. H. S.)

METALLURGY.

THE RICHARDS PULSATOR RIFFLE.—"The Richards pulsator riffle is the result of the application of an intermittent pulsating current of water to the Hungarian riffle, which will be recalled as that angle iron shaped riffle which has been so largely adopted in the working of placer deposits with sluice-boxes and also used on gold-saving tables on dredges. The riffle consists of a series of standard angle irons usually 1¼ by 1¼in. to 2½ by 2½in., which are spaced at various distances, depending upon the character and amount of gravel to be treated. One leg of the angle is at right angles with the bottom of the sluice and the other leg is parallel with the bottom of the sluice. The present riffle with



Launder with Pulsator Riffle.

automatic discharge is obtained by applying the pulsating current, which is the underlying principle of the Richards pulsator jig. The water pressure required is 7lb., equivalent to a head of 16ft. The quantity of water required is approximately 40 gal.

per minute per sq. ft. of riffle surface. The only adjustment is the water supply, which is controlled by an ordinary globe or gate valve in the water pipe.

The construction and operation of the pulsator riffle will be understood by reference to the accompanying diagram, where *A* is a suitable launder; *B*, bottom of launders; *C*, common angle iron riffles; *D*, openings between the angle iron riffles which may be ⅜in., ½in., or ⅝in., depending upon the size of gravel to be treated; *E*, a screen preferably of brass with openings to allow the passage of fine gold or mineral; *F*, guide plates inside of the jig hutch; *G*, the jig hutch which together with the guide plates is fastened to the under side of the launder or tables and under the screen; *H* is the inlet for the pulsating water current from the Richards pulsator valve; *I* is the spigot at the bottom of the jig hutch for discharging the gold particles or mineral concentrate. It is said that the riffle will not pack and therefore requires no clean-up, and has many other advantages over the Hungarian riffle. This device is made by the Denver Engineering Works Co., of Denver."—*Mining and Scientific Press*, Aug. 22, 1914, p. 316. (G. H. S.)

AN ADIABATIC CALORIMETER. "The calorimeters in general use are subject to many errors, namely, those of radiation through the surrounding wall, of conduction through the metallic parts and the thermometer, and those due to the inaccuracy of the thermometer readings.

The following description of the Riche adiabatic calorimeter deals with an attempt to eliminate all these errors, through the use of a *vacuum cup* with a suitable covering, proper precaution having been taken to insulate the different parts.

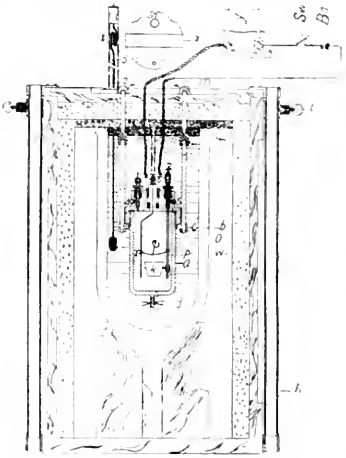
Description of Apparatus. In all calorimetric work the rise in temperature, multiplied by the quantity of water affected, gives the calories directly, provided there is no loss or gain in heat. Fig. 1 is a sectional sketch of the bomb and calorimeter, which has been designed. It consists of a bomb, *a*, which may be of any type; of a vacuum cup, *b*, containing water; a stirrer, *s*, and a thermometer, *T*. The stirrer is of the screw propeller type, which is very efficient. The vacuum cup is of standard size, 14 cm. wide, 30 cm. high, inside measurements, and was obtained through the Vereinigte Fabriken f. Laboratoriumsbedarf, Berlin, Germany.

The bomb is of the Mahler-Kroeker porcelain type and should last for 8,000 combustions; and, inasmuch as it has two outlets, 1, 5, determination of carbon may be made. The support for the bomb is a hinged ring, *c*, supported by two brass rods connected with the cover. The rods are insulated by passing them through hard rubber, *m*, and a hard rubber cap is screwed on the end of each rod.

To prevent any possible heat loss from the vacuum cup, it is surrounded by an oak box, *w*, lined with tin, pressed cork, *o*, glued to the wood. On the cover, in addition to the cork, is glued a piece of hair felt, *f*, ½in. in thickness, effecting a tight joint with the top of the vacuum cup.

A hinged door in the front facilitates the handling of the vacuum cup and the cover is raised and lowered by means of telescope tubes, thus giving opportunity to place the bomb in the split ring and to adjust the ignition wires. After the cover is lowered the thermometer is placed in position through a hole in the cover, into which is screwed a brass tube to support and protect it.

Calculation of Results. The reading at ignition is taken as the initial reading. This subtracted from the final reading gives the total rise. This rise multiplied by the weight of water (plus the hydrothermal equivalent of the apparatus) gives the total number of calories liberated. From this is taken the heat resulting from the ignition, and the nitric acid formed. This result, divided by the weight of substance, gives the calories for the unit of material oxidized.



The Riche calorimeter, therefore, saves from 50% to 75% of the operator's time. To this saving of time there is added the perfection of the entire calorimeter device. The calorimeter as an established laboratory device will soon be employed generally in connection with the large consumption of all fuels, foods and commercial products, the value of which depends upon their heat contents."—*Metallurgical & Chemical Engineering*, Sept., 1914, p. 606. (G. H. S.)

FREEMAN GRINDING PAN.—"One of the most interesting of the different types of pans in use is the Freeman, devised at the Great Fingall mine, Western Australia, and in favour also at the Sons of Gwalia, Lake View & Star, and other mines. It is of the standard size of five feet diameter, and about 30 inches deep, provided with dies on the inside bottom, and shoes on a revolving gear-driven muller. The speed is necessarily adjusted from experience with any particular ore, but between 50 and 60 revolutions per minute is generally desirable. The pan is centre-fed and has a classifying action, centrifugal force throwing the heavier and larger particles to the outside. About half way up one side of the body a three-inch pipe is fitted, this extending half way to the centre of the pan, ending with a short bend. The function of this pipe is to take off pulp of the grade desired. Particles of ore travel inward as they decrease in size, finally reaching the proper part of the pan for discharge. The pipe may, of course, be arranged to take the pulp at any point across from the periphery to the muller yoke; this depending on what grade of material is

required. So far, in Western Australia, the Freeman pan has been used in stage crushing such as the following: five 1200-pound stamps crushing a fairly hard ore through about six to eight-mesh screens (a duty of 8·47 tons per stamp day in 1913), the pulp flowing to one Freeman pan, which only allows one product to escape by the pipe classifier. This product goes to Wilfley tables, is further classified, and the coarse material is finally ground in tube-mills with more concentration. As an intermediate grinder, such as described, this pan does excellent work. Shoes and dies last only from 20 to 30 days; but considering the work done, this is not excessive. The cost of preliminary grinding at the Lake View & Star mill in 1913 was 18 cents per ton of ore milled, stamping being ·31 cents and tube milling the same."—*Mining & Scientific Press*, Aug. 29, 1914, p. 317. (G. H. S.)

CHLORIDISING ORES.—"Frequent references have been made of late in technical literature to processes for chloridizing low-grade ores containing gold, silver, copper and lead, according to methods devised by N. C. Christensen, Jr., and T. P. Holt, of Salt Lake City, Utah, and to the mills utilising such processes. In a patent recently granted to these men (Christensen assigning his interest to George H. Dern, of Salt Lake City) they disclose details of operation and furnace construction, practically as used at Park City, Utah, by the Mines Operating Co., under the direction of Messrs. Dern & Holt.

The preparation of the ore is accomplished by grinding, mixing with about 2% of fuel, unless the ore contains fuel such as sulphides, mixing also with salt, and finally adding moisture varying from 5% for coarse, non-absorbent ore, to 15% for fine, absorbent material. The mixture is then roasted in a shaft furnace, such as illustrated in Fig. 1, in which air is forced through the charge from beneath, and in which certain zones are maintained as indicated in the figure. In this manner a counter-current flow of ore mixture and air is achieved. The roasted ore is withdrawn from the bottom of the furnace and fresh mixture added from above. The roasting zone is always within the furnace charge and protected by roasted ore below and unroasted mixture above. The roasted ore is lixiviated with acidified water and the metal is precipitated from the solution as explained in previous articles to which reference has been made.

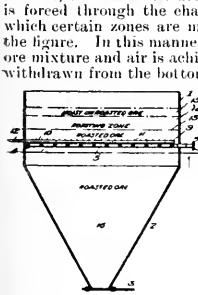


FIG. 1.—*Furnace for Chloridizing-Roasting.*

The furnace shown in the figure consists of a circular or square upper part 1 and a conical or pyramidal lower portion 2. The air-blast pipe 5 is disposed about the upper portion and connected with cross pipes, all having openings on their lower sides. A stirring grate 9 is placed just above the air pipes, and by its operation the roasted ore is gradually removed from the roasting zone and discharged through the gate 3 at the bottom. In starting the furnace, it is filled with roasted ore up to the level 13. A layer of live coals is then prepared and on this is charged the green mixture; the blast is applied, and the operations of discharging roasted ore and charging green ore proceed as desired. Gases from the furnace are dustless and are conducted to absorption towers to produce liquor for lixiviation."—*Metallurgical and Chemical Engineering*, Oct., 1914, p. 658. (G. H. S.)

MINING.

MODIFIED PRIMING DEVICE.—"I recommend in order to prevent the detonator pulling out that a double cord be used as shown, to take a clove hitch over the fuse or the wires. The method of doing this

Manner of Constructing Primer.

is evident from the drawing, and the cord must be included in the stick when it is manufactured."—W. W. JONES.—*Engineering and Mining Journal*, Aug. 1, 1914, p. 212. (G. H. S.)

STADIA READING REDUCTION ON TRANSIT.—"The Kentfel & Esser Co. offers a new device for facilitating the operations involved in the stadia method of surveying. It consists of two segment markers applied to the vertical circle, by one of which is read the percentage of the rod interval representing the horizontal component, at the angle at which the telescope is inclined, and by the other, the percentage representing the vertical component. The arrangement of these is shown. The transit itself thus automatically performs one of the operations in reducing the stadia readings.

Suppose the observed stadia distance to be 480 ft. and the telescope when sighted to be inclined at such an angle that the reading at the horizontal percentage scale is 0·97 and at the vertical is 0·17. Thus the correct horizontal distance is 480 × 0·97 = 455·6 ft. and the difference in elevation is 480 × 0·17 = 81·6 ft."—*Engineering and Mining Journal*, Sept. 12, 1914, p. 488. (G. H. S.)

MISCELLANEOUS.

BONDING ASPHALT TO CONCRETE.—"Among the several problems to be met in the recent construction of the concrete reservoirs in San Francisco was that of bonding asphalt to concrete. After making unsuccessful attempts to apply the asphalt direct to the concrete in various ways, the expedient of first painting the surface with coal tar was tried, and it was found that when applied in this way under right conditions

the asphalt would adhere so firmly that after it had cooled it could not be broken away without bringing pieces of concrete with it. Mr. Abbott states that, in order to secure this result, the concrete surface was first scrubbed clean with a bristle brush, and over this a thin coating of hot coal tar was 'painted.' The coal tar should be heated in small quantities, brought just to the boiling point, and then applied immediately. Heating the coal tar in large quantities, which necessitated some delay before it could all be used, did not give such good results, and the gangs were therefore not allowed to heat more at a time than they could apply quickly as soon as it began to boil. The asphalt was spread over the tarred surface in the usual manner." *Indian Engineering*, Oct. 17, 1914, p. 231. (J. A. W.)

Abstracts of Patent Applications.

(C.) 64/14. Seymour Edgar Hubbard (1), Alfred Edward Barker (2). Improvements in stope and like conveyors. 11.2.14.

This application is for a form of stope conveyor. The proposed arrangement consists of two endless chains driven by sprocket wheels, and passing over carrying idlers and tail pulleys.

The carrying portion is composed of short lengths of ordinary conveyor belt carried on iron bars attached at the ends to the endless chains. The chains are made up in detachable sections, each section being fitted with a length of belt as just described.

The driving sprockets are at the upper end of the conveyor, and operated by a crank handle.

The ordinary tightening gear is applied and is fully described in the specification, but is not specially mentioned in the claims.

The idlers and tail shaft are to be carried on ordinary drill bars set up from the floor to the roof of the stope.

Several arrangements of securing the lengths of conveyor belt to the chains are described.

(C.) 320/14. John von Nostrand Dorr. Apparatus for the separation of finely divided solids from liquids. 16.7.14.

This application is for an apparatus patent for the separation of finely divided solids from liquids. It consists of a number of superposed trays contained in a vat in connection with a hollow shaft to which is attached sweeps for collecting and discharging the settled solids.

(C.) 395/14. E. F. McConl. Improvements in grinding mills. 6.10.14.

This application relates to an improved grinding mill for "pulverising ore, rock, coal or other material."

The grinding is effected between two discs, one of which may be fixed and the other revolving, or both of which may revolve in the same direction but at different speeds.

The axis of the revolving spindle may be horizontal or vertical.

The spindle driving the rotating disc is carried eccentrically in a hollow shaft, the axis of the shaft and the spindle being parallel.

The driving power is applied to the hollow shaft and transmitted to the spindle by a form of epicyclic train.

A friction clutch connects the driving pulley to the hollow shaft and an arrangement is provided, which in the event of some hard substance getting into the grinder, throws the clutch out by the longitudinal travel of the spindle.

A further arrangement provides for an electric bell being put in operation by this longitudinal travel.

The ground material is carried away by a current of air produced by a fan attached to the revolving disc.

In one arrangement described balls are provided between the discs running in suitable races and crushing the material between the balls and the race.

(C.) 414/14. P. C. P. Booty. Improved valve device for air gas machines. 28.10.14.

This application refers to improvements in carburetters, consisting of a tapered gas valve and a petrol valve having a tapered groove, each valve being automatically controlled by a floating belt in the gas chamber.

(C.) 423/14. D. C. Reinohl. Improvements in recovering precious metals from their ores. 9.11.14.

This patent refers to an apparatus for treating ores and dissolving the precious metals therein contained. The salient points consist of a trough or conduit provided with paddles or conveyors for the purpose of agitating the pulp, pipes for supplying solution to the pulp and the necessary gear for driving the moving parts in the apparatus, etc.

(C.) 430/14. Holman and Holman. Improvements in or relating to rock drilling engines. 24.11.14.

This patent claims in a rock-drilling engine the provision of a sleeve-chuck having a longitudinal through-bore and so interengaged at its rear-end portion with an extension from the working piston that the latter can rotate it. This is the principal claim, and it also is combined with a claim for feeding water from the back-end of the rock-drilling engine through the piston, through the chuck, and eventually through the drill bit to the working point.

Changes of Address.

ADAMSON, R., *l/o* Rhodesia; Lady Mary Mine, Francistown, Bechuanaland.

BOYDELL, H. C., *l/o* Brakpan; c/o Consolidated Main Reef G. M. Co., P. O. Box 2, Maraisburg.

DRUCKER A. E., *to* 62, London Wall, London, E.C.

GOW, G. A., *l/o* New Zealand; Lebong Donok, Benkoelen, Sumatra, Dutch East Indies.

GUTHRIE, F. G., *to* Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

HUNTER, W. J. R., *to* P. O. Box 108, Germiston.

INGRAM, J., to Vogelstruis Estates and G. M., Ltd., P. O. Box 116, Roodepoort.

LEV, JAMES, *l/o* Germiston; P. O. Box 50, Nigel.

LOCKHART, T. L., *l/o* Johannesburg; Pickstone G. M. Co., Duchess Hill, Hartley, Rhodesia.

LUCKE, P. K., *l/o* England; General Superintendent, Cia Minera De Penoles Mines, Mapimi, Dgo, Mexico.

MACLEOD, D., *l/o* Germiston; Van Ryn Deep, Ltd., P. O. Box 225, Benoni.

SCOTT, D. E., *l/o* Rietfontein; Van Ryn Deep, Ltd., P. O. Box 225, Benoni.

SHORT, A. R., *to* P. O. Box 67, Jeppes.

SIM, A. C. M., *to* Streatley Avenue, Auckland Park, Johannesburg.

STUDOLPH, N. H., *to* c/o P. O. Box 374, Salisbury, Rhodesia.

STOKES, R. S. G., *l/o* New York; 1st London Field Co., Royal Engineers, 6th Division, 3rd Army Corps, British Expeditionary Force.

WARD, J. P., *l/o* Swaziland; Glenbain, Lufafa Road, Ixopo, Natal.

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Vol. XV. MARCH 1915. No. 8.

Proceedings
AT
Ordinary General Meeting, March 20th, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, March 20th, 1915, Prof. G. H. Stanley (President) in the chair. There were also present:—

24 Members: Prof. J. A. Wilkinson, Prof. J. S. Cellier, Messrs. R. A. Barry, J. Gray, H. Meyer, F. W. Watson, John Watson, H. A. White, Dr. W. A. Caldecott (Members of Council), R. W. Bennett, J. A. Campbell, R. A. Cooper, R. V. Crake, E. H. Croghan, J. M. Dixon, B. J. Hastings, G. Melvill, S. Nettleton, S. Newton, E. A. Osterloh, J. F. Pyles, W. A. C. Tayler, J. A. Taylor, and W. E. Thorpe.

8 Associates and Students: Messrs. H. C. F. Bell, O. A. Gerber, A. King, G. Lundholm, A. Milliken, W. J. Sutherland, S. E. Willows Munro, and J. A. Woodburn.

6 Visitors, and Fred Rowland (Secretary).

The minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBERS.

The Secretary announced that since the last meeting of the Society two Associates had been admitted:

PRYOR, THOMAS, Oorgaum P.O., Mysore State, S. India. Mining Engineer.

RUTGERS, CAREL ARNOLD, Paleleh Gold Mines, Ltd., Paleleh, North Celebes, Dutch East Indies. General Manager.

The Secretary: It seems that we obtain more support from foreign countries at present than we are doing from our own district.

The President: It is again regrettable to have to announce there are no members up for election. It is even more regrettable, I think, that there are no students. Among all the number of young employees on these producing mines of the Witwatersrand there are really very few indeed who belong to the Society as students. The Secretary tells me it is even worse than that: that there are none at all at the present time.

GENERAL BUSINESS.

DEPARTURE OF MR. WM. CULLEN.

The President: During this week Mr. Cullen, whom we all know and remember as a very active member of the Society in the past, for years a Member of Council and a Past-President, has left the country, we hope for an even more useful sphere of activity in England during this time of stress. I think we might send to him our best wishes for his future prosperity and our regret that he has had to leave us.

VISIT TO MODDER B. MINE.

The President: We had a very interesting excursion during the month to Modder B., where we were treated very well indeed by the Chairman and Directors and resident staff. They did everything possible to make our visit the huge success it was. I must say, when the project was mooted, I was very dubious indeed as to whether we should meet with success; but the result exceeded my anticipations. I think we must pass a very hearty vote of thanks now to the Chairman, Directors, and staff of the Modder B.

The vote of thanks was carried unanimously.

AWARD OF PRIZES TO EVENING CLASS STUDENTS.

The President: I next have to perform the pleasant duty of presenting the prizes won by successful students at the evening classes in Chemistry and Metallurgy held at the South African School of Mines. These prizes are, of course, presented by the Society, and

consist of technical books. The Secretary will now read out the names of the successful students.

Inorganic Chemistry:

Theoretical, Stage I.—£1 10s. R. Railton.
Practical, Stage I.—£1 10s. J. F. B. Davis. J. R. Railton.

In Stage II. no awards have been made.

Metallurgy:

Stage I.—£1 10s. G. Ross.
Stage II. £2. No award.
Stage III. (Gold).—£3. T. L. Thorne.

Assaying:

Stage I. £1 10s. D. W. Peck.
Stage II. £3. G. J. Snook.

The President: In giving these prizes for competition among the evening class students, the Society is furthering one of its aims and objects, that is, the forwarding in every way it can of the science and practice of Chemistry, Metallurgy and Mining. The Mining Department we do not deal with in the matter of prizes, because the Association of Mine Managers deals with that; but, for the Chemical and Metallurgical branches we present the prizes. While these prizes are not of very great intrinsic value, they do enable students to buy some useful books. There is, besides, the honour of having won the prize and having been the best student in a particular subject for a particular year. Of course, for this year the classes have been carried on and the prizes have been won under circumstances of quite exceptional difficulty. The School of Mines was faced with the difficulty of many students leaving on account of the war, and other students finding it difficult to attend because of having to work overtime on the mines in order to partly fill the places of those who have been called away; so that those who did continue to attend merit all the more some reward for their endeavours.

It scarcely needs any words from me to emphasize the advantages to students attending such classes, although it is a sorry fact that, of the total number of employees, only a very small percentage indeed seem to seek to benefit in this way. Quite apart from the mental benefit which it confers, and which should be obvious, there is the very important question of material benefit. Now, in that connection, I need hardly point out that those who do take the trouble to attend classes, and particularly those who achieve successful results, are generally marked men on the mines, and, where there is an opportunity not only for their own benefit, but for the benefit of their superiors, they are in most cases given more responsible positions. There is one, at least, of our former evening class students who is now a reduction officer on one of the mines, several are cyanide managers, and more are foremen. To-night, I find myself in a threefold capacity; I have the honour of being President of this Society which presents the prizes, I am on the staff of the South African School of Mines and, in particular, I have had the pleasure of teaching some of the students for whose work prizes have been given to-night.

I am sure that you will all join with me in wishing these successful students further success and the best wishes for their future. (Applause.)

CLAUDET MEMORIAL STUDENTS' PRIZE.

There is another very interesting item which has just come to my knowledge. Mr. Morris Green was a day student at the School of Mines some three or four years back and won the Chamber of Mines Scholarship. He elected to continue research work in the School of Mines here and, as a result of that, read two or three papers both before our Society and the Institution of Mining and Metallurgy, London. I hear that he has been awarded by the latter body the Arthur Claudet Memorial Students' Prize of ten guineas for his paper on the "Effect of Charcoal in Gold Bearing Cyanide Solutions with reference to the Precipitation of Gold." This paper was published in our *Journal* for November, 1913. (Applause.)

AMBULANCE SHIELD COMPETITIONS.

The Secretary: I wish to draw the attention of members to the Competitions which are being held next month for the Ambulance Shields presented for ambulance work by this Society and the Transvaal Coal Owners' Association. The Competitions will be held for the Surface Shield on the 17th April; those for the Underground Shield on the 18th April, and for the Colliery Shield on the 25th April. We hope that those members on the mines will do their best to see that a team is entered from each mine. The Managers, Secretaries, and Hospital Superintendents of each mine have been circularized; and I shall be pleased to let them have any further particulars on application.

LEAD ACTUATE.

The Secretary: You may remember, in the November, 1914, *Journal* one of our members, Mr. A. Macdonald, published a

letter dealing with lead acetate. In that letter he drew attention to the fact "that the average consumption of this substance by the mining industry of the Rand and outlying districts is 700 tons per annum. The present suppliers are Holland and France and, up to the outbreak of hostilities, Germany. Since the outbreak of the war the United States have made a bold bid to secure this market. The pre-war cost to the mines from the first-mentioned sources averaged £35 per ton. American consignments are being sold at approximately £50 per ton. The worst features of dealing with the United States are the slow and uncertain deliveries and the prohibitive price. The available amount of pig lead on the Continent is rapidly diminishing. In fact, acetate of lead produced at present is chiefly made from scrap lead. There is a golden opportunity for British firms interested in the manufacture of lead products. At present British manufacturers seem to content themselves with the production of white and red lead, litharge, and lead pipe-fittings."

About the same time I received a letter from the Manager of the Rhodesian Broken Hill Development Co. The Company has a certain amount of lead which they wish to dispose of, and they have asked our advice as to what to do in the matter. We sent them Mr. Macdonald's note, and said we did not know whether there was a possibility of their undertaking anything of this description, but there seemed to be a considerable market for this substance and it might be worth their consideration. A copy of that letter was sent to Messrs. Hooper, Speak and Co., who are the Consulting Engineers to the Rhodesian Broken Hill Development Co. They wrote to us by the last mail, notifying us that they had received this letter, and they say: "As General Managers of some important gold mines in Western Australia we experienced similar difficulties in obtaining lead acetate on the outbreak of hostilities. We, however, then learnt from Mr. W. K. Betty that in Mexico, litharge was used instead of lead acetate, and accordingly we had this tried at the Great Boulder Perseverance Mine in Kalgoorlie, where after several months' use it is still giving satisfaction. We believe that litharge could similarly, to a large extent, replace lead acetate on the Rand with advantage in cost. If so, the Rhodesia Broken Hill Co. would probably endeavour to supply your requirements. It is chiefly in the interests of that Company that we now write you with the object of inducing some of your members to give litharge an exhaustive trial. If, as we anticipate, this results in a considerable demand for litharge in South Africa, the Rhodesia Company can then arrange to supply it, or with pig lead if you found it preferable to utilize your own litharge from cupellation."

Mr. Cullen has sent in a note which Mr. J. A. Campbell will read.

Mr. Wm. Cullen *(Past-President):* Mr. Macdonald drew attention in the November *Journal* to the shortage of this material. My Company investigated the possibility of making the acetate in the country from such materials as scrap lead and wine vinegar, but enquiries soon showed that the price of acetic acid from this source was prohibitive. Lead nitrate suggested itself as a substitute, and fortunately we were in possession of suitable plant and material for the manufacture of this product. We are now producing a salt of approximately 99% purity, and should the necessity arise we can extend our plant to meet the full requirements of the Reef, assuming sufficient lead (from scrap or ores) to be obtainable in South Africa. There seems to be no reason why lead nitrate should not be quite as good as lead acetate for the purposes of the mining industry. The nitrate is the more stable salt and can be stored indefinitely without change. It is somewhat more soluble in cold water, and contains a higher percentage of lead. Lead acetate at £35 is equivalent to lead nitrate at £40 per ton. If there be sufficient demand the nitrate can be easily made and sold at the latter figure or less. Therefore there should be no necessity to pay fancy prices for imported lead acetate while an efficient substitute can be made in the country.

Mr. H. A. White *(Member of Council):* I should like to remark in connection with the note you have just heard read and the note read by Mr. Rowland, that the principal use of lead on these mines is undoubtedly in the form of lead acetate. Litharge is employed to some extent in the form of used cupels from the assay office, but, of course, that form is useless for "dipping" zinc or coating it with lead crystals for use in the precipitation plant. I may say I have had a cwt. of this lead nitrate bought in a commercial manner from the dynamite factory, and used in an ordinary way, and it was perfectly satisfactory in every respect. The price is fully 10% lower than the present price of lead acetate, and it contains approxi-

mately 10% more lead in the same weight of material. I think Mr. Cullen is to be congratulated on the success he has attained.

Mr. A. King *(Associate)*: I would like to substantiate Mr. Whites' remarks. A sample of this lead nitrate was sent to the Knights Deep, where it was used for dipping zinc shavings, with quite satisfactory results; and, as was to be expected, when equal weights of lead acetate and lead nitrate were dissolved separately the lead nitrate solution was found to give a good coating of lead on a greater weight of zinc than the lead acetate solution did.

BRITISH GLASS AND PORCELAIN WARE.

Mr. F. W. Watson *(Member of Council)*: We have just heard of one of the numerous cases where the war has interfered with supplies, but I think there is nothing that is interfered with more than the supply of chemicals. It seemed to me that it might be of interest to the members of this Society to mention some of the work that has been done by the Institute of Chemistry and the Society of Public Analysts. Early in August the Board of Trade was approached by these Institutions to appoint a Committee to inquire into the supply of chemicals previously supplied by Germany and Austria, and an influential Committee was appointed.

On September 22nd there was a meeting held by the two Societies " to consider what steps should be taken to ensure a satisfactory supply of laboratory reagents, and also to consider what steps should be taken to ensure a satisfactory supply of glass and porcelain apparatus and filter paper ". Two Committees were appointed for this purpose one for the reagents and the second for the glass, porcelain and filter paper. They took evidence: and these resolutions were passed: -

> " That this Committee, having taken evidence from various glass manufacturers, are of opinion that there is every prospect of the industry being taken up and worked successfully in this country, provided that the manufacturers can be afforded some guarantee of permanency for their enterprise, and that they may have some reasonable assurance that at the conclusion of the war the newly developed industry will not suffer from foreign competition, hitherto made possible by economic conditions which do not prevail in this country.
>
> " That publicity be given to the hope of the Councils of the Institute of Chemistry and the Society of Public Analysts that all users of chemical laboratory apparatus will give their utmost support to the development of new industries in this country, particularly in connection with chemical porcelain and glass apparatus and filter paper."

Several firms were approached, many of whom were prepared to supply the articles required.

On October 30th the Council of the Institute " resolved to appoint an Advisory Committee to conduct research on glass, to be carried out in the laboratories of the Institute with a view to arriving at suitable formulæ to be freely available to manufacturers willing to assist in maintaining the continued supply of glass apparatus ". It was also resolved " that a Sub-Committee be appointed to prepare a list of reagents ordinarily required, indicating the standards of purity which would be necessary for analytical work, and giving, as far as possible, some information as to the quantities likely to be wanted; that the standards of purity be based on those adopted by a Committee appointed by the Eighth International Congress of Applied Chemistry, and that the Sub-Committee should indicate those substances which are regarded as indispensable ". It is proposed that, as soon as the list has been prepared, manufacturers shall be asked to state how far they are able to supply the products scheduled, having regard to the degree of purity required. The hope was expressed that all users of chemical reagents would give their utmost support to the development of the industry in this country."

These proceedings were published in the *Proceedings* of the Institute of Chemistry in November, 1914, and since that time the first report of this Glass Research Committee has been presented and an abstract given in *Proceedings* published by the Institute in February, 1915. Results of experiments made by Professor Herbert Jackson and Mr. T. R. Merton were embodied in this report " giving the formula for an alumina-soda glass suitable for the manufacture of chemical laboratory ware. Copies of this formula were sent to firms likely to be interested in the industry. It was stated subsequently that the formula was similar to one used by several makers who had been conducting experiments with the same object in view.

Mr. Hehner having analysed several varieties of Jena glass, Professor Jackson and Mr. Merton conducted further experiments and reported to the Committee, giving a formula for a glass which, in their opinion, is a satisfactory substitute for Jena glass in respect of its resistance to water and reagents. This formula was also sent to manufacturers. Melts were made in accordance with both formulæ, and the Committee were fortunate in securing the assistance of Messrs. Kempton, of Vauxhall Walk, who kindly attended at the Institute to conduct blowing experiments on the small scale. They considered the glass to be suitable for blowing and manipulation." Then, the Home Office informed the Committee that "it was desirable to obtain a suitable formula for the manufacture of miners' lamp glasses. The Committee were provided by a manufacturing firm and by the Home Office with specimens of approved glasses, analyses of one of which were undertaken by Mr. Bertram Blount, Mr. Otto Hehner and Mr. Walter C. Hancock. It was found, however, that attempts to prepare experimental melts corresponding with the data obtained were not successful; but the analyses were very helpful in the synthetical experiments carried out by Professor Jackson and Mr. Merton, who have been able to prepare a very promising glass for miners' lamps, and have communicated the formula for making it. It has been arranged that melts on a fairly large scale will be made in the near future at Messrs. Kempton's works, the Institute agreeing to contribute to the expense involved." . . . . . "Messrs. Baird & Tatlock (London), Ltd., have erected a factory at Walthamstow for the production of chemical laboratory ware, and expect to be able soon to produce vessels of their own manufacture. Samples of beakers and flasks have been received from Messrs. Jules Lang & Son, whose factory is at Eu, near Dieppe, and samples of tubing, capillary tubing and flasks, from an alumina soda glass, have been received from Messrs. James Powell & Sons, of Whitefriars Glassworks, Tudor Street, London, E.C., but this latter glass has been made principally with a view to use in X-ray work. These samples, however, have been examined by members of the Glass Research Committee, and are reported to be similar to ordinary laboratory glass in their behaviour with water, acids and alkalis." Then, "Samples of porcelain basins and crucibles, etc., of Royal Doulton ware have been received and have been found quite satisfactory. These are supplied at approximately the same cost as the Royal Berlin ware. The Worcester Royal Porcelain Co., Ltd., have also reported that they are making ware for laboratory purposes. Samples of Whatman Filter Paper have been received from Messrs. W. & R. Balston, and are reported on as very satisfactory in respect of retaining fine precipitates. Samples so far examined indicate that acid treatment will result in the production of paper at least equal to the best foreign makes as regards the amount of ash and filtering properties."

I think it is satisfactory to know that some steps have been taken to meet the threatened shortage of laboratory supplies. Of course, we have not come really to the pinch yet, as old stocks are not quite exhausted; but the time may come when we shall be hard put to it, and it is hoped that the work undertaken by these people will bear fruit, and that we shall be able to keep going.

Mr John Watson (*Member of Council*): I jotted down a few figures regarding this matter which may be of interest. My friend, Mr. F. W. Watson, has spoken of the manager of a large chemical apparatus firm in South Africa, who did not know that the Institute of Chemistry, Society of Chemical Industry, etc., were doing anything towards the production of these goods. It is very reassuring, in these times, to hear that a good old British firm like Doulton and Co., of Lambeth, are turning out porcelain crucibles and evaporating basins which the Institute consider as good as Royal Berlin porcelain, at practically the same price.

Exports from Germany to the United Kingdom for 1912 of china and earthenware (including porcelain) amounted to £637,000. The German exports of glass, of all kinds, to the United Kingdom amounted to £1,033,722, whereas the glass exports from Great Britain to Germany only amounted to £63,500 value. Germany's total glass exports to the world for 1913 had a value of practically £6,000,000 sterling. These figures (for which I am indebted to the latest available International Whitaker and the *Statesman's Year Book*), show that there is a large field open to the British, or whoever may succeed in capturing this trade.

Prof. J. A. Wilkinson *(Vice-President):* In respect to what we have just heard, I have been more or less in touch with the manufacturers of glass and porcelain ware, etc., since the crisis began, and I believe now that the English firms are advertising that they will supply porcelain and laboratory glassware, etc., at about the same prices, and as good as any of the German apparatus to which we have hitherto been accustomed. Personal experience alone can verify the latter.

THE THEORY OF TUBE MILLING.

Printed in Journal, February, 1915.)

By H. A. White, M.Inst.M.M. (Member of Council).

DISCUSSION.

Mr. H. A. White *(Member of Council):* briefly introduced his paper which was printed in the February *Journal*.

The President: This paper is largely a record of experimental work and of certain deductions made therefrom. With regard to the experiments themselves, the best contribution to discussion and the most likely way in which discussion can come forward is by bringing forward other experiences and the deductions made therefrom may lead also to very useful discussion.

Dr. W. A. Caldecott *(Past-President):* The thanks of members are due to the author for his admirable discussion of this subject, and to the manner in which, owing to experimental facilities recently available, he has been able to supplement and enlarge his earlier paper on the theory of tube-milling read before this Society ten years ago. The conclusions drawn in the paper are confirmed by practical experience in many respects, and should serve to indicate the directions in which further progress may still be expected.

Though not dealt with in the paper, the author has recently investigated and confirmed a formula[*] for the determination of the tonnage of sand fed per tube-mill per 24 working hours, which may be of use to members in ensuring good working conditions. The formula referred to is arrived at as follows:—

Let a = Percentage by weight of −90 product in screen or pulp.
b = Percentage by weight of −90 product in tube-mill feed pulp.
c = Percentage by weight of −90 product in tube-mill outflow pulp.
d = Percentage by weight of −90 product in final tailing pulp.
B = Tons of ore crushed in stamp-mill per tube-mill per 24 working hours of latter.
T = Tons of dry solids fed per tube-mill per 24 working hours.

Then, the tons of −90 produced in a tube-mill per 24 working hours

= the tons of ore stamp-milled per tube-mill per 24 working hours multiplied by the difference $\frac{}{100}$ between the percentage of −90 in screen and final pulps.

= tons of dry solids fed per tube-mill per 24 working hours multiplied by the $\frac{\text{difference}}{100}$ between percentage of −90 in tube-mill inflow and outflow pulps,

That is, $$B \times \frac{d-a}{100} = T \times \frac{c-b}{100}$$

Therefore, $$T = \frac{B(d-a)}{c-b}$$

Whilst the conditions of the experimental work detailed by the author did not embrace all factors involved in ordinary tube-milling practice, he has supplemented this deficiency by data obtained under practical working conditions, and has laid stress upon the importance, which was realized here at an early date[*], of adequate sand feed to secure the best results. Until recent years the importance of this requirement has hardly been fully realized by operators elsewhere, notably in the United States, where the original scheme appears to have been to feed the coarser portion of the battery pulp to tube-mills in limited quantity only, with the object of obtaining the tube-mill outflow as a finished product similar to the tube-mill classifier overflow. The long survival of this tendency is contrary to the practice of the tube-mill circuit which was inaugurated by Mr. J. R. Williams with the first tube-mill started on the Rand at the Glen Deep Mine on 4th May, 1901, and which has been generally followed here ever since. The different practice abroad has probably been accentuated by the hill-side mill-site gravitation ideal, by the limited capacity of the

[*] See this *Journal*, May, 1905, p. 290, Vol. V.

[*] See paper by Noel Cunningham in the *Mining Press* (2nd edition), 1910.

[*] See W. B. Dowling, in this *Journal*, August, 1905, p. 69, Vol. VI.

mechanical classifiers frequently employed, and by the desire of avoiding the re-elevation of pulp involved in the operation of the tube-mill circuit; but the practice of very-limited tube-mill feed is nevertheless remarkable in view of the general acceptance of the necessity in dry-crushing by rolls of separating and returning for further reduction oversize particles in the roll product.

The author has referred to the expression of tube-mill capacity in terms of tons of −90 product per 24 working hours. This conventional measure is of much value for comparative purposes on the Rand, just as the same system, but using tons of −200 mesh (linear) product, is useful in all-sliming ore crushing. In the voluminous discussions here and abroad on the laws of crushing it appears that more progress would have been made and confusion saved if there had been some uniformity among writers as to the exact meanings of the terms used.

In the writer's view the term "crushing" is equivalent to "pulverization", "disintegration", or "deformation", and involves the change of the relative position of each constituent particle to every other particle in a mass of ore. In the case of ductile metals like lead, pressure causes a flow or relative change of the position of the constituent particles without disintegration, and in this case "deformation" would be equivalent to "crushing".

On the other hand, "breaking" is equivalent to "fracture", "rupture", "cracking", or "splitting", and involves the conversion of the mass of ore into two or more fragments, but without changing the relative position of each constituent particle to every other particle.

To apply these terms to a concrete case, it is obvious that much less power is required to break a one inch cube of ore into two fragments than to crush it fine enough to pass a 90-mesh screen. Likewise it is obvious that under similar conditions it will take twice as much power to produce two tons of −90 product as to produce one ton of −90 product, and that the resulting product in the first case exposes twice the area of the second case.

In actual practice, pressure or impact on ore appears to result in a combination of "breaking" and "crushing", owing to wedges or cones of ore in contact with the crushing faces being driven into the mass so as to shatter it, a concurrent partial disintegration simultaneously taking place in the production of a little fine powder.

The President: It is rather a sad state of affairs that we have no more discussion available at this meeting. Of course, there is the next meeting still: but it would be better, in furtherance of our new procedure, to get all the discussion possible at the meeting at which the paper is presented. The paper is a very valuable one indeed, not only on the Rand, but, as has been pointed out, to fields overseas, where problems connected with tube-mill working are receiving a lot of attention, and where a lot of experimental work is being done, judging by articles which have recently appeared. I hope an effort will be made to bring forward something of value at the next meeting; but we must not let this go any further without passing a very hearty vote of thanks to Mr. White for presenting this paper to us.

The vote of thanks was agreed to with acclamation.

NOTE ON A METHOD OF VENTILATING SINKING SHAFTS, WINZES AND DRIVES.

(Printed in Journal, February, 1915.)

By S. Nettleton (Member).

DISCUSSION.

Mr. S. Nettleton *(Member)* introduced his paper, printed in the February *Journal*, and demonstrated the device referred to.

The President: The paper is now open for discussion. It strikes me this is one of those very simple and yet very effective little devices of which we are very glad to hear. I am sure other members must have similar things which, with very little trouble, can be brought before us.

Mr. R. A. Barry *(Member of Council):* Whilst there is, of course, nothing new in the method described by Mr. Nettleton for ventilating these working faces, I cannot call to mind that I have ever seen brattice cloth or sacking used quite in the way he suggests for this purpose, and it may be that the idea is worth a trial. This might especially be the case in instances where there is no work going on except at the face itself, and where the appliance would not have to remain there for any great length of time. When there is intermediate work going on,

I fear that the appliance would be too easily damaged either by rough and frequent handling, or by neglect of workmen, or deliberate ill-use. Again, in cases where the appliance might have to be in position for a considerable time, as for instance, in a sinking shaft, calling constantly for extensions and additions, I very much fear that on account of its flimsy nature it would not withstand the hard usage of underground life. Mr. Nettleton does not give any figures as to cost, though he states that his idea would be cheaper than either wood or iron pipes. I doubt very much whether in the long run it would be cheaper.

Of course, the most effective of all methods is the use of an electric fan, but this is not always possible, and I have found in practice that where an isolated cross-cut or long drive of any size is being driven, or while sinking a shaft, the use of a good strong wooden launder with four sides is, on the whole, the most satisfactory method, unless a fan can be afforded. A very small jet of air inserted into this launder suffices to set up quite a strong draught, which rapidly clears the face from fumes or bad air. The cost of such a wooden launder is not heavy. It is no trouble to install, and can always be added to with ease, and when finished with, is always valuable for use elsewhere or for other purposes. The wooden launder can be made practically any size without weakening it, and it withstands the concussion of blasting, and all the very rough usage to which it is subject underground. It can be carried as near as is necessary to the working face—a little loose rock thrown over it protecting it efficiently from the blast. It is easily installed, and can be carried on the footwall. The fact that it is heavy does not really materially matter. The cost to us for 4-sided launders made of 2in. Oregon pine and measuring 20 in. × 20 in. (inside measurement) has been 5s. per ft. in 15 ft. or 20 ft. lengths. This is cheaper than galvanized iron piping, of similar capacity, and of sufficient strength; while the piping is always getting damaged more or less seriously. We have found in practice that there is very little leakage from the wooden launders, and we have used it fairly extensively and effectively. The addition of a water jet or atomiser inside the launder is simple and effective in laying the dust, the atmosphere being thoroughly moistened during circulation. Of course, in a narrow drive or winze a heavy launder such as I refer to becomes cumbersome, and at times impossible, but I doubt very much whether Mr. Nettleton's device would stand up against the rough handling which it must meet with underground, and I think it would call for constant repairs and adjustment. In such cases I use the rough and somewhat inefficient method of forcing a circulation by installing 6 ft. or 10 ft. lengths of galvanized piping, or wooden laundering, at several convenient spots, at various distances apart. A jet of compressed air blown through a nozzle is turned into such pipe, and creates quite a healthy draught with a minimum expenditure of compressed air.

Whilst there is nothing novel or particularly striking in any of the suggestions contained in these notes, my excuse for bringing them forward must be the hope that they may promote discussion and possibly lead to the disclosure by some one of improved methods for dealing with ventilation of isolated and small development faces where, in all probability, the seeds of disease are frequently sown owing to the bad air breathed by workmen employed therein.

Mr. S. Nettleton *(Member):* I think that I can reply to one or two of the points raised by Mr. Barry, who has very kindly offered to give the arrangement a trial. Wooden launders are certainly satisfactory for ventilation of a large shaft if kept at some distance from the face, but the cost —5s. per foot—is fairly high. Assuming that the brattice cloth arrangement is made from material 5 ft. in width, there would be produced a tube having sectional area of about 2 sq. ft. A 20 ft. length of conduit would require 11⅓ yards of canvas, which should be procurable at 8d. per sq. yard. Estimating cost of connections and making up at 17s. per conduit, there would be 20 ft. of tube for 24s. 8d., which is practically 1s. 3d. per lineal foot. Of course, the cost of conduits depends very largely upon the quality of the brattice cloth, which varies in price from 6d. to several shillings per square yard. With regard to durability, I have known brattice cloth show practically no deterioration in coal mines after being in use over 12 months, though the shock from blasting operations in collieries is not nearly so great as in the mines of this district, where detonation of heavy charges of explosives might tend to injure canvas arrangements.

I thank you very much for the manner in which you have received the paper.

GOLD AND PRICES

(Read at September Meeting, 1914.)

By Prof. R. A. Lehfeldt, D.Sc.

REPLY TO DISCUSSION.

There have been a few criticisms on this paper of mine, and the chief reason, to my mind, is that they raise questions that I did not have time to touch. I had to assume a good deal, and, as it was, I took up a good deal of the Society's time with a subject that is outside its usual business. In particular I had to assume the quantity theory of money. This, I think it is fair to say, is orthodoxy amongst economists. There are some who differ from it; but, still, it is the orthodox opinion; and I was occupied in drawing conclusions and not in proving the basis of my argument, and I cannot undertake to do that now.

There are some detailed criticisms of Mr. Aiken's eminently worth referring to; I am only sorry he is not here to-night. He criticizes my definition of Currency. The Society may remember that I counted up as Currency, coin, bank notes, and bank deposits. Now, he says that, "Large volumes of goods change hands in the first place on mere entries in books . . . . . and in still other cases little currency passes at all where people deal with each other as both buyer and seller. It is clear that the total sales of goods cannot under such circumstances agree with my statement of it". That is a point about which I should like to ask the practical business man; is it the case that many transactions are settled by cross-entries in books? Take a merchant who pays, perhaps, £50,000 in the course of a year. How much of that does he simply cancel off his books because he has bought from and sold to the same person? I should be surprised if the transactions that are settled in that way amounted to more than 1 or 2 of the total. This criticism is perfectly sound as far as it goes: it is a question of the magnitude of the effect. I think there is really very little of that sort of business done; so that the rough estimate which I made of the amount of currency available should be practically correct. If I am wrong on that point, and if any business man can tell me where I am wrong and can give me any facts, I should be only too glad to learn about it.

There is another point of Mr. Aiken's which is very much of the same kind. He says that the bank deposits do not fairly represent "the limit of the ability of the community to draw cheques. That limit is the credit which the banker is prepared to give by way of overdraft to his clients." Well, now, can anyone—as Mr. Aiken is not here—tell me whether the banks, either in South Africa or elsewhere, are prepared to give unsecured overdrafts to any large extent? I know if I have overdrawn 15s. I promptly get a note to that effect, but then I am not a business man. Of course, it there is any appreciable amount of credit given in that way, then my figures are, by so much, wrong; but, if I am right in supposing that this is quite an exceptional kind of transaction, then I take it that Mr. Aiken's criticism hardly holds. Of course, secured overdrafts are already taken into account. What happens is that you go to a banker and deposit securities and are entitled to overdraw to a certain extent, and you are credited with so much. Therefore, when you are estimating the amount of deposits in the country, you have already taken into account the possibility of overdrawing against securities. That I had already allowed for in my paper. The only point I had not allowed for was the possibility of overdrafts which are not represented by anything in the banks' accounts, that are extra to the bank deposits which are known to exist.

There is a further point that Mr. Aiken makes here: it is rather ingenious, but it is a misunderstanding. It is this: "Take, as an instance of the underlying fallacy, the case of a person who wishes to borrow £100. If he goes to his banker, and the latter, as often happens, opens a special account for the advance and places the £100 at the credit of the borrowed's ordinary operative account, the banker has, according to the Professor, increased the currency of the community by £100, and after the customer has passed his cheque, the bank's deposits are at the same figure as before the operation. But assume that the banker declines to make the advance and the borrower gets a cheque from another client of the bank, as a loan, then the currency is decreased on the Professor's theory by £100." Well, that is not the case. I never said it was "decreased". What I should say is that it would remain the same. Mr. Aiken goes on to say: "There is no difference in the real currency which has passed in both cases." That is a misunderstanding: there is a difference.

If you go to a bank and, by depositing securities, you get an overdraft of £100 and use that to make payments to other people, the currency of the country has been increased by £100. Whether that bank deposit remains in your own name or somebody else's makes no difference, because the mere handing of a cheque from one person to another does not alter the amount of deposits, it merely transfers some from one person to another. But if, being refused by my banker, I go to, say, Mr. Aiken and, as a favour, ask him to lend me £100, the operation does not increase the amount of currency. I don't mean because he would refuse me: but if he agrees, what he does is to take £100 from his own account and put it to mine, and the total currency is the same as before. The difference is a perfectly real and important one. The reason of it is that Mr. Aiken is not in a position to manufacture money, and the bank is; in fact, it is its function to do so. For that reason, I must maintain my point that bank deposits, subject to cheque, not time deposits, but bank deposits that can be drawn by cheque are the real measure of the currency of the country, in addition to bank notes and coin.

Mr. Aiken says, further: "There is one point in regard to the quantity of gold which I personally have never been able to satisfactorily explain on the quantity theory." The point he is referring to here is how the quantity of money affects the rate of interest. My paper tries to show how the quantity of money affects prices. I said nothing at all about rate of interest. So it is hardly a fair criticism that I did not explain that matter; but, seeing that Mr. Aiken has raised it, I may, perhaps, be allowed to refer to it because it is a point which interests Johannesburg: all the economic effects of gold production come home to us in one way or the other. Mr. Aiken says: "The rate of discount settled week by week by the Board of the Bank of England is based on the amount of surplus gold it possesses, having regard to the bank's liabilities. Now, if more gold were coming into the market than it required to carry on the business as it stood at any particular moment, it is legitimate to assume that in the free market for gold, of which London is the centre, it would show itself, if nowhere else, and were it to do so, the first effect one would anticipate would be a reduction in the discount rate. This has never been used as an argument by the quantity theorists, for the reason that the trend of discount rates fixed weekly by the Bank of England does not support the theory"—but I think they do, as a matter of fact. It is a point which has been remarked on before this that, if there is a considerable superfluity of gold produced at any time beyond what the market requires, it tends to accumulate principally at the Bank of England; and, if the Bank of England finds that it has more gold than it needs in comparison with the business that it has to do, the effect is to reduce the discount rate; and, I think, history, as well as theory, shows that point. But that is not the only effect of an increased supply of gold. There are two effects, and they are contradictory; and that is perhaps the reason why there is often a want of clearness as to the result. The immediate effect of an excessive gold supply is that the banks find they have an amount of gold which is out of proportion to the business they have to transact, and they endeavour to get more business by reducing their rate of discount. That is the immediate effect; but, if there is a long continued excessive supply of gold, the effect is to raise prices; as I have endeavoured to point out in my paper (it has often been pointed out before, of course) and the effect of rising prices is to raise the rate of interest. Consequently, if an excessive production of gold goes on for a number of years, the long continued influence prevails, and there is a rise in the rate of interest which overpowers the temporary effect due to the large gold balances in the hands of the banks. So that, the same cause produces two opposite effects; a rapid immediate effect of lowering the rate of interest; but if the over-production is kept up for years, the opposite effect of raising the rate of interest. There were certain points in Mr. Weston's contribution to which I should like to refer, because there is a misunderstanding on Mr. Weston's part which rather destroys the value of his criticism. He goes so far as to say, apropos of the uncertainty of numerical data, why try to solve the equation? That is to say, why write the paper at all? Seeing that I have tried to write the paper, I should like to point out to him that the margin of error of 1 which he refers to is not 1 on the annual production of gold, but on the total amount of gold in the world; that is a matter of £20,000,000. Mr. Weston says: "It is not right to take an assumed average value over a term of years and give that value to a particular year. This is especially so when he is work-

ing with such a small allowable margin of error as 1″ to 1½″. In other words, he wishes to prove that had the Rand produced only £950,000 to £1,450,000 less of gold in 1911 a rise of prices might have been avoided." I did not attempt to prove anything of the sort. The 1″ to 1½″ is a matter, not of £1,000,000, but of £20,000,000 or £30,000,000 of gold production. It is also rather interesting to quote this remark: "It will be noted also that Dr. Lehfeldt assumes that all credit money has as its only basis gold. In my reply to my previous paper, I stated that many economists held that such currency tended more and more to be based on the whole wealth and credit of the nation." Now, there again there is a misunderstanding. I agree entirely that the currency nowadays is based upon the whole wealth and credit of the nation. The existence of vast quantities of bank deposits which act as currency do represent, not gold, but all sorts of real property; but, under our existing state of affairs, that currency is not made available unless there is a proportion of gold to meet it. To realise this one has only to think of a banker with £50,000,000 worth of deposits backed by sound securities, but with no gold at all in his till. I think you will agree that the gold is an essential part of the mechanism of the currency at the present time. Mr. Weston says: "Mr. Lloyd George apparently believes that the £1 and 10s. notes now accepted as legal currency in England have other backing than any gold reserve. They are scraps of paper—with the whole credit of the nation behind them." Well, now, as a matter of fact, as soon as these notes were issued the Government began to accumulate gold reserves, and have at present £25,000,000.

NOTES ON THE USE OF POTASSIUM PERMANGANATE AS CYANICIDE IN SAND-FILLING SOLUTIONS.

(Read at October Meeting, 1914.)

By R. A. Cooper (Member).

REPLY TO DISCUSSION.

In reply to Mr. White's discussion of my paper I must first thank him for the interest which he took in the subject. Unfortunately, since the data for that paper was collected the war has caused reduction in numbers of the laboratory staff, but has had no such effect on the work of that department, and consequently it has not been possible for me to do as much more investigation of the subject as I should have liked. Nevertheless, the practical side of my argument has, I consider, been demonstrated to be true by actual work on the sand-filling plant carried out during the past few months. In that connection I must enter a counter protest against Mr. White's accusation that our "neglect of addition of permanganate" was either "light-hearted" or a "highly dangerous precedent," and quote his own further remarks in reply. He says that the paper "illustrates the point that on every new mine where it is proposed to use current sand for stope-filling the whole of the relevant conditions will repay investigation." Investigation of the relevant conditions on the East Rand Proprietary Mines, as quoted in my paper, showed that in every case the solutions sent underground contained such a small percentage of cyanide that no addition of cyanicide could be demanded at any time, and its addition in any case was only a precaution against accidental increase of cyanide, and was not occasioned by a normal requirement. Authoritative permission to discontinue the use of permanganate was therefore unnecessary in this particular case. In view of what I have published, I strongly suspect that in the event of sudden accidental increase of cyanide the permanganate would not be a very satisfactory protection. During the periods when permanagate has not been added, both circulating solutions and the solution delivered to stopes have been carefully tested to ensure absolute safety.

Since the reading of my original paper in October the permanganate addition was suspended entirely for several months and no alterations whatever have taken place in the cyanide strength of solutions. The strength of circulating water overflowing from cones on the surface varies between 0·002 —and 0·001″ KCN—the solution delivered to stopes averages about 0·0004 KCN, and stope drains contain traces of sulpho-cyanide only and are slightly alkaline. This omission of permanganate did not result in any increase of cyanide at any time.

I think that Mr. White has misunderstood slightly my remarks as to the very considerable dilution of original solution and the apparently unexplained disappearance of cyanide from stage to stage; possibly I should have expressed myself more clearly. Although the total water used is about 37 ,

it is to be remembered that the sand is tipped into a circulating solution of 20,000 gallons —which solution after washing the sand to the cones is there largely removed for re-use. This circulating solution, or cone overflow, does not accumulate cyanide in solution even when permanganate is not in use, and therefore it is clear that a very large proportion of the cyanide in the original sand is eliminated at this stage. The total water used therefore does not give a direct measure of the dilution of the cyanide, and a drainage water of 1:3 solution strength has not to be reckoned with.

The water used is the same as that used in ordinary reduction works practice, and is always kept alkaline with lime, and it may be presumed that it is not a cyanicide, as the works cyanide consumption is not increased by its use: it does not contain more than traces of organic matter, and is, in fact, the ordinary neutralised mine water of the property. The accumulated slime which is being treated may leave organic matter in the sand, and consequently destroy much permanganate and evolve ammonia. As cyanide is undoubtedly most effectively disposed of during this process of sluicing sand from dump to cones, even in the absence of permanganate or other added cyanicide it appears that cyanide must be decomposed during circulation and aeration of the circuit water. Further, although the final wash from the sand may contain 0·01 KCN, it apparently does not follow that the solution retained by the sand remains at that strength after discharge. Owing to the difficulty encountered by shiftsmen of accurately testing very weak working solutions it is likely that final washes often contain less cyanide than is reported. On testing a wash reported at 0·007 KCN I found that it actually contained 0·008 KCNS and free and other cyanides as determined by evaporation with ammonium sulphide 0·005 KCN. A series of samples of sand was taken from the vat from which this wash was drawn: the moisture varied from top to bottom of vat between 9 to 15 %, and cyanides in retained water from 0·004 KCNS 0·004 KCN at the top to 0·008 KCNS and 0·005 KCN at the bottom, and after discharging and tipping, almost all cyanides were converted to sulpho-cyanide: apparently the decomposition in drained sand is very rapid. With that speed of decomposition it is not surprising that there is no accumulation of free cyanide in circulating solutions and that water and sand delivered to stopes are almost free of it. Under these conditions the "disastrous results" foreseen by Mr. White as a result of mixture of stope drains and mine water need not cause much alarm. Apparently the decomposition of cyanide observed on the surface is completed in the stope either by a continuation of the same reactions or by the neutralization in the stope of mine waters by the alkaline solutions bearing the sand, and a consequent precipitation of iron hydroxides and formation of "prussian blue". The regulation of ferroso-ferric sulphates may be necessary to obtain pure products, but we have the fact that cyanide will combine with excess of freshly precipitated ferrous-hydrate only, evidenced by the efficiency of ordinary "cyanide antidote" as used on the Reef. The reference which I made to the absence of hydro-cyanic acid gas or cyanides in a tunnel through a sand dump did not refer to a dry dump as Mr. White presumes; the tunnel was driven during a dry season, but was extremely wet —apparently owing to expressed water from sand tipped above. The water was strongly acid and charged with ferrous and ferric sulphates.

I have not been able to make any further experiments as to time of contact, etc., necessary for decomposition of cyanides by permanganate, or as to any effect of precipitated oxides of manganese in suspension as suggested by both Mr. White and Mr. Green. Certainly the steady reduction of solution strength in cyanide, which I have observed and which Mr. White confirms, takes place just as steadily when no permanganate has been used for months as when it is in constant use.

Mr. White's statement that "the paragraph with reference to the high cost of cyanides may be dismissed with the remark that the process now in use is not likely to be changed for any cheaper one unless some other advantage is secured," is very remarkable. Surely a reduction in cost is an argument strong enough to appeal to practical men, even without inducement of "other advantages," when it can be demonstrated that under some existing conditions at least there is no disadvantage attached. The arguments I now advance in favour of the disuse of permanganate in certain cases are purely practical in character, and are based essentially on tests made on working plant rather than on laboratory experiments or theoretical considerations. Were our knowledge of the exact reactions involved as perfect as it will doubtless one day become, all con-

siderations would point to one definite conclusion, but as it is the practical test is the only safe one. It would certainly be of interest if someone could ascertain whether such results as I have observed obtain on other properties or are entirely due to relevant conditions on the East Rand Proprietary Mines.

PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M.

(Printed in Journal, Dec., 1914 - Jan., 1915.)

The President: As you will remember, Mr. Laschinger promised at the last meeting to contribute to this discussion, but has been unable to do so owing to pressure of work. He has promised to have his contribution ready either for insertion in the next *Journal*, or will read it at the next meeting.

Prof. J. Cellier *(Member of Council):* Mr. Rickard's paper has been most interesting. The author deals very ably with the subject and is in a good position to do so, having travelled much and being in possession of all literature past and current. His conclusions regarding the Rand I fully endorse, and in this brief discussion wish to confine my own remarks to these. The instances quoted of outcrops richer than deep-level and deep-deep-levels can be considerably amplified as, *e.g.*, May Consolidated, and Glencairn, Glen Deep, and Simmer East; Henry Nourse, Nourse Deep, and South Nourse; Crown Reef, Crown Deep, and South Rand, not to mention impoverishment in depth in the same mine, *e.g.*, Lancaster West*, Luipaardsvlei Estate, Robinson Deep, Simmer & Jack, etc., etc.

As Mr. Meyer points out, there may be local exceptions to this rule. For instance, the writer wears a tie-pin consisting of a small nugget recovered in a mortar-box on a banket mine during a month when this mine registered one of the biggest losses in its history, but yet was showing free gold underground. Thus individual samples prove very little. Yet we see these regularly quoted by chairmen of companies and other gentlemen connected with the industry! It is to be presumed they act on the Dutchman's maxim: "If it gives no relief it consoles anyway." Judging by a number of cuttings of such speeches I am inclined to think that they are most unduly optimistic. That ore much below the profitable limit of extraction was worked in days gone by there is no gain-saying. This tended to obscure the high grade of ore then exploited, and occasionally still to be found in the hanging or foot-walls of stopes worked then. Sampling at that time was, for a certain 200-stamp proposition, now forming the nucleus of one of the largest gold mines on earth, carried out by a single individual whose only training had been that of a veterinary surgeon. Plans of mines working three contiguous reefs in the Germiston district were more often than not kept in all black, with the result that middle reef some 7 ft. thick was worked, though it has never since become profitable. Directors of Companies have been hard to convince, but in 1913 the Consolidated Gold Fields of S.A. had written off £1,400,000 as depreciation on investments as a result presumably of Mr. Webb's authoritative statement (Messrs. Hellman and Schmidt's having proved of no avail). Many reasons have been given for the lowering grade: decreased working costs, increased tonnage, tube-mills, shortage of native labour, and over-abundant machine drills; but only when these had been somewhat exhausted did the consulting engineer become convinced of what the sampler had observed in the deep levels, such as they were, in 1899.

In justice to the engineering profession it must, however, be confessed that one of the ablest engineers the Rand ever had, in making estimates of potential ore reserves of blocks of ground now absorbed in the big amalgamations, invariably placed the value of a block of ground lying to the dip of another at 1 dwt. lower, when nothing definite was known about the value of either. Both higher working cost and falling grade in depth result in a lower payable tonnage per claim. Unfortunately these three factors work in the same direction, with depressing results on profits. Had some account been taken of this, Simmer Deep shares would not have sold at 1s. 9d. to-day when we find written*: "Taking it that the Simmer Deep will yield no more than 30,000 tons of milling ore per claim, there would be in its 1,035 claims 31,000,000 tons. On the basis of 6·7 dwt. ore, profits would run to about 10s.

* This mine is advertised in the local press as for sale, "lock, stock and barrel," on Thursday, 25th March, 1915.

* "Round the Rand," by J. W. Broomhead, 1908, p. 40; published by the *Financial Times*.

per ton, or a total of £15,500,000, whilst at the market price at the date of writing, the claims stand at no cost. Indeed, the amount already spent on the properties, together with the cash in hand, exceeds the current market valuation of the shares by over one million sterling," etc.

Dr. W. A. Caldecott *(Past-President)*: The following historical reference may be of interest. In 1857, a few years after the discovery of the Australian goldfields, a commission of enquiry into the state of gold-mining was appointed by the Parliament of Victoria. Among the evidence taken was that of Prof. McCoy, who reported that "the experience in every country proved that the yield of gold decreased with the depth after a certain small limit". The Commission agreed with this view, but likewise held that "although in any one spot the gold would speedily be worked out in a vertical direction, the horizontal extension of these beds and veins are immensely greater than people are aware of".

As the author has already visited the most important goldfields in the world with the exception of the Rand, it is to be hoped that in due time we shall have the pleasure of a visit here from him, when he will have the opportunity of seeing actual conditions for himself. Such a visit would also serve to familiarize him with the desirable scope and relations between the local technical Societies, a subject in which he has recently displayed an interest*.

TWENTY-FIRST ANNIVERSARY OF THE SOCIETY.

The President: This is our 21st anniversary meeting. That it is held under most unlooked for circumstances, I think you will all admit and, under these circumstances, it has not been possible to arrange any sort of celebration of the event. We hope that before the end of my year of office, at any rate, that the reason may have passed away and that we may have cause for rejoicing in other directions as well—(applause)—but at the present time, that concludes the business of the meeting.

The meeting then terminated.

The following papers will be submitted for discussion at the meeting of the Society to be held on Saturday, the 17th April, 1915.

THE USE OF SCOOP DISCHARGES IN TUBE-MILLS.

By Walford R. Dowling, M.Inst.M.M. (Past-President).

The fact that there are now about three hundred tube mills installed on the Rand, and that in modern plants two-thirds of the ore crushing is done in these machines, makes their crushing capacity of great importance, and the author trusts that the following records and results of work done with tube-mills with scoop discharges may be of interest to members. Though the time now available does not permit of dealing with certain other factors in modern tube-mill practice, some reference may be made to these later in the reply to any discussion that may arise.

The first three tube-mills installed on the Knights Deep Mine in 1905 were of Davidsen make, with the shell dimensions 18 ft. 8in. long by 4 ft. 9 in. diameter, and were originally fitted with peripheral discharges. The outlet and inlet trunnions are of the same internal diameter, namely, seven inches. Back-flow of pulp at the feed end was a continual source of trouble and expense in clearing up the pulp spilt. As the necessity for a thicker feed became more apparent these difficulties increased. After various fruitless attempts to correct this defect at the feed end of the mills, it was decided to instal a lifting device inside the mill at the discharge end, and thus lower the internal pulp level. A scoop discharge with radial arms and perforated plate of 2 ft. diameter was installed, which at once corrected the evil and improved the crushing. The earliest blue print on record of this apparatus is dated 20th April, 1907, and is shown in the accompanying illustration (Fig. 1). Scoop discharges, but of improved pattern as experience was gained, have been worked in these three mills ever since.

Some time later Dr. W. A. Caldecott went further into the subject of scoop discharges, and decided to carry out experiments with an apparatus of which the arms were sufficiently long to

* See *The Mining Magazine*, February, 1915, p. 68.

utilize an effective diameter of 50 inches of the perforated plate referred to later. Before proceeding further I may describe the apparatus used, which is equivalent to an enlarged discharge trunnion, but without the disadvantages of the latter. A reference to the accompanying illustration of the scoop (Fig. 2) will show that there are four arms attached to a plate or casting. This, when bolted to the discharge end of a tube-mill, provides four chambers, which in turn as the mill revolves bring up their quota of pulp and deliver it into the outlet discharge trunnion. To prevent the pulp from running back into

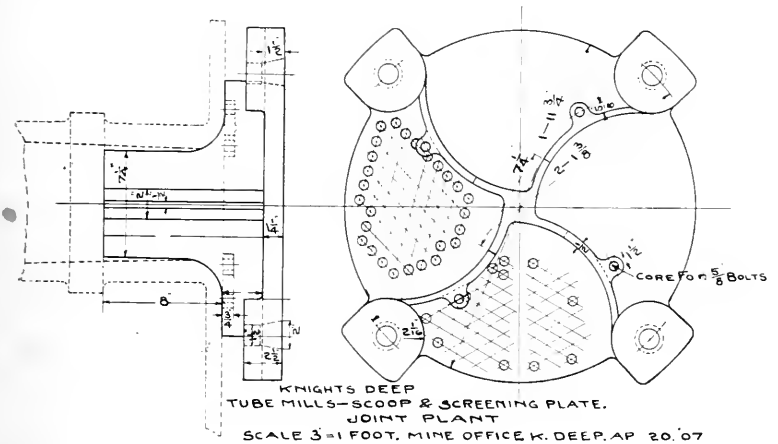


Fig. 1.

a lower chamber, that portion of the casting which projects into the discharge trunnion is divided into four channels, so that each chamber discharges through its own channel. To ensure that pebbles shall not be discharged with the pulp, a perforated plate, with the necessary number of holes of suitable size, is bolted in front of the scoop and at a distance away from it sufficient to enable pulp to pass through the perforations and flow to the periphery of the scoop. The apparatus maintains the level of the pulp in the chamber formed at the discharge end of the tube-mill at a point governed by the length of the elevating arms. In the course of experimenting it was desired to vary the depth of pulp in the tube-mill proper with a view to locating the point at which greatest crushing capacity was obtained, without incurring other disadvantages. As the easiest and cheapest method of arriving at this, different effective diameters of perforated plate were used in front of a scoop whose diameter was 50 inches. Such a scoop gave a range from the existing internal diameter of the trunnion, which in the case of the experiments was 17 inches, up to what was practically peripheral discharge when flint liners of 6 in. to 7 in. thick were used in a 5 ft. 6 in. diameter mill. In the course of this paper, varying diameters of scoop discharges will be referred to. The diameter specified in each case is that of the portion in operation of the perforated plate placed in front of the scoop to prevent the pebbles from passing through. The effective diameter required was obtained by blanking off by means of segments of plate the necessary ring area of the outer portion of the perforated plate. Care was taken to make these segments watertight by the use of insertion between the segments and the perforated plate. The advantage that a scoop has over a trunnion discharge is that the larger the latter is made the greater is the loss of fall for launders and plates, whereas with the scoop even the smallest existing opening in the discharge trunnion in any tube-mill could probably be further reduced, thereby increasing fall. When the diameter of the outlet trunnion becomes unduly great,

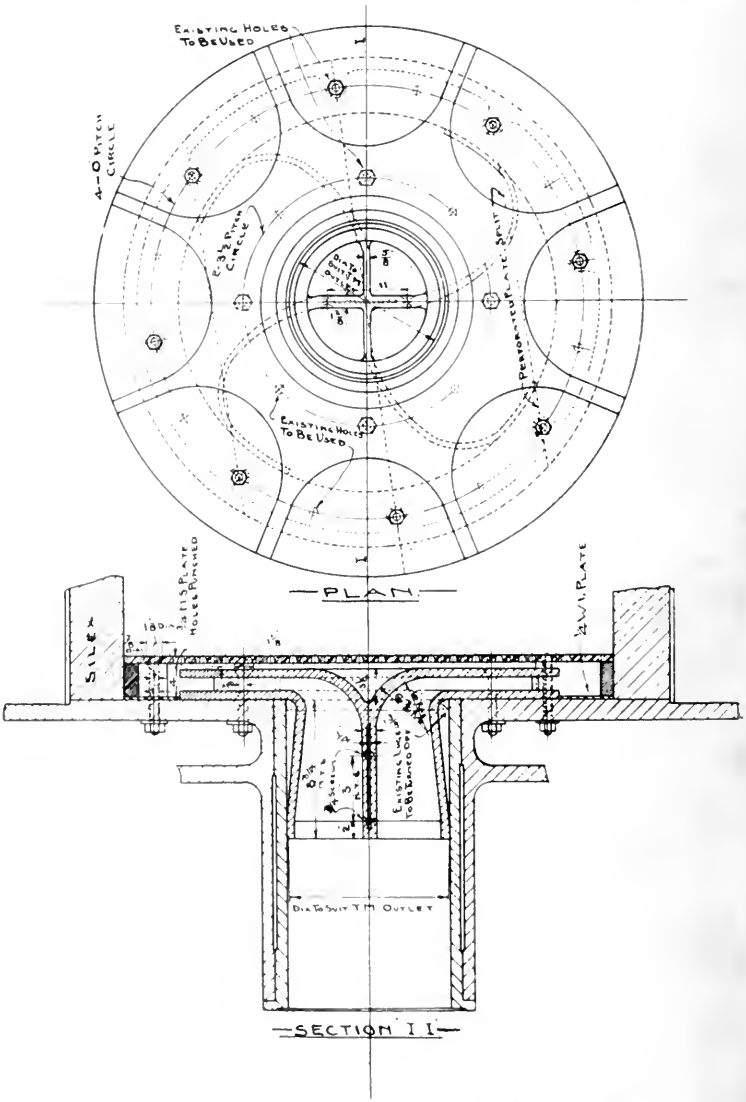


Fig. 2.—Knights Deep, Ltd.—Type Mat Supported Screening Plate, 23th Aug., 1911.

mechanical difficulties arise in the use of plain bearings. The scoop discharge can of course be adapted to any existing tube-mill.

The first 5½ ft. × 22 ft. mill tried with the above scoop discharge, having a diameter of 50 in., was at Knights Deep in November, 1911, and this mill was lined with 6 in. flint blocks and was driven from the main engine. It was impossible to obtain accurate power consumption readings, but information very useful otherwise was obtained. It was very quickly found that although crushing was improved, liner and pebble wear were excessive, and with the arrangements available it was impossible to maintain a sufficient pebble feed to equal the consumption, which rose from a normal 7 tons daily to 14 tons of pebbles. The liner wear was excessive throughout the mill, and at the end of 16 days it was found necessary to renew 3 ft. of liner at the discharge end of the mill. The average life of similar liners in the usual mill, where the outlet trunnion was 17 in. in diameter, was about 70 to 80 days.

Various internal effective diameters of scoop discharge were tried, and the accompanying Table No. 1 exhibits the results obtained. The following comments may be made on this set of results:—

In Trial No. 1, with a 50 in. scoop, which was the first scoop trial made, a daily feed of 12 tons of pebbles was not sufficient to make up for the wear. This large pebble consumption, together with the excessive wear on liners already referred to, seemed prohibitive, and these facts, combined with relatively small increased capacity, were the main cause of change to smaller diameter scoops. Even in Trials Nos. 2 and 3 with smaller scoops the pebble feed under the conditions existing could not be maintained. Column No. 8 of the table is rather interesting where the duty is expressed in tons of −90 per 24 hours per ton of pebbles in the mill. The figure with the 50 in. scoop is close on double that obtained with the ordinary 17 in. discharge trunnion. It should be noted that the figures given of each of the trials is the average of not less than four runs of the mill, observations being taken on as many days. In fact, as is well known to those with practical experience in tube-milling trials, the constant liability of essential conditions to vary renders it very necessary for data to be confirmed by averages from several runs, the results from individual tests being otherwise liable to be misleading through unavoidable experimental error. In order to reduce this error as much as possible, tonnage determinations were made, and samples of tube-mill inflow and outflow pulp taken for grading, at frequent regular intervals throughout the course of every run recorded in this paper.

The experiments at this mine were stopped after the completion of the trials mentioned in the table, as two important essentials were lacking, namely, pebble feed facilities and power consumption measurements, and the scoop was removed to the Simmer & Jack Mine, where experimental work was commenced in May, 1912. At the latter mine a 190 H.P. motor was installed with a wattmeter for the accurate determination of power consumption. After some preliminary experiments in which the machinery was tested and the operators became familiar with the new conditions, very important results were obtained, of which typical sets are shown in Table No. 2. The tube-mill used was lined with Osborn liners (steel-ribbed) and the internal diameter was 62 in. By examination of this table the following deductions may be made:—

High pebble load (trial No. 1) with consequent high power consumption produced only slight increase of −90 product above trial No. 5. Also, the smaller scoop of 24 in.

TABLE No. 1.—TUBE MILL EXPERIMENTS WITH SCOOP DISCHARGE AT KNIGHTS DEEP, LTD.

Trial No.	Diameter of Scoop Discharge in Inches.	Actual Diameter of Tube Mill in Inches.	Tons of Sand Feed per 24 hours.	Percentage Moisture in Sand Feed Pulp.	Tons of Pebbles in Mill.	Tons of −90 Product per 24 hours.	Tons of −90 Product per 24 hours per Ton of Pebbles.	Mean Height above or below Axis of Pebbles in Mill in Inches.	Tons of Pebbles Fed per 24 hours.
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)
1	50	56	326	34·46	5·72	114	19·93	−9½	12·0
2	33½	58	314	36·35	7·39	123	16·34	−7	14·8
3	25	59	316	36·16	9·08	116	12·77	−4	11·0

TABLE No. 2. TUBE MILL EXPERIMENTS WITH SCOOP DISCHARGE AT SIMMER & JACK PROPRIETARY MINES, LIMITED.

Trial No. (1)	Duration of Trial. (2)	Diameter of Scoop Discharge in Inches (3)	Actual Diameter of Tube Mill in Inches. (4)	Revolutions of Tube Mill per Minute. (5)	Tons Sand Feed per 24 hours. (6)	Percentage Moisture in Sand Feed Pulp (7)	Mean Height of Pebbles in Mill, above or below Axis, in Inches (8)	Tons of Pebbles in Mill. (9)	Horse Power Consumed (10)	Grading Analysis Feed. 60 Per cent (11)	60 Per cent (12)	90 Per cent (13)	60 Per cent (14)	Grading Analysis Discharge. 60 Per cent (15)	90 Per cent (16)	Tons of 90 Product per 24 hours. (17)	Consumption of Pebbles per 24 hours, tons. (18)	Horse Power per ton of 90. (19)	Tons of −90 per Horse Power. (20)
4	9 hours	24	62	32·15	366	36·8	+ 8·12	16·08	159·0	51·0	29·7	16·3	9·2	31·3	59·5	158·02	—	1·00	0·99
5	9 ,,	24	62	32·15	358	36·1	+ 1·50	12·84	138·6	59·4	26·4	11·2	11·0	31·1	57·9	151·80	—	0·90	1·11
	Average	24	62	32·15	362	36·5	+ 4·81	14·16	148·8	56·7	28·1	15·2	10·1	31·2	58·7	156·41	16·71	0·95	1·05
6	10 hours	31	62	32·25	359	36·4	+ 5·75	15·36	163·7	58·0	26·6	15·4	12·4	31·2	56·6	148·41		1·10	0·90
7	10 ,,	31	62	32·25	366	37·6	2·75	11·18	135·3	61·1	24·0	11·9	15·0	31·9	53·1	139·89		0·96	1·03
	Average	31	62	32·25	363	37·0	+ 1·50	13·27	149·0	59·5	25·3	15·2	13·6	31·5	51·9	144·06	19·04	1·03	0·97
8	Average of 8 trials of 24 hours each	24	59½	32·10	357	13·2	− 5·00	8·79	125·0	57·4	27·9	14·7	16·2	33·3	50·5	127·99	8·21	0·97	1·02

diameter gave more tons of −90 product than the 34 in. scoop (trials Nos. 6 and 7) for the same horse-power and with smaller pebble consumption. The series of trials (No. 8) where the pebble load was maintained at such a weight as to require only 125 H.P., although yielding a good −90 duty, indicate that the crushing capacity of mill is restricted.

After some months' run at the Simmer and Jack, the scoop illustrated, with the 190 H.P. motor and wattmeter were transferred to the Robinson Deep Mine, and experiments started in January, 1914*

The Robinson Deep has installed at the tube-mill plant a very efficient pebble feeding plant. The plant consists of a large stock pebble-bin fed by belt from the breaker station. This bin supplies by means of conveyor belt smaller bins of 8 tons capacity, one to each tube-mill, placed at the feed end of the mills. A bumping feeder shoot actuated by a lever, lifted by a pin in the end casting of the tube-mill, gives the hinged shoot the necessary shock to cause the pebbles in the small bin to move down the shoot and into the large Schmitt feeder. Pebble feeding on this plant, so long as the requisite supply from the breaker station is sufficient, is excellent and practically automatic. The only attention necessary is such as is required on other feeders, such as the well known ore feeders in stamp mills. The feeders are capable of supplying more pebbles than are required by the tube-mills, even when the consumption is high as with the scoop discharge, and the plant is eminently suited to such conditions. The quantity of pebbles is easily measured from the small bins, and experience soon teaches how often to run the feeder, but the reading of the wattmeter is really the best guide for maintaining a uniform pebble-load and crushing capacity of the tube-mill.

A long series of experiments at the Robinson Deep Mine (Table No. 3) confirmed the results obtained at the other mines, and with the ample data thus available the following conclusions may be drawn as to the value of the scoop discharge, and generally the best way to work it.

If it be assumed that the present average duty under ordinary conditions of the standard tube-mill of 5½ ft. × 22 ft. is 120 tons of −90 product per 24 hours, and this is probably as much as can be allowed, the performances recorded in the experiments

* See p. 25 of Report of Consolidated Go'd Fields of S.A., for year ended 31st July, 1914.

TABLE No. 3.—TUBE MILL EXPERIMENTS WITH SCOOP DISCHARGE AT ROBINSON DEEP G.M. CO., LTD.

Trial No. (1)	Duration of Trial. (2)	Diameter of Scoop-Discharge in Inches (3)	Actual Diameter of Tube Mill in Inches. (4)	Revolutions of Tube Mill per Minute. (5)	Tons Sand Feed per 24 hours. (6)	Percentage Moisture in Sand Feed Trap (7)	Mean Height of Pebbles in Mill above or below Axis, in Inches. (8)	Tons of Pebbles in Mill. (9)	Horse Power Consumed. (10)	Grading Analysis Feed.			Grading Analysis Discharge.			Tons of Product per 24 hours. (17)	Consumption of Pebbles per 24 hours, tons. (18)	Horse Power per Ton of −90. (19)	Tons of −90 per Horse Power. (20)
										+90 Per cent (11)	−90 +400 Per cent (12)	400 Per cent (13)	+90 Per cent (14)	−90 +400 Per cent (15)	400 Per cent (16)				
9	5 trials	24	58½	32	367	38·55	+ 4·0	12·63	115	52·05	29·79	18·16	11·32	30·79	57·89	145·97	17·10	0·99	1·00
10	6 ,,	24	58½	32	406	38·32	+ 7·7	14·47	115	48·39	33·20	18·41	16·55	32·10	57·35	157·91	17·40	0·92	1·09
11	6 ,,	24	58½	32	381	40·24	+ 5·0	13·10	115	45·39	34·13	20·48	9·47	30·51	60·02	150·88	18·02	0·96	1·04
12	6 ,,	24	58½	32	429	39·59	+ 5·0	13·10	115	50·36	31·59	18·05	13·50	33·04	53·46	151·83	18·37	0·95	1·05
13	4 ,,	24	59½	32	498	37·66	+ 4·1	13·08	115	53·10	29·70	16·90	18·10	33·00	48·90	159·40	16·80	0·90	1·10
14	9 ,,	24	59½	32	467	41·80	+ 7·5	14·67	115	51·30	30·80	17·90	13·20	33·20	53·60	114·77	15·80	1·00	1·00
15	12 ,,	24	60	32	479	41·50	+ 6·1	16·10	145	60·50	21·70	11·80	24·20	30·30	45·50	146·05	14·31	0·99	1·01
		24	59·0	32	132	10·09	+ 5·6	13·92	115	51·63	30·56	17·81	15·61	31·85	53·11	150·97	16·83	0·96	1·04

where the scoop discharge was used show considerable improvement. The result recorded for trial No. 5 in Table No. 2 of 154·8 tons of −90 for 138·6 H.P. might be considered as a typical one when a scoop is employed and used for a comparison. This result was obtained with a sand feed of 358 tons per 24 hours, a pebble load a little above the axis of the mill, and the sand feed as far as size of particles was concerned was finer than is the case when the stamp mill is crushing through a coarse screen, thus somewhat handicapping the tube-mill. This excellent duty was attained, but as the tube-mill had an internal diameter of 62 in. it is only fair to reduce this duty to the figure that would be obtained at an average diameter of 59½ in. in proportion to its cubic capacity. The duty is thus amended to 143 tons of −90 product, and this is the figure that is comparable with the normal duty of 120 tons of −90 product, the increase being equivalent to 19 , or 5·7 nominal crushing units.[*] Reference to the tables will illustrate similar increases, but one example is sufficient, as the others bear out the point. The Robinson Deep results (Table No. 3) where coarse screening is in use in the stamp mill are an improvement on the Simmer & Jack case quoted. Although the internal diameter of the tube-mill is somewhat under the average, the pebble load is somewhat greater.

Increase of pebble load to a height materially above the axis of the mill is of little assistance since the power consumption and pebble wear increase out of all proportion to the increase of −90 product. Comparison of Trials Nos. 1 and 2 illustrates this point, where the pebble load of 16·08 tons took 20·04 H.P. more to produce only 3·22 tons more than a pebble load of 12·84 tons. This is probably due to the overcrowding of pebbles in the tube-mill interfering with their free and forcible projection.

It will be noted that although the scoop was capable of being run with a diameter of 50 in., the effective diameter was reduced to 24 in., and this is the size of scoop that yields consistently good results combined with reasonable wear of liners and pebbles. With the large diameter scoop the wear is excessive and beyond practical limits.

Moisture in sand feed should be such that the free and independent fall of pebbles is secured and the projection of masses of pebbles and sand is prevented.

[*] See "Rand Metallurgical Practice," Vol. I., p. 136.

As to power required, this is mainly governed by the pebble load in scoop discharge mills as in the present practice of ordinary trunnion discharge, provided that the moisture is in keeping with the quantity of sand feed. Assuming that in present practice one ton of −200 product is produced for each horse power employed per 24 hours, there does not appear to be any saving of power in the use of scoop discharge, as the figures with the latter are much the same. The real advantage is in the increased capacity of the tube-mill, which is such that five scoop discharge mills will do the duty of six ordinary trunnion discharge mills, so that both capital cost and operating charges are reduced.

Small sand feed decreases the tons of −200 product, probably owing to there being insufficient coarse grains near the tube-mill outlet to fully utilize the impact of the falling pebbles, and as increased sand feeds up to even 500 tons do not increase the power consumption of the mill, there is no reason apart from pulp elevation considerations why big feeds should not be given. The most suitable feed of sand per 24 hours as a pulp containing 40 % moisture for a 5½ ft. × 22 ft. mill appears to be about 400 tons. In this connection, a paper, published in *Metallurgical and Chemical Engineering* for January, 1914, by J. H. Haynes, shows that the tons of −200 mesh (linear) product in tube-mills steadily increases with the tonnage of sand fed.

As to the question of why greater duty is obtained with the scoop discharge than without, the reason appears simple and obvious. The scoop lowers the level of the pulp, and falling pebbles strike those temporarily at a lower level, crushing the adhering grains of sand upon them and the liner, instead of falling largely into a mud bath as in present practice. This lowering of the pulp level also explains why the pebble load level may be lowered, and a height little above the axis of the mill appears to be the most desirable.

Whilst utilization of the fact that so large an increase as 19 % of the crushing capacity of tube-mills may be readily obtained is of great value to a plant over-supplied with ore, yet to obtain this result a 150 H.P. motor and wattmeter should be installed for each 5½ ft. × 22 ft. tube-mill, and provision made for transporting and regularly feeding an adequate tonnage of pebbles, as well as an ample sand feed to the tube-mill.

The experimental work extending over a long period carried out on the mines mentioned above owes much of its value to the facilities due to the Superintending Engineer, the Mechanical Engineering Department, and the Mine Managers of the Consolidated Gold Fields; and to the careful and skilled attention of the various Mill Managers concerned in the trials—Messrs. A. J. Herald, W. H. Stout, and J. M. Dixon.

V

SAFETY MEASURES IN THE USE OF EXPLOSIVES.

By Wm. Cullen, M.I.M.M. (Past-President).

The gist of this paper formed the subject matter of a lecture which was delivered by the writer at the last Mining Exhibition in 1914. Some of the points have been elaborated as the result of experience gained since then, and a few omissions have been rectified, but on the whole the paper is much the same as the lecture. My justification for the repetition is that even to-day some of the points are not appreciated. If they were and the necessary precautions taken, accidents would be of less frequent occurrence. Some of these points are perfectly obvious; others are not quite so obvious. I am aware that much the same ground has been covered by Mr. Archbold and others in connection with the "Safety First" movement, and indeed this paper forms part and parcel of the same propaganda, but I hope to go further than he and others have been able to go.

During 1913 there were 251 separate accidents due to explosion on the Witwatersrand mines, which resulted in 223 deaths and injuries to 253 persons. The death rate calculated per 1,000 persons per annum at work underground works out at 1·54.* The classification of these accidents is as follows:—

* Mr. Archbold's lecture, Series II.

	No. of Accidents.	No. of Deaths.	No. of Injuries.
Whilst charging up ...	8	8	9
Handling detonator ...	23	—	23
During blasting operations	45	25	26
Due to fumes	36	51	23
Withdrawing charges ..	1	—	1
Drilling into unexploded holes	115	118	150
Sundry and unknown	23	21	21

I shall endeavour to show that at least a proportion of these is preventable, but I do not propose to follow the foregoing classification rigidly, as it does not lend itself to my method of treatment. Let me also at this stage say how delighted I am to learn that the death-rate for 1914 is the lowest on record. The direct cost of accidents to the mines stands, however, at £130,000 per annum, which is admittedly a very high figure,* but I hope that as a result of persistent effort this heavy assessment will be gradually reduced. Before considering the subject in detail I ought to say that the quality of explosives and their accessories, such as detonators and fuse, supplied to the market is very good, and it is difficult indeed to suggest radical improvement. I shall have to make certain criticisms, but these will not affect the general statement with regard to quality. Indeed, I can go further, and say that after having had practical experience of several other important markets, I consider that not one is so well catered for as the Witwatersrand.

By far the most fruitful cause of accident is "drilling into unexploded charges", but the Rand is not alone in this respect. Now this cause can never be entirely eliminated, but I have the temerity to suggest that it can be considerably reduced. For many years an attempt has been made to attack the problem from the point of view of finding a detector for unexploded charges. Some promised quite well, but the truth is that unless a device is absolutely certain it will do more harm than good. In other words, miners will rely on the device and become careless in consequence, which is only human nature all the world over. No! the problem must be tackled from the other end, and fortunately it is comparatively simple. Let me at this stage say that I have had some practical experience in mining work and considerable experience in blasting problems, for over twenty years, and that in the compilation of this paper I have had the opportunity of drawing upon the experience of Mr. W. Waters, who has had a very wide experience in many mining countries. During the entire period of our work neither of us has ever had a partially exploded charge or a partial misfire. The following are, in my opinion, the main causes for unexploded charges. There are, of course, others, but they are not so important:—

(a) Neglect of precaution to securely fasten detonator and fuse to the primer cartridge.
(b) Cut out shots.
(c) Insensitive explosive.
(d) Bad charging up.
(e) Damp detonators.

Now I hold that of these five causes (a) "neglect of precaution to securely fasten detonator and fuse to the primer cartridge" is by far the most important. Indeed I would hazard a guess that it is probably responsible for more mishaps than all the others put together, and yet the neglect of this precaution is a violation of Clause 106, Sub-Section 19, of the Mining Regulations, which reads as follows:—

> "A ganger or miner who is the holder of a blasting certificate shall, in making up a charge, after the fuse with attached detonator has been inserted in the primer cartridge, securely fasten it to the said cartridge so that the fuse and detonator cannot be inadvertently withdrawn."

In other words, what I say is that in many cases, probably in the majority, the fuse with detonator properly crimped as provided in Section 18 of the same Regulation, is *not* securely attached to the primer cartridge. Indeed, it is not attached at all, but merely stuck in. I know that in the majority of our mines the regulation has not been observed for years. Is it to be wondered at therefore that during the charging of the hole, the fuse and detonator become detached from the primer cartridge, and that an unexploded charge is left? The wonder would be if it were not. If the explosives used on these fields were of the nature of the American so-called "straight" dynamite, mishaps of this kind would be far more common. It is simply because the explosives in use here have a gelatinous or sticky consistency, and therefore "cling" to the detonator, that the two do not part company more frequently. In the coal mines where dynamites are almost exclusively used they must part company very frequently. Miners—I shall not put the status any higher—when

*Mr Palmer Carter's Address to Mine Manager's Association.

their attention has been drawn to this violation generally say "we have never done any other thing all our lives", or, "we have not the time," or, "to securely tasten costs money," but really none of these answers is worth rebutting. The only thing to do is to insist on the literal observance of the regulation. In this connection it has recently been suggested that if a primer of dynamite be substituted for the ordinary gelatinous primer, the consumption of explosives would be reduced and the detonation made more certain. I express no opinion about the first claim, but with regard to the second, I am very certain, in the light of what has just been said, that precisely the opposite effect would be produced. A practice which is not so common, but which is fairly general, is to break the primer cartridge along its shorter axis, insert the detonator with fuse attached and close it up again. I have been informed, but have not seen it myself, that some miners even do this to the second or the third cartridge, but what I am given to understand is not only done but advocated, is to put the detonator with fuse into the bottom cartridge, which thus becomes the primer cartridge. Now I shall return to this later, but all I shall say about all these practices at present is that they are the height of criminal folly, and should not be tolerated. The underlying idea is sound enough of course, viz., that if the bottom cartridge becomes the "primer", no unexploded portion of the charge will be left in the hole. I thought at one time it would have been possible to make a fuse with a coating sufficiently fireproof to prevent the ignition of the explosive charge, but even though this were feasible, the practice must stand condemned. I am also informed by friends that when this practice has been followed in the "States", the mining results have been poor, which is quite in accordance with theory. I have considered this point at considerable length, because, as already said, I am convinced of its great importance, and I would earnestly implore all concerned to see that Sub-Section 19 of the Regulation is strictly carried out. Before I leave it might I suggest that the mining effect could be settled once and for all by making the bottom cartridge the "primer" and firing it by an electric detonator. I should like also, before leaving this branch of the subject, to direct attention to the so-called "Nowag"* clip which won the highest award at the last Mining Exhibition (exhibit specimens). This clip is meant to save the "tying", which is admittedly slow work, and this it does. It has some defects, which, however, are not serious, and I am certain that it is a practicable device, which must come into general use if the regulations with regard to tying are insisted on. The idea has been handed over to the industry to exploit as they like, free, but so far beyond one enquiry made to me I think nothing whatever has been done. My Company does not propose to make or to sell the clips, but a few thousands were ordered in Europe prior to the outbreak of war. Naturally they have not arrived, but when they come we trust that they will receive a serious trial.

We must next consider (b) which, as you will remember, is "Cut out shots". In the ordinary course of mining it is difficult altogether to obviate "cut out shots", and I have no panacea to advance except this, viz., to use the best and most regular burning fuse only, and to allow for a longer time interval between the different shots. Even under the most favourable circumstances, however, there are bound to be "cut out shots", and although, in my opinion, the employment of delay action electric fuses would reduce their number, this style of blasting does not seem to catch on. I have already said that no detector of a cut out shot has been found in practice to be altogether satisfactory, and certainly I am not in a position to recommend any one of them. The only way to minimise accidents is to faithfully observe the Mining Regulations.

"Insensitive Explosive" I have designated as cause (c). Now I wish to be perfectly frank upon this matter. Even to-day we do not altogether understand sensitiveness, but we do know that some explosives are sensitive while others are not. It is generally understood among explosives people that on prolonged storage, particularly under tropical or semi-tropical conditions, gelatinous explosives become insensitive. This is not universally true, but, broadly speaking, the statement is correct. On the Rand, and throughout South Africa generally, explosives are used up very rapidly so that stocks can suffer little from this cause, but they may suffer, particularly if they are a little insensitive to start with, which is not at all uncommon. In speaking of sensitiveness, I mean of course sensitiveness to the impulse of a detonator. Supposing we had two 6 ft. holes, one charged with insensitive and the other with sensitive gelatine, the chances are that the detonation of the insensitive

* This *Journal*, February 1914.

would be incomplete, and that a plug or even two might be left at the bottom of the hole. The sensitive explosive would, of course, be completely detonated. Moreover, an insensitive explosive responds very quickly to bad charging. What I mean is this, that supposing from some cause or other—say a badly drilled hole—it is difficult to get the cartridges into actual contact with one another by means of the tamping rod: in other words, there is an actual air gap between individual plugs. A good sensitive explosive will easily bridge the gap on detonation, but an insensitive one may not do so, and the result is obvious. You may ask how sensitiveness is attained and regulated, but my reply is that this is a very valuable trade secret. This much I may say, however, that the use of a larger detonator is to an extent a corrective, therefore on these fields I strongly recommend the use of No. 8. Although it is well known that the bulk of the explosives used on these fields is in a perfectly sensitive condition, and hence can be perfectly detonated with a No. 6 detonator, this is not true of the whole of the explosives used, and hence an all round use of a No. 8 detonator would greatly minimise the possibility of incomplete detonation in all instances.

"Bad charging" (d) has already been referred to under insensitiveness. An air gap is occasionally quite unavoidable, and I am of the opinion that sometimes sufficient care is not always taken to squeeze the cartridge home. To get the best results the cartridges should be in actual contact. Let me repeat again that during all the experience of Mr. Waters and myself with holes actually charged up by us we have had neither a misfire nor, let me add, have we ever had a partial explosion. We may have had an abnormal run of luck, but I think not, and that our success is due entirely to the exercise of common sense. Nowadays one hears little of interposed cartridges of sand tamping, a practice which is very wrong. I have not come across it for years, but in any case it is specifically barred by Regulation 106, Sub-Section 22. Some ten or eleven years ago somewhat elaborate experiments were made at the Langhaagte Deep with interposed tamping, but misfires could not be got. I now realise that the explosive then used was sensitive enough to overcome the "inert gap," but it would be different in many cases to-day.

I have now to consider the effect of damp detonators (e). It is, of course, well known that detonators do deteriorate very badly when stored in a damp atmosphere, and manufacturers are insistent on this point. I assume that everyone connected with mining knows how detonators are packed, but for the sake of those who are not so connected, I may say that the top portion of each detonator is filled with coarse sawdust, which has to be removed by tapping before the fuse is inserted. This is not always effected without patience, and a grain or two of sawdust left behind will easily prevent the spit of the fuse reaching the detonating composition. This may in turn cause a misfire, with all the attendant dangers. Now, I do not wish to minimise the danger of storing detonators underground, but I do believe that with the class of detonators supplied to this market these dangers are exaggerated. Mr. Archbold refers to some experiments which were carried out at Leadville, U.S.A.,[*] and I have taken the liberty of quoting the results in detail:—

Fresh detonators.—Complete detonation

24 hours underground.—Incomplete detonation, *i.e.*, a portion of the charge left in hole unexploded.

48 hours underground.—Incomplete detonation, with red fumes showing that some of the charge had burned.

72 hours underground.—Cap exploded without detonating the charge at all.

144 hours underground.—Cap did not explode.

Now, no particulars are given about these detonators or about the explosives, but these results were so remarkable that I determined to imitate them with Nobel's ordinary detonators as supplied to this market, and Modderfontein blasting gelatine. I strongly suspect that in the experiments quoted by Mr. Archbold the sawdust had been removed from the detonators, so notwithstanding the fact that this would be highly unpractical I decided to do the same. Detonators with and without their sawdust were exposed for varying periods to an atmosphere saturated with moisture at a temperature round about 75° F. It would take too long to detail the experiments now, and, as a matter of fact, they are not quite complete yet, but quite sufficient is known to warrant certain definite conclusions.

Detonators without sawdust.—After three days' exposure the detonators behaved quite normally. On the fourth day there was a weakening, which gradually increased. After the third it would have been unsafe to use them.

[*] Mr. Archbold's lecture, Series II.

Detonators with sawdust.—Right up to the tenth day no deterioration could be detected, but from the fourth day onwards great difficulty was experienced in removing the sawdust.

Now, the effect of sawdust remaining in the detonator has already been referred to, but the experiment showed clearly that this and not the dampness was the real danger, so that it really comes back to the same thing, viz., that it is unsafe to store detonators underground for longer than a very limited period. It has been suggested that in order to overcome this danger detonators should be packed in smaller boxes, containing, say, 30 to 50, instead of 100, and that only the approximate number required should be taken below. I do not think that this would be any real solution of the difficulty, quite the reverse, but I offer one which would materially help. At present the miner has under his control a box containing explosives and another box containing fuse and detonators. No special effort is made to protect the detonators contained in these boxes, so they remain exposed to damp air for indefinite periods. It would be both simple and inexpensive to make it compulsory to have a special air-tight box containing, say, two boxes of detonators permanently fixed inside the box which is intended to contain the fuse and detonators only. Stored in this manner detonators should keep in perfect condition for months. I submit a specimen box. It may be objected: what is the use of such a box if the miner will insist on forgetting to close it? Well, it could if necessary be provided with a spring, but this has its dangers, and besides even it it is no real improvement if a man has really made up his mind to be careless.

So far my remarks on sensitiveness have had a clear and definite meaning, but there is another form of the same thing to which I wish to direct your very particular attention. If you refer back to the classification on page 2 you will find that accidents involving 8 deaths took place during the process of charging up. No doubt a proportion of these is due to downright carelessness, but some cannot be explained on such grounds. Only the other day on the East Rand an accident of this kind happened. The other kind of sensitiveness which I have in mind is due to exudation of nitroglycerine. Normal, well made gelatinous explosives stand a great deal of rough handling, and they are fairly impervious to friction when enclosed in a wrapper. If this were not so the accident rate on these fields would be appalling. When they exude nitroglycerine, and the nitroglycerine gets to the outside of the wrapper, which is not at all uncommon, it is a very different story. Not only is the explosive in this state sensitive to friction, but it is dangerously so. How is exudation caused, it may be asked? Well, it may be caused in a variety of ways. Some causes are incidental to manufacture and some to transport, storage, and climatic conditions. No gelatinous explosives should show exudation three months after storage, otherwise there is something wrong with the manufacturing end. Long journeys by rail or wagon cause, or at least help on, exudation because the irregular vibratory or shaking effect exercises pressure on the lower cartridges of the packet or case, and all gelatinous explosives tend to exude nitroglycerine when gentle pressure is exerted. Then the rapid alterations of temperature which are experienced in this part of the world assist in the same direction. All of these causes can to an extent be overcome by increasing the percentage of collodion cotton, but it is just in respect of this very necessary adjustment that so many manufacturers fail. Explosives which show exudation are not allowed into certain markets; per contra, I have seen explosives taken into use almost swimming in nitroglycerine, and yet many old miners like their explosives to be in this state, for then they are "sure". Now, I could elaborate this point, but I shall content myself by saying that explosives which show exudation on the outside of the wrapper should never be used. They should simply be returned to the maker. I am certain that exudation has caused many unexplained accidents. Just imagine charging cartridges smeared over with a thin film of nitroglycerine into a tight hole which contains grit in abundance. The conditions are simply ideal for causing an explosion, and it is precisely for this reason that I deprecate the use of tamping rods sheathed with copper or any other metal. Tamping rods should be of wood alone. But the danger does not end with the charging. Exuding explosives may leave a smear of nitroglycerine along the hole, and they may even leave a smear of the explosive itself. Naturally, the risk is increased if the cartridge is broken across to insert the detonator. When the burning fuse gets down to the hole, conditions may thus have been created which will give not a bad imitation of a so-called running fuse. This is not theory, but fact, as has been demon-

strated. I have already referred to the danger of making the bottom cartridge—or for that matter any other cartridge but the top one—the "primer", but, in order to set the matter at rest once and for all, lengthy series of experiments were carried out. It will be remembered that I characterized the practice as criminal folly, and I may now state my contention, which is that ordinary time fuse, no matter how good the quality, will tend to ignite the explosive—along which it lies in parallel—before the detonator is reached. Now, at this time of day it is not necessary to say anything about the terrible effects of ignited explosives. The subject of gassing has received a great deal of attention both here and elsewhere, and in the classification already referred to 23 deaths were due to this one cause. It would take too long to detail the experiments, so I shall merely give the conclusions, which are very definite. It must be remembered, however, that the explosives supplied to this market are wrapped in two kinds of wrappers, viz., parchment and paraffined paper. The wrapping, for most of the cartridge, is double, but slight differences in the diameters of the cartridges may make this wrapping treble or single ply at the overlap. In order to imitate mining conditions as nearly as possible, the cartridges in some cases were charged into a steel tube of suitable diameter and tamped with sand in the usual manner. It was found

(a) That ignition seldom if ever takes place from side spitting of the fuse unless it has been very badly abused. The amount of abuse which fuse can stand will be referred to later.

(b) That when ignition does take place it is caused by the heat of the smouldering fuse. Smouldering always takes place, no matter how good the fuse, and its effects are most marked about 2 in. behind the point where the powder core is burning.

(c) Unwrapped explosive in direct contact with fuse is always ignited. This indicates the great danger of breaking cartridges.

(d) Paraffined wrappers offer more resistance to the igniting effect of fuse than parchment.

(e) When the contact of the fuse is at the single ply ignition takes place more readily than at double ply, *i.e.*, at the overlap.

(f) Exudation enormously increases the risk of ignition.

The conclusion of the whole matter is this, that the Mining Regulations as they stand are all right, and no departure from them ought to be tolerated, even though the paraffined wrappers produce rather more carbon monoxide than the others, but this is now of much less significance than formerly, owing to the greatly improved ventilation. Of course, they were originally adopted by the Transvaal factory because there was really no choice, as the parchment of those days dried and crumbled away owing to lack of moisture in the atmosphere. Nowadays parchments are procurable which are not much affected by atmospheric conditions, but notwithstanding this I prefer to adhere to the paraffined wrappers, because I have formed the opinion that they facilitate the charging up, and they certainly prevent exudation.

Accidents not infrequently take place through attempting to force a cartridge into a hole which is in reality too small for it. When such a cartridge is in an exuding state the risk of an explosion is very great indeed. In this connection the Modderfontein factory has for a number of years put a number of 1 in. size cartridges into every 10 lb. packet. The smaller size cartridges are of course used for the bottom of the hole. The practice, it is hoped, will soon become universal. When on the subject of sizes might I suggest that it is high time we adopted definite standards of length and diameter of cartridges. The matter is not so simple as it looks, but standardization is badly wanted.

I next quote in full the text of a note handed me by Mr. W. Waters:—

"I frankly confess that the dangers of re-charging and re-blasting holes or sockets of holes, while the rock is hot from the original blast are not fully appreciated by miners. When a cut has completely failed to break, the miner knows that time must be lost and resigns himself to the state of affairs. He, in this instance, usually waits long enough for the rock to cool down, this cooling down being assisted by the hose-pipe water. What I wish to call special attention to is the common practice of cleaning out the bottom of a cut—which may be broken fairly well, but not quite clean. The miner in this instance usually re-charges either one or two of the cut hole sockets with from two to four cartridges, and lights up same when firing the "round", these socket holes being arranged to explode before the 'easers'.

"As mentioned above, when the cut completely fails at the first blast, the miner re-

signs himself to the loss of time. When, however, he finds that the cut has come fairly well at the first blast, and only requires a few plugs to clean it out, he is often in a hurry to blast his round and thus, we fear, re-charges the cut sockets before the rock has sufficiently cooled down. The exploding heat required for blasting gelatine and gelignite is roughly about 100° F., and it may take some seconds or even minutes before the explosive in these socket holes heats up to the exploding point. Thus the miner may be lighting up his last holes, namely, the 'lifters', or perhaps be just walking away from the face when the cut sockets prematurely explode. The usual reason assigned for such an accident is a 'running fuse'. I think otherwise."

Cartridges, as we all know, are frequently found among the broken rock, and this is particularly noticeable in the coal mines, for a reason which I have already referred to, but even at the risk of repetition I shall give it again. Dynamites, owing to their want of "oiling", do not hold the detonator well, so it is liable to get detached from the primer. This is one of the principal reasons for the finding of unexploded cartridges, but another is "cut out" shots, which, as I have said, are difficult to avoid, unless the fuse is extraordinarily regular. Now, some years ago certain manufacturers introduced coloured wrappers in order to render the cartridges more conspicuous. It would be very interesting to know how the claims for the coloured wrappers work out in practice. Of course, in coal mines it is very difficult to pick out unexploded cartridges undergound, consequently a large number are caught on the surface belt. It is generally known that nitroglycerine explosives freeze at 8° C. and when in that state they are very dangerous indeed, and will easily explode when touched with a pick or shovel. Fortunately for us here mines are all warm, but in colder countries, particularly in Canada, accidents from this cause are very common, notwithstanding the fact that explosives are invariably thawed in an approved apparatus before use.

The perfect "cheesa" stick has still to be made. A good many makes are on the market, but there is not one which answers all the requirements of the practical man. What those requirements are most of you know, and they are very difficult to put into words. I can speak feelingly on this point, for I have been experimenting with "cheesa" sticks off and on for years in conjunction with the factory staff, and only within the past few weeks—when materials are unobtainable—has success been achieved. I shall attempt to demonstrate by experiment what I mean, but a reliable stick should have the following properties:—

(a) It should burn with a good strong flame without ejecting bits of incandescent material.

(b) The flame should be strong enough to keep alight when the charges detonate.

(c) It should not be put out by a few drops of water.

(d) It should not go out when dropped from, say, a height of 3 ft. on its long axis.

(e) It should comply with the Government specification with regard to nitrous oxides.

Now, these tests do not appear to be very stringent, but they have so far proved very difficult of attainment, and failure of any one may lead to serious accidents.

I shall conclude with a brief reference to fuse, on which subject most of us consider ourselves expert. Undoubtedly a good many accidents are caused by fuse, but on the whole the fuse now being supplied to this market is of excellent quality, and is fairly free from such gross defects as "interruption of powder core" and "side spitting". It is also sufficiently waterproof for all our local conditions, and some of these are extraordinarily severe. Now, just at this stage let me say that though I have had hundreds of thousands of coils tested under all sorts of severe conditions, I have never come across a so-called "running" fuse or anything the least like it. In order to see if burning speed could be influenced by harsh treatment, a certain well known brand of fuse was subjected to the following tests. These tests were carried out specially for this paper:—

(a) It was hammered almost flat.

(b) It was hammered almost flat, turned on its edge, and again hammered flat.

(c) It was drawn into numerous tight knots.

(d) It was drawn into numerous tight knots, hammered flat and untied.

(e) It was twisted strongly in the direction opposite to its natural twist.

It was found that in no instance was the burning speed affected in the slightest degree. I have not carried out these experiments to show that fuse can with safety first of all be used for tying up bundles,

'picks and jumpers, and then afterwards be used as fuse, but to show what an amount of abuse it really can stand—but the practice is dangerous. Fuse should be treated with the greatest care, for all varieties cannot withstand these drastic tests, but even the ordinary practice of tying up "miner fashion" puts a very severe strain on it. It is not generally realised how severe on the fuse covering our climate is. Owing to the absence of moisture in the atmosphere, the waterproof coating tends to become dry and crack. This of itself tends to side spitting, but if it is used as parcel twine before it is used as fuse this tendency will become accentuated.

The general test to which fuse is subjected is to immerse a coil in a bucket of water with the ends projecting, leave it there from two to twenty-four hours, then see whether it will burn through. Most fuse will survive this ordeal, but the only way to really try it out is to subject it to tests which are the exact counterpart of practice. This may appear delightfully vague, but practical men will know what I mean. It is not generally known that fuse burns slower at this altitude than at sea level, and I have often wondered whether miners unconsciously allow for this in the deep levels.

Now, there are many points on which I have not touched at all: some because they were too obvious, and others because they were too speculative, but all those I have discussed I consider important, and I trust that their reiteration will lead to the exercise of greater care and attention, with a consequent reduction in the accident rate. Progress in this direction is lamentably slow, but it need not necessarily be discouraging.

I wish to express my thanks to Mr. W. Waters, who though not associated with me as I should have liked in the writing of this paper, has given me the benefit of his wide, practical experience.

OBITUARY.

The deaths of the following members are recorded with much regret.

Mr. George Australia Darling, a foundation member of the Society, died on Sunday, the 21st March, at the age of 56. Mr. Darling was well known on the Rand, having arrived in Johannesburg in August, 1889.* He received his first appointment at the old Salisbury G.M. Co.'s battery on the Natal Spruit, and was afterwards associated for nearly a year with Mr. S. H. Pearce (Past-President of the Society) who was then the chief chemist and assayer to the S.A. Metallurgical Co. In November, 1890, Mr. Darling joined the Gold Recovery Syndicate and received an early insight into the working of the cyanide process from Mr. Alfred James at the experimental cyanide plant erected by the Cassel Gold Extraction Co. on the Salisbury Mine. As a result of his experience there Mr. Darling was placed in charge of the cyanide plant erected by the Gold Recovery Syndicate on the Robinson Mine, and when the plant was taken over by that Company on July 1st, 1891 he remained and became the first Cyanide Manager of that Company, a position he held until about 1906. Mr. Darling claimed to be the first to prove the success of the cyanide process, working on a commercial scale, and in the *Journal* for January, 1909, gave an interesting description of his early work in that connection on the Rand. After leaving the Robinson Mine, he was successively engaged in cyaniding and assaying on the Lisbon Berlyn Mine, the New Goch Gold Mines, the Spes Bona Tribute, and at the New Kleinfontein Mines, where he had been employed for the past three years. He had been ailing for about a year before his death. Mr. Darling was a member of the Council of the Society from 1889 to 1903, but latterly ill-health had prevented him from taking any active interest in the Society's work. The funeral took place at the Kleinfontein Cemetery on the 22nd March, at which the Society was represented by the Secretary. Wreaths were sent, in addition to those from his relatives, and friends on the mine, by the Society, the Transvaal Chamber of Mines, and the Directors of the Robinson G.M. Co., Ltd.

Lieut. William Godfrey Froude, who joined the Society in April, 1912, was well known on the Rand, having been employed on various mines as a mine captain. Reasons of health compelled him to leave Johannesburg, and he went North for a London mining syndicate undertaking exploratory work in Manicaland and Rhodesia. Mr. Froude subsequently took charge of the Mali Dyke Mine in the Pilgrims Rest district, which closed down owing to the war. Mr. Froude joined the Imperial Light Horse as a Lieutenant, and was killed in action on the 25th November during the operations against the rebel Kemp in the

*See this *Journal*, Vol. IX., Jan., 1909, pp. 227—230.

Western Transvaal. Lieut. Froude especially distinguished himself during the South African War, on the Bechuanaland Border, being mentioned several times in despatches and receiving the D.C.M.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

STORAGE OF COAL. "From an investigation recently made by the United States Bureau of Mines, we learn that the loss of calorific value is really much less than has been believed, though it differs materially with different coals, different climates and different methods of storage. Freshly mined coal suffers the greatest loss and hence loss is most discernible in the first few months after storing, when a good deal of methane is evolved. Loss in heat value varies from 0·16 upwards and may be taken to average from 1 to 3% in one year, after which further losses take place slowly, with slackening and crumbling of the coal, chiefly near the surface. But there are other more serious effects of storage due to oxidation, which may be followed by spontaneous combustion. Inflammable gases escape from the coal surfaces, absorb oxygen from the air and so become heated. The process is accelerated when fine coal is stored because then a larger surface is in contact with the air; also when the coal stack is under the influence of even a moderate degree of external heat, as from the sun or from adjacent steam pipes, etc. In such cases not only is there risk of spontaneous combustion of the coal but even of explosion of the escaping gases. As precautions the following steps are recommended. Freshly mined coal should be seasoned for six weeks before being stored; the stacks should never exceed 10 ft. or 12 ft. in depth and no point within the stack should be more than 10 ft. from an air-cooled surface. As little dusty coal as possible should be stored, it being advisable to screen it when on the whole the supply contains much dust. Fine and lump coal should be well mixed together in storing in order to have as little air space as possible within the stack. As the idea is to have as little oxygen as possible in contact with the coal within the stack the old idea of ventilating the stack by means of pipes must be considered as exploded; further, care must be taken to prevent infiltration of air through brickwork, timber, or porous foundations such as a coarse cinder. As the storage of fine coal is unsafe, any fine coal required for burning should be broken up only when required for use; and it is also recommended that the coal stack be re handled about once every two months. Alternate wetting and drying of the stack must be guarded against as far as possible. Submerged storage is recommended both because loss in calorific value is minimised and spontaneous combustion is eliminated. The objection, of course, is the absorbed moisture, but it the coal be allowed to drain for a few hours before being passed on to the furnace the effects of the moisture are more than counterbalanced by the reduced loss in calorific value. Worked on a large scale with concrete vats and mechanical boiler feed the method will be found to be less expensive than stacking in the open. A method has been tried and recommended of only partial immersion, it being found that the exposed coal absorbs all the moisture it needs to make it safe by capillary attraction."—*Indian Engineering*, Oct. 3, 1914, p. 193. (J. A. W.)

PRODUCING BORON.—"Boron is one of the rarest elements, but its compounds are abundant. It is made by bringing a volatile boron salt with hydrogen gas into an electric arc, where they are heated to a very high temperature. The salt is reduced by the hydrogen to 'metal,' and the vapours produced are chilled before they have a chance to recombine. It is the same operating principle as is used in the fixation of atmospheric nitrogen in Norway. This boron is being put on the market for use in casting 'conductivity' copper. This is one of the most recent productions of the electric furnace."—*Mining Science*, —Dec., 1914, p. 20. (H. R. A.)

THE ACCURACY OF NEUMANN'S METHOD FOR THE ESTIMATION OF PHOSPHORUS. — "Neumann (*Z. physiol. Chem.*, *37*, 115 (1902); *43*, 32 (1904) modified the usual titration of the phosphomolybdate precipitate by boiling off the NH_3 before titrating the excess of NaOH with 0·1 *N* HNO_3. Wardlaw finds that neither Neumann's nor any one of the other modifications give correct results. The amount of P_2O_5 calculated from acidimetric titration is in every case too high, the error increasing with the amount of P_2O_5 present (with 11·25 mgm. P_2O_5 present the error averages about +1·8%, with 22·3 mgm. about +3·4%, with 44·5 mgm. about +13·2%). Analysis of the yellow precipitate gave 12·75 MoO_3 instead of the $12MoO_3$ usually written in the formula. (Hissink and van der Waerden, *Chem. Weekblad*, 1905, 179, found 12·65.) The error due to this excess of MoO_3 carried down with the precipitate does not depend on the rate of precipitation nor upon the time of contact between the precipitate and the mother liquor. The error cannot be reduced by lowering the temperature of precipitation since this leads to incomplete precipitation. The article contains a good bibliography."—H. S. H. WARDLAW, *J. Proc. Roy. Soc., N. S. Wales*, *48*, Part I., 73-93. *Chemical Abstracts*, Nov. 10, 1914, pp. 3510-1. (J. G.)

DETERMINATION OF PHOSPHORUS IN PHOSPHOR-BRONZE.—"Previous methods being long and unsatisfactory, the following method was devised: Dissolve 1 gm. of bronze in 10 cc. concentrated HCl and 5 cc. concentrated HNO_3 in a tall-form 300 cc. beaker. When solution is complete, add 150 cc. of hot H_2O and 10 cc. of $CuCl_2$ solution (1 gm. $CuCl_2$, 25 cc. of HCl, 200 cc. H_2O), then solution of NH_4OH (1:2 of H_2O) from burette with constant stirring until solution has a greenish blue cast, requiring in all from 25 cc. to 35 cc. Now add 4 cc. AcOH and boil 5 or 10 minutes; remove from hot plate and let precipitate settle (requires several minutes). Siphon or decant off as much of the clear solution as possible, add 100 cc. of hot H_2O and again siphon; repeat this washing six or eight times. With careful manipulation all but about 30 cc. of the liquid can be decanted each time without disturbing the precipitate. After sufficient washing add 15 cc. of HNO_3 and 3 cc. of HCl and heat till precipitate is in solution, cool, add 5 cc. (1:1) NH_4OH, and precipitate the P as usual with $(NH_4)_2MoO_4$. The P may be precipitated in duplicate on three samples within an hour. Sn is determined on a separate sample either electrolytically, by titration, or from the combined oxides. Excellent results reported."—E. W. HAGMAIER, *Met. Chem. Eng.*, *12*, 524-5.—*Chemical Abstracts*, Oct. 10, 1914, pp. 3279-80. (J. G.)

METALLURGY.

ELECTRIC FUME PRECIPITATION.—"At the Garfield Smelting Company's smelter in Utah (U.S.A.) it was found that a considerable amount of lead escaped from the stacks of the converter plant, and experiments were made to recover this by means of the Cotrell electric fume-precipitation process. This process consists in general in the conversion of high-tension alternating current at about 30,000 to 60,000 volts into an intermittent continuous current of the same potential and the application of this current to electrodes in a flue conveying the gases to be treated. The electrodes consist of a series of narrow metal plates, spaced regularly 5 in. to 12 in. apart depending on the voltage, between each of which is stretched a fine wire forming the discharge electrode. The silent or glow discharge which takes place from this wire causes all dust or condensed fume or moisture to be deposited on the plates. The tests at the Garfield plant showed that pipe electrodes of about 5 in. diameter were preferable to plate electrodes and that if the temperature of the gases at the electrodes could be kept under 100° C. practically complete clearance of the smelter gases could be obtained. Further, it was shown that by treating the gases at different temperatures in a series of such plants it might be possible to obtain fractional precipitation; for example, the bulk of the lead in one plant and the arsenic, sulphuric acid, etc., in a second. The Garfield Company have installed seven units to treat 250,000 cub. ft. gas per minute. The electric equipment includes seven complete motor-generator rectifier sets and transformers, and switchboard panels to control them. Each set consists of a 30 h.p. 250 volt direct-current motor, direct connected to a 20 kw., 60 cycle, 220 volt, single-phase generator; the alternating current is stepped up from 20,000 to 30,000 volts by means of a 20 kw. transformer and is then led back to the rectifier, which is on the same shaft as the motor generator." —*Indian Engineering*, Nov. 7, 1914, p. 281. (J.A.W.)

LARGE VERSUS SMALL STAMPS.—"For some years the tendency in stamp milling has been toward the use of heavier stamps, and while it has not come into extensive use in this country, the heavy Nissen stamp has found much favour in South Africa, and in Cornwall as well. Mr. Algernon Del Mar describes the large steam stamps which have long been standard practice in crushing the native copper ore, and raises the question whether they may be used to advantage in stamp milling gold ores.

In the first place, we must take it for granted that a stamp is an uneconomical crushing machine. The stamp is not intended, and should not be used, for crushing any ore except ore in which native metal is scattered through brittle rock. For that purpose it is the logical machine, reproducing on a large scale the method anyone would instinctively use to free a mass of metal from adhering rock; namely, the striking of blows which crack open the brittle gangue and leave the ductile metal free. No matter how superior their mechanical efficiency, crushers which work by trituration or pressure are not suited to this work, as has been proved by experiment in the Lake Superior district. The stamp is therefore the proper machine for the crushing of gold ore and metallic copper ore, and with this as a major premise the characteristics of different types of stamps may be critically considered. Natural impulses are often safe guides to follow. Anyone would naturally employ a large hammer to drive a large nail and a small hammer to drive a small nail. If the use of a large stamp, striking a heavy blow, for crushing ore which is fed to it in small pieces, and the use of lighter stamps when the feed is finer is critically considered it will be found to be logically sound. The problem is not so simple as this, however. The use of large steam stamps for crushing gold ore can not be put on simple efficiency basis with their use for metallic copper ore. For many years it was the custom in gold milling to catch as much gold as possible inside the mortar, and amalgamated plates and grooved liners were used for this purpose. The problem in stamping was not simply to crush the quartz, but to crush as much as possible without scouring the inside plates. Mr. T. A. Rickard, in his 'Stamp-Milling of Gold Ores,' has lucidly discussed the effect which this and other considerations in gold milling has had on the design of the mortar and the force and speed of the stamp blows. In copper milling, on the other hand, provision must be made for the large masses of native copper, which, if allowed to remain between the stamp and die, would uselessly consume large quantities of energy in being pounded into flat masses and gradually broken into fragments which would pass the screen. This involves the use of a roomy mortar, with spaces into which this copper can find a retreat, which is the probable reason for the otherwise illogical use of a die smaller than the stamp shoe. The hydraulic discharge for allowing the copper to escape from the mortar box has greatly helped to solve this problem, and it may be that the mortar now employed on the steam stamps is too roomy to give the maximum efficiency of discharge. If any studies of this feature have been made they have not come to our attention.

As long as amalgamation was the *sine qua non* of stamp milling of gold ores, the steam stamp had another disadvantage in that it is necessary to lubricate the steam piston and oil was constantly finding its way down the stamp stem into the pulp, a proceeding which the millman regarded with horror. Much of this can be prevented by care, however, and now that the catching of gold on the plates is not so important a consideration, so long as it is caught before the tailing escapes from the cyanide plant, the efficiency of the stamping operation is more of a consideration than it once was. It may therefore prove to be more economical to give stamp mills, in gold practice, a coarser feed, and use a steam stamp to strike a harder blow. Unquestionably the stronger blow is the better for the coarser feed. The trouble with the steam stamp, however, is that the length of the stroke of the steam piston is necessarily irregular, depending upon the amount of ore on the die. Thus the steam escapes at the end of the stroke with only part of its energy utilized, thus producing poor steam efficiency. In order to overcome this, the steeple-compound stamp was devised by Mr. Bruno Nordberg. Even this only partly overcomes an inherent difficulty. More recently Mr. F. L. Bosson has brought about the use of a low-pressure steam turbine to utilize the exhaust steam from the stamps, mixing the flow with boiler steam if necessary. This gives a better net total of efficiency, but detailed figures showing the gain have not yet been made public. This arrangement has already become the standard practice of the Calumet & Hecla, Copper Range Consolidated, and

other large Lake Superior companies, though only a few steam turbines have so far been installed. It is to be hoped that detailed studies of the mechanical efficiency of this combination may soon be made public, so that the managers of large gold mills may have the necessary data on which to base conclusions as to whether large steam stamps working in conjunction with steam turbines would yield them a better net efficiency than the present practice of using heavy gravity stamps.

If gold mill operators are prepared to abandon the use of the stamp as an amalgamating device, and will send the stamp-mill pulp directly to tube mills, passing the tube mill discharge over amalgamated plates before sending it to the cyanide plant, the question may be raised as to whether it is necessary to retain stamps for the preliminary crushing of the tube-mill feed. The verdict of experience seems to be that the tube mill is an excellent crushing device to free gold from its adhering gangue, the impact of the pebbles furnishing a sufficiently heavy blow for the comparatively small particles with which the tube mill is fed. The logic of the situation would then seem to be that the crushing of the tube mill feed should be done by the most efficient machine or machines, regardless of their principle of crushing action. This scheme of milling is about to be put into practice in the new Large mills of the Alaska Gold Mines, and publication of the results of operations there will be awaited with the keenest interest by mill operators everywhere." —*Mining and Scientific Press*, Oct. 3, 1914, p. 511. (G. H. S.)

COATING METALS. "Iron and steel articles are provided with a rust-preventing coating, consisting of a combination of iron with nitrogen, by heating them to a temperature above 400° C. in an atmosphere of dry or moist ammonia, or chemical combinations, organic and inorganic, which, on heating, liberate ammonia or products of decomposition of ammonia. Such bodies are, for example, ammonia salts, hydrazine, and combinations thereof, amines, imides, and other nitrogen compounds. The articles to be treated may have a layer of scale thereon, or may be previously cleaned, and the surface may be smooth or etched." H. HANEMANN and E. HANAMAN.—*Chemical World*, Nov., 1914, p. 312. (J. A. W.)

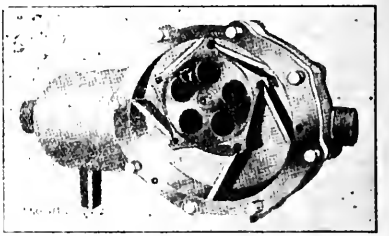
MINING.

ORIGIN OF SAND FILLING. "The whole subject has been discussed unfortunately and authoritatively in *Bulletin* 60 of the United States Bureau of Mines. From that, confirmed by private communication from Mr. R. V. Norris and Mr. William Griffith, it appears that the distinction of developing the system belongs to the engineers of the anthracite region in Pennsylvania. As early as 1881 by-trade filling was used, by John Veith at the Buck Ridge colliery near Shamokin, and the subsequent year Mr. Pardee used it at Hazleton. Shortly after it was employed at the Kohinoor colliery of the Reading company near Shenandoah, and by 1886 it was in common use throughout the Lehigh region. In 1890 filling was introduced as a regular practice at the Buck Diamond colliery at Plymouth. About that time the district was visited by several German engineers, who spent some time in studying the process and gave full credit to the Americans for its novelty. Since then it has been more rapidly adopted abroad than in the United States because the higher price received for coal and cheaper labour permitted refinements that have been regretfully postponed by many a thoroughly well informed American engineer. —*Mining and Scientific Press*, Nov. 21, 1914, p. 791. (G. H. S.)

COMPRESSED-AIR METER. — "A meter which measures easily and accurately the amount of compressed air being used by any tool or flowing through any pipe line is an instrument which should find much use and favour among mining men. The Kreutzberg volumetric meter, placed on the market recently by J. S. McChesney & Co., Chicago, is designed to do this.

The meter operates on the principle that if the flow of a fluid through a container be divided into sections of definite volume and the number of sections be recorded as they pass a given point in a given time, the result is the velocity and volume of the fluid.

This illustration shows a cross-section of the meter. The flow of the air is divided mechanically into sections of definite volume and the number of



Compressed-Air Meter.

revolutions of the drum is recorded on a register which gives direct readings of the flow. The pressure on each side of the vanes is theoretically the same, and in practice differs only by the energy required to rotate the drum. In pressures greater than 5 lb. this amount is negligible. The wear on the vane is at the end and this is automatically taken up. The energy actually required to operate the meter is a small fraction of an ounce.

The machine is constructed entirely of phosphor bronze, to prevent erosion, and the machining is accurate to within 0.00025 in. The meter is made in sizes capable of recording capacities from 0 to 75 to from 0 to 500 cub. ft. per minute."—*Engineering and Mining Journal*, Dec. 19, 1914, p. 1109. (G. H. S.)

LARGE-SCALE SAMPLING. —"At a recent meeting of the New York section of the Mining and Metallurgical Society of America, Sidney J. Jennings described some unusual sampling expedients found necessary by the U.S. Smelting, Refining and Mining Exploration Co. in its work on the Ebner property, near Juneau, Alaska, which it has under option. His description follows:

The ore was supposed to carry $2 in gold, from which a profit of 75 cents per ton was expected. An error of 25 cents per ton in sampling might cut 33⅓% from our profits. The question was, were we taking specimens or samples? We were cutting channels 6 in. wide by 2 in. to 3 in. deep, 5 ft. apart in the crosscuts; and while one might suppose that if there were enough crosscuts the law of averages would

hold and errors would balance themselves, we found that successive 5 ft. samples would assay from a trace to $12 or even higher per ton; hence, it seemed unwise to depend on averages, when the total number of samples would necessarily be small.

We therefore took samples over a strip 2½ ft. wide along the crossents for the full height, getting 30 ton samples for each 12 ft. crossent. These we ran through a five-stamp mill. At first 30 mesh screens were used, but finally a slotted screen equivalent to 60 mesh. The pulp from the stamp was run into a 12 in. launder and the stream was cut at periodic intervals, taking out $1\frac{1}{2}\%$ as a sample. This sample was amalgamated, and the tailings from the amalgamating barrel were sampled, which could be done with accuracy to within 4 cents per ton.

Yet we were never satisfied that the gold recovered in the amalgamating barrel was the right proportion of that in the original ore. Suppose you took 32 samples of ¾ lb each from the same ore, crushed to 40 mesh. If you crushed each sample to 100 mesh and split it, the halves would vary, say, 4 cents. But if you took another of the original 32 samples, the variance between it and the first might be from $1.50 upward, showing that even with 40 mesh material individual pieces of gold would go through the quartering process and not be proportionately distributed. Our millman finally put in a Willey table to concentrate the original 30 tons. We concentrated out about $1\frac{1}{2}\%$ of the original, which carried all the free gold, the pyrite, galena, etc. The tailing from this table could be sampled accurately. The concentrate was amalgamated in a barrel, the tailing from which could also be sampled. In order to ascertain the value of the original sample of 30 tons we had the following factors: Net dry weight of ore crushed; weight of concentrate recovered; total amount of gold recovered in the amalgamating barrel; assay of the tailing from the barrel; assay of the tailing from the Willey table; total amount of gold remaining in the sand around the dies in the battery. This plan has been working three to four months and we really have secured results that check."—S. J. JENNINGS.—*Engineering and Mining Journal*, Dec. 26, 1914, p. 1138. (G. H. S.)

MISCELLANEOUS.

DEBRIS TRANSPORTATION BY RUNNING WATER.—"The U. S. Geological Survey, in *Professional Paper* 86, publishes some interesting results of a study of the laws governing the transportation of débris by running water. The investigation was carried on in a laboratory established at the University of California, in Berkeley.

For each combination of discharge, width, and grade of débris there is a slope, called competent slope, which limits transportation. With lower slopes there is no load, or the stream has no capacity for load. With higher slopes capacity exists; and increase of slope gives increase of capacity. The value of capacity is approximately proportional to a power of the excess of slope above competent slope. This is not a deductive, but an empiric law. This power is not fixed, but depends on conditions. It ranges from 0·93 to 2·37, being greater as the discharges are smaller or the débris is coarser.

For each combination of width, slope and grade of débris there is a competent discharge. The stream's capacity varies as a power of the excess of the stream's discharge over competent discharge. This power varies from 0·81 to 1·24, the values being greater as the slopes are smaller or the débris is coarser. Capacity is less sensitive to changes of discharge than to changes of slope.

For each combination of width, slope and discharge there is a limiting fineness of débris below which no transportation takes place. The stream's capacity varies with a power of the excess of the degree of comminution over this competent fineness. This power varies from 0·50 to 0·62, the values being greater as slopes and discharges are smaller. Capacity is less sensitive to changes in fineness of débris than to changes in discharge or slope.

Most of the experiments were with straight channels. A few with crooked channels yielded nearly the same estimates of capacity. The ratio of depth to width is a more important factor. For any combination of slope, discharge, and fineness it is possible to reduce capacity to zero by making the stream very wide and shallow or very narrow and deep. Between these extremes is a particular ratio of depth to width, corresponding to a maximum capacity. The values of this ratio range, under laboratory conditions, from 0·5 to 0·04, being greater as slope, discharge and fineness are less.

The velocity which determines capacity for bed load is that near the stream's bed, but attempts to measure bed velocity were not successful. Mean velocity was measured instead. If slope be the constant, in which case velocity changes with discharge, capacity varies on the average with the 3·2 power of velocity. If discharge be the constant, in which case velocity changes with slope, capacity varies on the average with the 4·0 power of velocity. If depth be the constant, in which case velocity changes with simultaneous changes of slope and discharge, capacity varies on the average with the 3·7 power of velocity. The power expressing the sensitiveness of capacity to changes of mean velocity has in each case a wide range of values, being greater as slope, discharge and fineness are less.

In general, débris composed of particles of a single size is moved less freely than débris containing particles of many sizes. If fine material be added to coarse, not only is the total load increased but a greater quantity of the coarse material is carried.

Some particles of the bed load slide; many roll; the multitude make short skips or leaps, the process being called saltation. Saltation grades into suspension. When particles of many sizes are moved together the larger ones are rolled.

When the conditions are such that the bed load is small, the bed is moulded into hills, called dunes, which travel downstream. Their mode of advance is like that of aeolian dunes, the current eroding their upstream faces and depositing the eroded material on the downstream faces. With any progressive change of conditions tending to increase the load, the dunes eventually disappear and the debris surface becomes smooth. The smooth phase is in turn succeeded by a second rhythmic phase, in which a system of hills travels upstream. These are called antidunes, and their movement is accomplished by erosion on the downstream face and deposition on the upstream face. Both rhythms of débris movement are initiated by rhythms of water movement.

The laboratory formulas discovered are not immediately available for the discussion of river problems.

The energy of a stream is measured by the product of its discharge, its slope, and the acceleration of gravity. In a stream without load the energy is expended in flow resistances, which are

greater as velocity and viscosity are greater. Load, including that carried in suspension and that dragged along the bed, affects the energy in three ways: (1) It adds its mass to the mass of the water and increases the stock of energy *pro rata*; (2) its transportation involves mechanical work at the expense of the stream's energy; (3) its presence restricts the mobility of the water, increasing its viscosity, and thus consumes energy. For the finest elements of load the third factor is more important than the second; for coarser elements the second is the more important. For each element the net result is a tax on the stream's energy.

In flume transportation capacities are notably larger than for stream transportation, and their laws of variation are different: rolling is an important mode of progression; for rolled particles the capacity increases with coarseness, for leaping particles with fineness; capacity increases with slope and usually with discharge also, but the rates of increase are less than in stream transportation; capacity is reduced by roughness of bed.

The level of maximum velocity may have any position in the upper three-fourths of the current. In loaded streams its position is higher as the load is greater. In unloaded streams its position is higher as the slope is steeper, as the discharge is greater and as the bed is rougher." *Engineering and Mining Journal*, Sept. 5, 1914, p. 430. (G. H. S.)

Abstracts of Patent Applications.

(C.) 431/14. Holman and Holman. Improvements in or relating to rock-drilling engines. 24.11.14.

This patent application is for a rock drilling engine having a rotatable tool-chuck combined with a grooved sleeve-portion interengaging with a correspondingly grooved extension from the drill piston, the provision of a bushing surrounding and engaging with the said extension to form a guide support therefor and separated from the tool chuck to prevent the transmission through the bushing of the rotative effort exerted by the piston on the chuck.

(C.) Wilhelm Mauss. "Improvements in planetary centrifugal separators. 7.4.14.

This specification describes an improved form of the self-discharging centrifugal separator (No. 7643) previously patented by the same applicant. The improvements claimed are essentially for a shield and adhesion rings in the drum to ensure that the deposit of solids shall be carried right to the inner side of the drum before detrainment and discharge.

(C.) 151/14. William Holman James. Improved process for the extraction of metal from metal bearing material. 7.4.14.

This application refers to a wet chemical process for the extraction of base metal bearing material, in which the active chemical agent is formed by the electrolysis of a mineral salt, the original solution being regenerated for further use after the metal dissolved in the process has been recovered therefrom; and also in apparatus for carrying out the process.

(C.) 440/14. Albert Emerson Vandercook. Improvements in thickeners. 7.12.14.

This application describes an apparatus for a pulp thickener or dewaterer, the object in view being the partial separation of liquids from solids contained in pulp formed during metallurgical operations.

(C.) 441/14. Albert Emerson Vandercook. Improvements in process of and apparatus for treating mixtures of liquids and solids. 7.12.14.

This application relates to a process of, and apparatus for, treating pulverized ore or pulp containing precious metals, by means of agitation and filtration in the presence of cyanide solution. The apparatus described consists of a cylindrical vat in combination with revolving hollow arms fitted with jets, a false bottom formed by a filter medium and collecting chamber for filtrates; the chamber is connected to a vacuum pump which, working in combination with the moving jets, constitutes the filtering means.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non receipt of Journals and Notices.

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BEATTY, G. H., *l/o* Knights; Van Ryn Deep, Ltd., P.O. Box 225, Benoni.

BOSQUI, F. L., *l/o* Johannesburg; *c/o* Central Mining and Investment Co., London Wall Buildings, London, E.C.

BOYDELL, H.C., Maraisburg; Randfontein Central G.M. Co., P.O. Box 42, Randfontein.

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GARDINER, B. L., *l/o* Rhodesia? *c/o* H. C. Jorissen, Esq., 734, Arcadia Street, Pretoria.

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HOWARD, T. C. L., *to c/o* Rhodesian Broken Hill Dev. Co., Broken Hill, N. Rhodesia.

ISAACS, R. McL., *l/o* Rhodesia; Consort Mine, Barberton.

LADDLER, H. J., *to* Ivanhoe Gold Corporation, Ltd., Boulder, W. Australia.

LEA, A., *l/o* Germiston; Sub-Nigel, Ltd., P.O. Box 50, Nigel.

MACKESY, W., *to* Cam & Motor Mine, Eiffel Flats, Rhodesia.

MCMILLAN, Q. C., *to* 25, Floss Street, Kensington, Johannesburg.

NEWTON, S., *l/o* Johannesburg; Crown Mines, Ltd., P.O. Box 102, Crown Mines.

NICHOLAS, D., *to* City Deep, Ltd., P.O. Box 1411, Johannesburg.

NORMAND, J. N., *l/o* Rhodesia; Orchard Siding, Cape Province.

NORTH, W. J. R., *l/o* Germiston; Modder Deep Levels; P.O. Box 326, Benoni.

REYLESBACH, L., *to* 29 & 30, Holborn Viaduct, London, E.C.

RUSSELL, W., *l/o* Scotland; 16, South Street, London, E.C.

SALKINSON, A., *l/o* Langhangte; Government G.M. Areas, Ltd., P.O. Box 28, Brakpan.

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Vol. XV. APRIL 1915. No. 9.

Proceedings AT Ordinary General Meeting, April 17th, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, April 17th, 1915, Prof. G. H. Stanley (President) in the chair. There were also present:—

21 Members: Prof. J. A. Wilkinson, Prof. J. S. Cellier, Messrs. J. Gray, John Watson, H. A. White, Dr. W. A. Caldecott, W. R. Dowling, A. McA. Johnston (Members of Council), H. R. Adam, J. A. Campbell, H. M. Dixon, G. Melvill, H. H. Morrell, P. T. Morrisby, S. Newton, E. A. Osterloh, W. S. V. Price, J. F. Pyles, W. A. C. Tayler, A. Thomas, and E. M. Weston.

8 Associates and Students: Messrs. O. A. Gerber, J. Gibson, L. D. Hinde, A. King, A. Milliken, H. Olsson, W. J. Sutherland, and W. Waters.

5 Visitors, and Fred Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBERS.

The President: I think this is the second occasion on record on which we have to announce that there are no members up for election; consequently, it is not necessary to appoint scrutineers. Of course, under the exceptional conditions under which we are living at present it is not, perhaps, matter for very great wonder that there are no new members; but I hope that all present members will do their best to get a few more into the Society. We are losing somewhat heavily by resignations and, unfortunately, by deaths: and we need some new blood, too.

Mr. E. M. Weston (*Member*) made a suggestion to the Council which he thought, if carried out, would mean quite a large addition to the membership.

The President considered it would be rather a new departure, but promised that it would receive the consideration of the Council.

GENERAL BUSINESS.

QUESTIONS AND ANSWERS.

The President: There is another matter which might be considered now. We have had a letter from one of our members suggesting that in the *Journal* we should invite queries to be sent in by members, which can be replied to through the *Journal*. The Council in considering the matter thought this could perhaps be dealt with at the monthly meetings; and it was decided that at this meeting an expression of opinion should be invited from members to ascertain whether they would be likely to come forward with such queries and discuss them. Has anyone any observations to offer?

Mr. John Watson (*Member of Council*): I think this is a matter which might be tried again, as it was tried once before. One sees it in papers of all kinds, in connection with different branches of Science. When things crop up in the course of the month one man sits down and writes a question about it, and someone else who has experience answers it in the following issue.

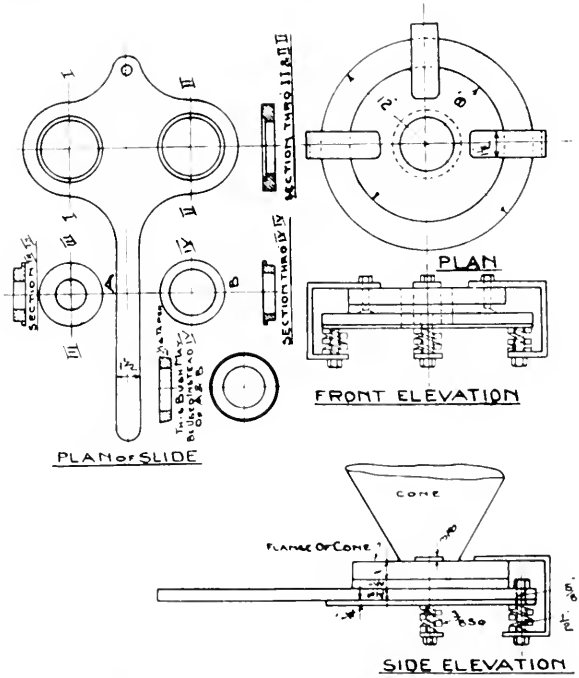
The President: Our previous venture in that direction was through the *Journal*, but it met with such little success that it was dropped. The proposal now is that it might be done at the monthly meetings. There does not seem to be any great enthusiasm with regard to it. Has any member any question in mind at present that he would like to put to the meeting, just to test the possibilities?

There was no response.

THE ROBSON-SALTER GATE FOR CONE DISCHARGE.

Mr. H. A. White (*Member of Council*): The inventors have desired me to bring before your notice this patent device for regulating discharge from cones. The model you have in front of you is simple enough to explain itself, and I draw your attention to the springs for regulating the pressure, which are efficient enough to close off a cone containing water only, and are readily adjustable by means of the bolts shown. The paramount advantages of this arrangement are: the ease with which the nozzle may be changed by throwing over the gate, and the freedom from using dry pulp. A feed chute may easily be attached to the bottom plate to guide the underflow to the tube entrance or wherever desired.

I am informed that this gate is now in use for safety cones, sand cones and fine sand return cones. Personally, I have had one under observation for 10 months upon primary tube cones with feed up to ¾ in. from the battery. It has given no trouble whatever, and is far superior to the usual feed gate regulator. It has given me pleasure to bring this device to your notice, because it appears to be simple and efficient, but I am not responsible for the invention and have no interest whatever in its exploitation.



Robson and Salter's Patent Improvements in means for Regulating Feed of Cones for Use in the Treatment of Sands, etc.

THE USE OF SCOOP DISCHARGES IN TUBE MILLS.

By W. R. Dowling (Past President).

DISCUSSION.

Mr. W. R. Dowling (*Past-President*) introduced his paper, which was printed in the *Journal* for March, and pointed out that a typographical error had occurred in his

paper on page 219, in the second paragraph, where "Comparison of Trials Nos. 1 and 2, etc.," should read "Comparison of Trials Nos. 4 and 5," etc.

Mr. H. A. White *(Member of Council)*: I should like to propose a very hearty vote of thanks to Mr. Dowling for his paper, in which I am sure you will all join me. It is just nine years ago that the author of this latest paper on tube-mill practice read before this Society the first paper* we had on the subject.

I am inclined to think that the use of some form of scoop discharge will soon be widely extended on the Witwatersrand, and the time is therefore opportune for a full discussion of the subject.

It will be noted that the author lays most stress on the increased capacity effect, which is so marked as to be indisputable, but I believe that the smaller increase shown in mechanical efficiency (tons of −90 produced per 24 hours per H.P.), more suggested than proved by his experiments will be found to be very real in practice where each tube does its own work, without receiving an unfair proportion of the returns from other mills.

Beside the obvious necessity for increase in motor power provided, special stress should be laid upon the necessity of ensuring that the pulp cannot run back into the tube and so partly or entirely negative the effect of the scoop, and this is done in the simple arrangement shown in the author's plan. It will be noticed that the author suggests the practical limits of effective scoop diameter at 24 in. with a feed of 100 tons per day. Now, I do not think these limits represent what we may ultimately look forward to, nor are they rigidly fixed when the different conditions on different mines are borne in view. Obviously, if we could use cheaper or more indestructible linings for our tube-mills and at the same time make the small alterations required to ensure an abundant supply of rock suitable for use as pebbles, we should be able to realize the highest results shown in any of the experiments quoted, and I do not know why the present figure for most suitable tonnage feed might not largely be exceeded, when it might be found most profitable to use a larger scoop as is indicated theoretically.

It might be pointed out that the actual level to which the semi-liquid pulp is reduced, on the average, through the mill with the 21 in. scoop experimented with, cannot certainly be known, nor is it possible to suggest any method by which this could easily and certainly be ascertained, though something could be done by stopping the feed and the tube-mill simultaneously, measuring excess outflow if any, and calculating from pebbles measured at rest, and the other data required. I do not anticipate that much difference would be found in level of pulp at inlet and outlet of mill, though I have seen drawings which suggest that the draught-man believed there would be a continuous slope from the lip of inlet tube to the outside holes of the scoop discharge screen. Considering the action of the splash, it is possible that the pulp level is even lower than the effective scoop diameter would indicate.

It is interesting to watch the very simple and reliable arrangement for feeding pebbles at the Robinson Deep, which was referred to in the paper, and the conclusion may be drawn that the difficulty of feeding the large amount of rock required for this purpose will not stand in the way of any future advance.

It has been suggested that one difficulty in the way of the scoop discharge is the impossibility of getting at the screen with rods to clear it in case of choke, but in practice it is found that the increased discharge area made possible practically ensures freedom from choking in the absence of very gross overloading, and in that case the mere stoppage of the tube for the shortest possible time clears the screen at once. For those who imagine they can exactly determine the tube load by passing a thin iron rod through the screen at the discharge end of the tube there is certainly a drawback, but, as the author points out, "the wattmeter is really the best guide for maintaining a uniform pebble load", and most tube-mills are provided with the necessary equipment for this purpose.

Mr. A. J. Herald *(Member)*: The author has referred in his paper to the wear of tube-mill liners, and as the use of tube-mills on the Witwatersrand for recrushing the coarser portion of the stamp-mill product has now been practised for a good many years, it may be interesting to review the various kinds of liners which have been used or suggested.

The following is a list of some of the tube-mill liners which have been either proposed or tried by various operators at different times:—

* See this *Journal*, Vol. VI., April, 1906, pp. 308–315.

A. *Rib Liners.*—(1) Alternate tapered steel bars on edge and flat iron bars set longitudinally inside the shell.

(2) Alternate flat and tapered bars set circumferentially.

(3) Worn tapered steel bars set side by side longitudinally.

(4) Two worn steel bars, in substitution for new flat bars between new tapered edge bars, to ensure retention of pebbles.

(5) Worn tapered steel bars with new tapered bars at intervals to prevent slip of pebble load.

(6) Tapered bars set upon narrow or wide liner plates.

(7) Spiral cast iron ribs.

(8) Cast iron segments curved to suit the shell of the mill, with longitudinal ribs projecting, the segments being bolted to the shell.

(9) Devices for holding tapered steel bars set longitudinally in position, such as chucks, slotted circumferential rings, etc.

(10) Flint liner with steel bars or rails between each row or more of flint blocks.

(11) Steel rib lining with tube-mill pebbles cemented between vertical steel bars.

(12) Wide manganese steel circumferential rings.

B. *Peg Liners.*—(1) Discarded steel scrap, such as drill ends, steel bars, rails, worn tapered bars, and grizzley bars, etc., cut into lengths and set vertically in a block of cement concrete. The lengths of steel are so spaced that each length forms a projecting peg at the required distance from other pegs.

(2) Discarded boiler tubes and piping cut to required length and set vertically in concrete blocks, or packed closely in cement mortar.

(3) Cast-iron pegs cast on a cast-iron base.

(4) Other iron and steel scrap set on end in concrete blocks.

(5) Rails cut into suitable lengths and set vertically in cement mortar (*Eng. and Min. Journal*, January 30, 1915).

C. *Blocks.*—(1) Imported silex and local flint or chert blocks.

(2) Blocks of rock, or steel or iron scrap set in concrete.

(3) Steel bars cut to suitable lengths and set longitudinally in concrete blocks.

(4) Light iron castings shaped to the curve of the mill and divided into small compartments, which may or may not be filled up with pebbles embedded in concrete.

(5) Truck wheels embedded in concrete in tube-mill or made into blocks before placing in tube-mill.

(6) Manganese or other hard steel plates bolted to shell of tube-mill.

(7) Cast iron blocks with longitudinal corrugations.

(8) Wood, rubber and corundum blocks.

No doubt, members can recall other forms of liners which are not mentioned in the above list, and it would be of interest to have these recorded.

On the introduction of tube-milling in gold ore treatment it soon became apparent that to obtain maximum efficiency the lining had to give long life, prevent slip of the pebble load, take up a small amount of space in the mill, and give low total cost per ton of sand crushed during its life. It is generally admitted that the longitudinal tapered rib liner, which is in common use on the Rand in various forms, gives satisfaction in all these respects, although its first cost is high.

It will be seen from the above list that many attempts have been made to utilise some portion of the large amounts of scrap iron and steel which are to be seen lying around the surface works of the mines. Several of these methods of utilising scrap, particularly those in connection with the further use of worn bars from the longitudinal tapered rib liner have been used by the writer for some time, and promising results have been obtained. The writer is now also considering the utilization of worn battery dies for lining tube-mills.

Another method which may be of interest, though not of prime importance owing to limited supply of material, is of a different type, and was designed by the writer, in conjunction with Mr. R. Moore, foreman mason of the Knights Deep, Ltd. The material used in this particular case was the shanks from broken or worn-out battery shoes. These were partially embedded in cement mortar, each shank being made into one block and the blocks built into the tube-mill. The wooden moulds used for making the block were 6 in. × 6 in. and 4 in. deep, being slightly narrower at the top than the bottom so as to suit the cylindrical shell of the mill. A shoe shank was placed in the mould and the space round filled in with cement mortar. The blocks were allowed to harden and then stored away for three or four weeks in a cool place to become thoroughly set. Stem ends might be similarly used, but being of milder steel they probably would not give good results.

In the first experiment, a portion of a tube-mill was fitted with these liners, alternate

tapered and flat bars set longitudinally being used in the remaining portion. In building in the blocks the method adopted was much the same as that used in building in a flint liner, each successive row breaking joint with the previous row, so that a few blocks had to be used which were only half as long as the shoe shank blocks. The whole was built in with cement mortar and set under steam for eight hours. To protect the last ring of blocks which would otherwise have been in contact with the rib lining, a white iron ring in three sections was fitted in, the depth of the ring being the same as the blocks, namely, 1 in. Shoe shanks break off irregularly, so that the dimensions of those used varied in length from 5 in. to 6 in., and in diameter at the base from 4½ in. to 5½ in. The diameter at the top was in all cases 3½ in. It will be seen from the above dimensions that that part of the shank which projected above the cement varied from 1 in. to 2 in.; the distance between shoe shanks varied from 2 in. to 2¾ in. at the top, and was about 1½ in. at the bottom.

The lining was examined after the tube-mill had been running 12 hours. The cement mortar was seen to be wearing fast, a slight packing of pebbles between the shoe shanks had taken place and the pebbles were in good condition (well rounded). After running 48 hours longer the lining was again examined, and it was found that a complete packing of pebbles between the shoe shanks had taken place. This liner has now been running 90 days, and has given satisfactory results. It has been examined frequently during that time and the packing of pebbles has always been found to be complete, and the pebbles exceptionally well rounded.

It is rather premature to give cost figures, because the cost of a tube-mill liner should be calculated per ton of sand crushed so that it may be comparable with that for other linings and again the cost of producing the first lining of the above type would be greater than for subsequent productions probably on a larger scale. The total cost, however, of making the necessary number of blocks and installing them in the two-thirds section of a tube-mill of dimensions 18 ft. 8 in. × 4 ft. 9 in., was £35 2s. 6d.

The weak point in this type of liner will probably be found to be the last inch or inch-and-half of the cement mortar. When the lining has been worn down to that extent the packing of pebbles will cease and the cement will wear away so fast that the whole liner may collapse suddenly. To try to overcome this anticipated difficulty, another liner is being made with a washer round the base of the shank to fit against the shell of the mill.

Dr. W. A. Caldecott *(Past-President):* In connection with the ample sand feed to tube-mills advocated by the author, the question of dimensions of mill is of course involved, since smaller mills naturally require a smaller sand feed than do larger ones. I am not aware of mills longer than 22 ft. being used on the Rand, though shorter ones are not uncommon. These latter are either the ordinary 5½ ft. × 22 ft. mill shortened internally by means of a perforated partition or false end inside the shell, thus leaving a chamber at the outlet end of the mill, or are originally made with shorter shells. In the former case, the shortening may be done in order to conform to the limited power of a motor already installed, especially when a scoop discharge is employed, or because the limited sand feed given cannot fully utilize the falling pebbles near the discharge end of the full length mill. In some tube-mills a reduction of length by one-fifth has been accompanied by increasing the shell diameter to 6 ft. Since increased diameter involves fewer revolutions per minute, and consequently fewer drops of each pebble, the advantages gained in ordinary practice by this design are not obvious. Short tube-mills appear to have originated in Western Australia, where all-slimming of ore is commonly practised, with concurrent relatively fine tube-mill feed. Under these conditions it may be difficult to feed a sufficiently large tonnage of sand to ensure ample over-size particles near the discharge end of a long tube-mill, and the use of short tube-mills may be desirable in such cases. Under any conditions the feed tonnage requisite is affected by the coarseness of the feed, since the greater the percentage and the size of coarse particles in the feed the less tonnage is required to usefully employ all the pebbles in the mill, and *vice versa*. In other words, the coarser the battery screen pulp or the final tailing pulp, the smaller the feed tonnage required, as compared with fine battery screen pulp or fine tailing pulp. Under similar local conditions of crushing to sand and slime, short mills without increase of diameter require a larger number to be installed to obtain the same crushing

capacity, thus contravening the dear bye rule for reducing capital expenditure, attendance, and operating cost by installing as few and as large units as may be practicable in each case. Even as regards short (5 ft. ×11 ft.) tube-mills, recent investigations in Mexico have shown that "with a light feed, the cost of grinding, elevating and classifying per ton of initial sand fed was 1.88 dollars U.S. currency. With a medium feed, the cost per ton was reduced to 97 cents. Now, with a heavy feed, the cost has dropped to 68 cents per ton."*

It is, of course, conceivable that a tube-mill might be designed to crush the ordinary stamp-mill ore feed, and stamps entirely dispensed with. This might be done by sufficiently increasing the diameter so as to ensure considerable drop of falling pebbles, and by the use of very large ore pebbles. This principle, but employing steel balls in place of pebbles, was illustrated in the late Giesecke mill trials, which demonstrated large crushing capacity, but failed economically on account of the excessive wear of the steel balls employed.

The "tons of −90 product" measure of crushing capacity used by the author is a convenient conventional method, but cannot be taken to express the whole truth under widely differing conditions, since the higher the percentage of −90 product in a final pulp, the smaller the average size of particle passing this screen. In other words, two products, both of which pass a 90-mesh screen, may have different proportions passing a 200 mesh screen, which is equivalent to saying that the surfaces and the gold exposed in one case may be greater than in the other, and the percentage of extractable gold hence increased.

In actual practice on the Rand, the modern system of tube-mill classification with safety classifiers is capable of very close adjustment, and may be so regulated in proportion to the ore tonnage, together with the number of crushing units in stamps and tube-mills operated, as to vary the fineness of crushing in accordance with the value of the ore milled at different periods. Since in a modern reduction plant the percentage of gold extracted is dependent upon the fineness of crushing, which richer ore warrants carrying further than poorer, the works records soon show the percentage of −90 product in the final tailing pulp required to obtain the economic limit of total residue value, say 0·25 dwt. per ton.

The author does not deal with the feature of amalgam held up in tube-mill liners. Owing to non-cylindrical tube-mill shells and to liner bars not quite straight causing crevices, this may be equivalent to several hundred ounces of fine gold in one tube-mill with ordinary steel-rib liners, and in view of the large number of tube-mills now in operation on the Rand the loss of interest on gold thus locked up is considerable, besides causing fluctuations in the monthly gold output due to variable absorption of amalgam or release when relining. The modern system of elimination of stamp-mill plates has naturally increased the amount of amalgam thus held up owing to the richer product entering tube-mills. Allowing the cement setting of liners to harden well before use reduces the crevice space, and hence the amount of amalgam retained therein, but when a crushing plant is well supplied with ore time is not usually available to allow tube-mills to remain long out of action for setting of liners. The importance of this question has caused attention to be paid to it for some time past, and in time it is probable that the difficulty will be in great measure overcome.

The President: I am sure you will agree that this paper marks a very notable advance in tube-mill practice, an advance which is obviously not the result of sudden inspiration, but of a series of very carefully conceived and carried out experiments. Undoubtedly the experiments sometimes occasioned a certain amount of disappointment, but the experimenters stuck to it; and the results is a very great advance indeed in the way of tube-mill capacity.

A vote of thanks has been proposed to the author, and I am sure we heartily agree.

SAFETY MEASURES IN THE USE OF EXPLOSIVES.

(Printed in March Journal.)

By Wm. Cullen (Past-President).

DISCUSSION.

The President: Before I ask Mr. Waters to introduce Mr. Cullen's paper, I would just like to tell you that Mr. Colquhoun, Mr. Cullen's successor at the Dynamite Factory, is down with enteric fever. I am sure we all join in wishing him a speedy recovery.

* See paper by E. T. Mishler, *Engineering and Mining Journal*, Sept. 12, 1914, p. 469.

Mr. W. Waters *(Associate)*: briefly introduced this paper, which was printed in the *Journal* for March.

Mr. W. S. V. Price *(Member)*: With reference to the author's remarks regarding a man blasting his cut, then returning to the working face and charging and blasting the round in the same shift, this is now prohibited by law. So, if the regulations are followed, that cause of accident will be stopped. I remember a case of a miner who had blasted his cut and had gone back to blast the round; finding the cut had not come away well, he sent his white helper to the box for more gelatine. On his way back the helper heard a report, and being frightened, ran away without telling anyone of the occurrence. The first man to find it out was the night shift boss, who found the man blown to pieces. It was evident that as all the other holes were intact, the miner had charged the hot cut holes with gelatine. Whenever I have had to blast the cut and return to blast the round I have invariably flooded with water the old cut holes that I might need to reblast; and left them until the last to be charged.

I have often thought that misfires might be caused by the fulminate in the detonators becoming damp through men blowing into them to remove sawdust that could not easily be shaken out. The moisture of a man's breath may at any rate retard the action somewhat and so cause a cut-off or even misfire.

The author states he has never had a misfire or a partially exploded discharge. I have seen several. In one case, a well drilled cut, where the holes at the back converged to within an inch of each other, one hole misfired and the gelatine remained in the hole—a very marked case of insensitiveness. In another case I saw a stick of gelatine (the maker's name I will not mention)—the primer cartridge, in which the detonator only had exploded, shattering the end of it so that it resembled in shape a partially opened hand, but not exploding the whole charge.

Cutting-out shots.—The author appears to be very severe on any one putting a primer in anywhere but at the top of a hole, owing to the alleged danger of the gelatine being ignited. If a man can afford the time, and has room in his stope to blast only two holes at a time, which is the case on some properties, the primer at the top is quite all right, for there is no danger of a cut-off; but if there is ground with transverse parting planes, say in a drive, and the primer is put in last, it is asking for cut-offs. Personally, I always put my primer in about the middle of a hole; and if a good fuse is used and the fuse is greased, there is always sufficient sludge left in the hole to keep it damp, for I only blow out a hole sufficiently to allow the first cartridge to go right to the back of the hole. I consider the practice perfectly safe, and I have certainly found my results very fair, giving good footage per round, showing that the detonation was good. I note the author does not agree with the use of dynamite for primers. To save expense on one mine where I worked we were ordered to use dynamite primers, as it was claimed that the force of the explosion was transmitted more quickly through the whole length of the charge, and the difference between the cost of a gelatine and a dynamite primer was saved. I was contracting, so of course was trying to save, if possible; but all I made out of it was a headache through handling the stuff, as I found that the usual amount of gelatine had to be used to get the same result. I quite agree that No. 8 detonators should be used in all cases, as they give much better results; but, unfortunately, we cannot always obtain them.

The author mentions men breaking a stick of gelatine across the middle when charging a hole. When I am charging up, and the hole is getting too big for the size of the gelatine (for the hole decreases in size as it goes in) I twist the stick of gelatine longitudinally, then when the gelatine is gently pressed home, keeping a little weight on the tamping stick, I give that a slight turn in the same direction, which fills up the hole and expels the air. Here, again, a little sludge helps to fill up any spaces that may have been left in the hole.

Mr. E. M. Weston *(Member)*: Mr. Cullen's paper interested me because, as a matter of fact, I had been writing something about the same subject myself. Mr. Cullen omitted, I think, to mention bad crimping causing misfires; but on the Rand to-day one is very often supplied with most abominable crimpers. I had quite a collection of badly crimped detonators that I picked up among the gelatine, or which I had given to me during development work. This is quite a noticeable cause of misfires. The core of the fuse is often practically

squeezed out, so that it is impossible for the charge to burn down and ignite the detonator.

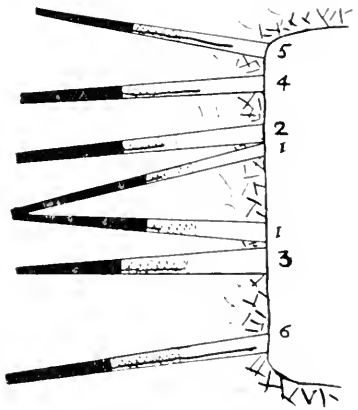
I see that Mr. Cullen is just a little discouraged about electric blasting. One does not like to say: "I told you so"; but I did try to warn people very carefully about the danger from stray currents. I was then told it was quite impossible that a stray or accidental current could possibly cause an accident. I also said that if an accident was caused it would not be just one hole that would go off, but my phrase was: "A regular bombardment would be caused." I do not say I thought of the exact manner in which the particular stray current was to be introduced; but we know, during shaft sinking, lightning discharges did, on two occasions, cause a most lamentable accident; and I am still sure it might also occur in the other ways that I mentioned. I do not think, with the class of labour we have here, electric blasting is necessary or desirable. It is rather instructive to learn how much fuse is still being burnt in the "electric blasting" mines.

I should like to ask the author if he has any objection to the practice of tearing off the ends of the packing of the cartridges, so that the gelatine touches throughout the charge. We are always told that the wave of detonation is liable to be reduced in strength as it travels along; that anything interposed between the cartridges is wrong. Well, we have, even under the best circumstances, 1/8th in. air and paper. In some cases I have recommended that the ends might be torn off.

There is another point, that Mr. Price brought up, we have, say, the last cartridge that has to go into the machine hole; a cartridge 1¼ in. in diameter. It has to fit into a circular hole 1½ in. or 2 in. in diameter. I have put in several longitudinal cuts with a knife into such a cartridge so that it collapses and fills up the hole with the use of less force. I should like to know what the opinion of the author is on that practice; whether he considers it dangerous, or whether he considers it allowable.

I believe coloured wrappers are quite useful, and it is very often much easier to pick out fragments of cartridges after blasting. The author mentions that frozen gelatine is not found on the Rand. I have known gelatine to freeze in the magazines of the Rand during the winter months, and have often had to thaw it before using it, especially during shaft sinking, when gelatine used to be brought up in sacks rather wet and put in the magazine. It seems to freeze rather more easily then.

I do not think the author, or Mr. Price, has dealt with what I consider the greatest cause of misfires on the Rand, and also, of course, because of that, one of the greatest causes of accidents. It is, I believe, due to a mistaken wish to economise, and this has lead to the increased use of cheap fuse with an irregular burning rate, although Mr. Cullen says that the fuse on the Rand is generally good. Fuse is now supplied for development work which has a marked burning rate of 90 to 110 seconds per yard. A fuse like that is absolutely, in my opinion, unsuitable for development work on the Rand at all. Such fuse is, say, the 3d. to 3½d. fuse, whereas by paying another 1d.



Rough Sketch of a Development End.

or so one can get a fuse with a regular burning rate. The reason for accidents is not far to seek. Turning to the rough sketch of a development end, we see that taking the holes in pairs to be fired in rotation, even in soft ground with only one easer there are six holes to be fired in rotation as numbered. Now, with the 90 to 110 seconds fuse, assuming we find a very much smaller difference of burning rate between different coils and packets, we require, in order to make sure that the holes will go off in proper rotation (if they do not this will cause great loss and probably also misfires and dangerous sockets) we must cut our fuse in differences of length varying from, say, 4 in.

to 6 in. This means that at the time the cut goes off No. 6 hole has unburnt fuse in from 2 ft. to 3 ft. long. If there are more holes and easers the condition is worse. I have tried to show what this means by drawing the fuse in the sketch. It is this *protruding unburnt* fuse that gets cut off and is the cause of more misfires on the Rand than anything else. Were the miner given fuse he could depend on when cut, say, with only ½ in. difference of length, misfires would nearly disappear.

Now, we understand why some miners put their primers at the bottom of the hole, a practice which the author quite rightly condemns. Mr. Price tells us he puts his primer under two sticks. This is bad, because the detonator should point towards all the charge, not away from any of it. He does it because he is given poor fuse and wishes to prevent fuse being cut off. The charge occupies about 2½ ft. of the bottom of the hole, and, as he admits, it is quite unusual to get 3 ft. of the mouth of a hole cut off by the explosion of other holes, so that if the fuse is burnt well down to the detonator there is no need to do this. If, however, there is 12 in. or 18 in. unburnt fuse sticking out, it is cut off if only 1 ft. or 2 ft. (which is common) of the hole is blown away.

This matter is costing the mines thousands of pounds, the miners trouble and loss, and I believe valuable lives also. On a mine I was connected with we realized this. The manager was only too willing to help, and the men only too willing to pay another penny a coil for good fuse. The first act of a new manager, without consulting any of the underground officials, was to make an economy by buying cheaper fuse. Fuse blasting has got a bad name and been abused by advocates of electric blasting, simply because the miner has not been supplied with the best material safety and economy demanded he should be supplied with. The Industry has, I am glad to see, adopted my suggestion to call in the bioscope to educate the miner. I hope they and the Government will look into this matter also.

Mr. Cullen refers to an old controversy I stirred up over interrupted charges in holes. Since then I made an experiment to see if there was any perceptible weakening of the wave of detonation down a long charge. I placed 18 3in. sticks of Modderfontein blasting gelatine in a row between 5 ft. and 6 ft. long on a slime dump. I tore off the paper ends of the wrappings so that the gelatine was continuous. I expected from what I had read that the crater formed by the explosion at the end would be much smaller, even if all cartridges were exploded. The results showed a crater of perfect oval form. I published the photo in the *Engineering and Mining Journal*, New York. From this I gathered that there is no perceptible diminution of the wave in any charge used on the Rand to-day.

I should like to ask Mr. Cullen what is the relative strength of an explosion caused by a blow, as on an anvil, compared with that caused by a detonator.

I beg to thank the author for his paper, which can only do good.

Mr. A. McArthur Johnston *(Past-President):* After the interesting discussion we have had, it gives me very great pleasure to move a vote of thanks to Mr. Cullen for his excellent paper. Mr. Cullen has been so long connected with the use of explosives and has advocated safety measures on the Rand so strongly, that I think it very appropriate he should leave us with this paper as a little memento of his work here. Very few along the Reef have helped more than Mr. Cullen has to encourage miners to be more careful with explosives; both by his words (for he has given many lectures—by his experiments at the Factory, and also by the routine work which he and his staff have carried out on our mines.

I am pleased to note that Mr. Cullen has managed to make a success of the "cheesa", or firing stick. I am only sorry that Mr. Waters was not able to bring a sample here to show us. Regarding the tests he has made with fuse, I think Mr. Weston will agree with me when I say that Mr. Cullen must have used an extra good fuse for these tests he carried out.

I agree with Mr. Cullen when he says that it is difficult to realise the deterioration that takes place in the fuse covering. If it were at all possible, I should very much like all fuse imported into this country to be used within a month. I think our mines should try to obtain fuse which has been very recently imported, and not carry a large stock, because in a few months the deterioration is very appreciable. I am sure if Mr. Cullen had carried out tests on fuses which had been six or 12 months in the country his results would not have been so satisfactory.

Mr. H. A. White *(Member of Council):* I should like to ask the last speaker if that deterioration to which he referred takes place inside the zinc-lined cases in which the fuse is generally imported.

Mr. A. McArthur Johnston *(Past-President):* Yes, I think it does to some extent. I have tested fuse which has been lying in zinc-lined cases for three years, and found deterioration. This may be due to the quality of the gutta percha or the rubber covering.

Prof. J. Cellier *(Member of Council):* Might I ask what the exact deterioration is?

Mr. A. McArthur Johnston *(Past-President):* The deterioration is mostly in the waterproof material.

Prof. J. Cellier *(Member of Council):* Not the rate of burning?

Mr. A. McArthur Johnston *(Past-President):* No, I have not found this to be affected so much.

Mr. W. Waters *(Associate):* Mr. Weston has stated that the regularity or otherwise of the burning rate of safety fuse is controlled by the cost, and that cheap fuse is irregular, while the more expensive fuse is regular. Mr. Weston is not quite correct in his assertions. I have tested fuse costing 9d. per coil, and I have not found it to be more regular in burning speed than fuse costing less than half this price. The question of cost does not necessarily control the regularity of the burning speed, as a cheap fuse can be made quite as regular as the more expensive article. Regularity is not a question of cost of materials, but of skill in manufacture. Usually, the difference in price of fuse is controlled by the quality of the water proofing material. It has been proved that the highest, and hence the most costly, grades of black powder are unsuitable for safety fuse, as these very high grades cause the fuse to explode. All safety fuse imported into South Africa has to be marked in accordance with the Mines Regulations as burning between the limits of 90 to 110 seconds per yard.

Mr. E. M. Weston *(Member):* Has Mr. McArthur Johnston made any tests with the different burning rates.

Mr. A. McArthur Johnston *(Past-President):* The shot-firer made fuse generally has a more irregular rate of burning than the other.

Mr. E. M. Weston *(Member):* Mr. Waters states that regular burning speed is a matter of skill in manufacture and has nothing to do with the price paid. All I can say is that one usually has to pay for skill in manufacture as for every other quality. Mr. McArthur Johnston bears me out when he states that he has found that cheaper fuses have a more irregular burning rate. I can only say I have tried the cheaper kinds, tested them, and seen careful miners test them, and for the reasons I now give I consider it the duty of the Government to prohibit the use in development work of any fuse but that having the most regular and dependable rate of burning speed that it is possible to manufacture. By doing so the Mining Industry will save money, accidents will be prevented, and lives saved.

Prof. J. Cellier *(Member of Council):* To show the importance of this subject from both a welfare and economic point of view a few figures, for the year 1913, are given.

Accidents in Transvaal gold mines[6]:—

	Killed.	Injured.	Total.
Total	808	2,005	2,813
Due to explosives ...	223	253	476
Percentage of total	27·6	12·6	16·9

Transvaal Gold Mines[7]:

Total working cost	£23,628,007
Total stores purchased	£10,579,712
Powder, detonators, fuses, and lighting torches purchased	£1,568,028
Percentage of total working cost	6·6
Percentage of total stores purchased	14·8

Regulation 106 (19), as far as it is applicable to Rand conditions, appears to have been originally drawn up when there was only "red dynamite" being manufactured, and seems to have been forgotten since then.

In Witwatersrand gold mines there are no ten other regulations put together so consistently broken as this one; in fact one does not see it observed more than a few times in a lifetime. The small demand for

[6] G.M.I. Report for 1913, Table No. 18.
[7] Transvaal Chamber of Mines Annual Report, 1913, pp. 225 and 7.

Nowadays clips is only another proof of this

Even when employing gelignite, containing some 35 % of non-plastic material, there appears to be no necessity for tying on the cartridge. That making a positive connection between cartridge and detonator may prove safer I do not dispute. It has, however, this disadvantage, that primers may be stored at the end of the shift, and this is very poor practice. A detonator well pressed home in a hole made in the cartridge by a slowly tapering stick—always carried along with the cartridges—gives such quick and good results that there is no necessity to prime cartridges in a place other than at the hole to be charged.

Blowing into a detonator to remove sawdust is, as Mr. Price points out, very likely to impair the detonator, not so much from condensation of the moisture in the breath as from the danger of blowing some saliva into the detonator.

To the advocates of placing primers other than on top of the charge I would recommend the reading of an article appearing in the *Mining and Engineering World*, July 25, 1914, Vol xli., No. 4, p. 149, and reprinted in this *Journal*, Vol. xv., No. 5, (Nov., 1914), pp. 105-107.

In sinking the Witwatersrand G.M. Co.'s vertical shaft the use of the primer in the bottom was fully justified, as pointed out by Mr. Hutton.* Water tamping was employed, thus rendering side-spitting by the fuse almost impossible. In sink holes there is no disadvantage in firing the charge upwards. In squaring holes, in sinking, the burden is not so large as to prove a drawback to this direction of firing.

Regulation 106 (22) provides that "No tamping material shall be inserted between cartridges". Where long holes are drilled—and it is known that long holes are cheaper than short holes on account of the rigging time being constant, etc.,—it is impossible in stoping to concentrate the whole of the charge in the base of the hole for fear of bull-ringing. Hence the insertion of tamping between cartridges by the most competent miners is almost universal. Fortunately for the miner at charging up time Mining Inspectors are very seldom met with.

As Mr. Weston points out, there are many inferior detonator-crimpers in use. No tool which shows the least tendency to cut either detonator or fuse—and these are usually the variety crimping on two opposite sides only instead of all the way round—should be permitted to be used. The insertion of a spiral spring—such, for example, as is seen in pruning-shears—between the handles of a crimper would add very little to its cost and make it a pleasure to use the tool, instead of the labour it is at present.

Messrs. Cullen and Waters never having a misfire or a partial explosion by no means convinces me that such are not of frequent occurrence, owing, in many cases, to the quality of the explosive used. It is to be presumed that they employed Modderfontein explosives and Nobel's fuse and detonators in their experiments. The latter two are probably equalled, but perhaps not excelled by any varieties used here. In the fresh condition in which it is used here, the former has no equal for either sensitiveness or plasticity, both so welcome to the miner. I venture to state that if all the explosives used on these fields were of the same standard we could confidently expect to see explosives accident-rates decrease by 50 %. If these accident-rates were analysed according to makes of explosive—only three are in use on this field—the above conviction, honestly acquired by actual experience, would be borne out by statistics.

The burning speed of fuse is defined by the Explosives Regulations, Chapter XIV., 1 (b): "All safety fuse shall burn between limits of 90 seconds and 110 seconds per yard, and must burn evenly throughout its length." From this it will be seen that the rate of burning, though allowing, as Mr. Weston points out, for limits varying by some 20 %, has to be even for any specific brand, and unless different brands are mixed no allowance need be made for this comparatively wide divergence.

The President: We all know the enthusiasm and energy with which Mr. Cullen has taken up this subject of "safety first" on the mines, and this is not the first paper by any means he has read to this end. He feels so keenly on the matter, that I know you will be interested to hear he proposes to have this paper re-printed and circulated. He feels that the class of man he wants to get at—the miner who uses the explosives—does not see our *Journal* at all, and he therefore proposes to have his paper circulated to reach him.

A vote of thanks to Mr. Cullen was carried unanimously.

* G.M.E. Report, 1912, p. 106.

NOTES ON THE DUST ALLAYING PLANT AT THE FERREIRA DEEP DUMP.

(Read at the November Meeting, 1914.)

By S. Newton (Member).

REPLY TO DISCUSSION.

Mr. S. Newton *(Member):* In reply to Mr. B. A. White, there is no doubt that the Ferreira Deep dump is well situated, as travellers along the Booysens Road can see what has been done to stop the dust nuisance. Calcium chloride was not used in our first experiments.

As there seems to be some doubt as to the hygroscopic nature of the salt used, I should like to point out that it is well known to indicate moisture in the atmosphere. The common salt used was not analysed to ascertain what particular part of the salt absorbed the moisture. Mr. White also refers to details of the experiments which were carried out. As we had to consider the large area to be sprayed and the peculiar nature of the current face, from an economical view point, something easily obtained in large quantities and not costly only could be considered, and residue slime was outstanding among the various possibilities.

Water glass and magnesium chloride were found to be too costly and difficult of application.

Molasses solution was not capable of withstanding the rains.

Cement alone as an emulsion and in combination with slime, cost too much.

Waste and crude oil were fairly effective, but too costly.

Mr. Moynihan I must thank for his very encouraging remarks.

It is difficult to say how much the mines and surrounding districts really benefit both as regards health conditions and wear and tear on machinery. The practice of mixing a certain quantity of slime residue pulp with each truck of sand was found to cause the truck of sand to become very compact and caused great delay in clearing the mass from the truck.

Mr. E. M. Weston remarks on growing vegetation on the dump. There is no difficulty in getting vegetation to grow on the flanks of the dump if covered with ash. But the current tipping face is the cause of most of the nuisance, and until the mine is finished and a coating of ash can be applied, no vegetation is likely to grow.

In reply to Prof. J. Cellier, I am sorry I cannot give the exact details of all the experiments carried out.

As regards ash, of which large quantities were at hand in years past; this is now not so plentiful on account of the increased employment of electricity as driving power. However, ash for the flanks of the dump, if available, is excellent. But, again, it is the current tipping face which causes the dust storms, and an emulsion of some kind seems to be the only preventative.

Messrs. Toombs and E. H. Johnson's valuable contribution on the details of their spraying plant on the East Rand Proprietary Mines show that another substance other than slime and brine can be used very effectively, so that if one material is not at hand the other can be substituted.

In reply to our President, I should like to say that about 12 years ago I procured from Norfolk some of the marrum grass seed from where it is grown and cultivated on the sand dunes. But it did not withstand the acidity of the flanks of the tailing dumps.

I wish to thank all those who have contributed to the discussion on the paper

PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M

(Printed in Journal, Dec., 1914 - Jan., 1915.)

DISCUSSION.

The President: This paper has produced a very large amount of discussion before the Institution of Mining and Metallurgy, in fact more than any paper of recent times; and it seems rather surprising that locally we have had so little to say on the subject. We will print in the *Journal* a resumé of the further discussion which has taken place in London for the information of members.

Discussion before the Institution of Mining and Metallurgy, London.

Bulletin No. 124, Jan. 14, 1915 (abridged).

Mr. H. F. Marriott did not propose to enter into a general discussion of Mr. Rickard's very able paper, although had the times been otherwise he would have been

tempted to do so. His present purpose was to exhibit a diagram which he thought would be found of considerable interest, and material evidence against the views of those who claimed that the Rand was one of the instances of decrease in value in depth. It was from the portion of the Rand represented on the diagram that the major portion of the gold had been won, and also today about half the gold was still being won; therefore, it could not be said that he was unduly selecting any section of the Rand. Some of the freely quoted statements made in the past had not been based on broadcast evidence of this nature, but on restricted areas that had been unfortunate from the very commencement. The diagram exhibited showed a portion of the Rand about 8 miles long and 2 miles across.

A general map of the Rand on the same scale would be about five times as long as the one now shown. In what other mining district in the world or in discussing what other vein in the world would anyone, who had opened out 1,000 ft. along the strike, go down only 5 % of this or 50 ft., and then on the disclosures of that 50 ft. decide whether the vein was deteriorating or not in depth? That was a just comparison, and before deciding that the Rand was deteriorating it must be judged in the same way as other veins in the world.

He would point out a fallacy which had taken root owing to the nature of the records of the earlier days. The early published results were obtained out of about 50% of the reef deposit, and the gold contents were recorded in assay value over the reef width. Latterly the gold contents had been recorded over the stoping widths. These were wider than the reef widths sampled, so that in comparing the successive figures in annual reports a considerable difference was noted in the new exposures, whereas on the same basis it might be of the same value as formerly.

He then proceeded to describe the scheme of the diagram, and pointed out how clearly it demonstrated the continuity of gold contents in the lower portions of the mining area as compared with the records of the upper levels.

He would point out a very interesting piece of evidence afforded by the map. The mine indicated, when 1,000 ft. deep, was very rich, and the adjoining mine was also promising in its results. Afterwards both of these mines fell on very bad days, and, promptly, that part of the Rand was declared to be decreasing in value in depth, and the whole industry was saddled with this opinion. Later on, on further development at depth, the first opened out a large deposit of extraordinarily rich ore.

The mine next below it was then opened up. A continuation of the rich ore was proved, and the Rand was saved again. Then the lower mine became poorer. It was a most depressing mine. But they went on in spite of difficulties, and at still greater depth had come into another zone of excellent ore, and now the deepest workings had shown the richest ore yet found in the mine. In the meantime, the adjoining mine had also got out of its poor zone in the upper levels, and now, at the greater depth, was in the height of its prosperity.

There was another point of interest to mention. It would be seen that one rich section, which occupied a large tract of country in the lower levels, was only indicated on the outcrop at one spot, and that between workings of distinctly poorer grade. If those who took the risks and did the work on the Rand had been influenced by those who preached deterioration, one of the most important sections of the Rand might never have been opened up.

Mr. S. J. Truscott said that with improvements in pumping and hoisting, mining had been able to reach to great depths, and one after another the continuity of ore deposits had given way to the mining appliance. That surrender one by one of the mines in depth had caused engineers and mining people to consider the position, and he thought the result had been that in general the conclusion had been reached that there was an impoverishment in depth, and in 1912 it was the considered opinion of mining engineers that ores did not persist in depth.

Although this was the case generally, there were some deposits which could claim a greater relative persistence than others. Ordinary bedded or sedimentary deposits of course had no extent in depth except that which was given to them by being tilted. Enrichment deposits were essentially surface deposits. He referred more particularly to the porphyry copper deposits. The limestone or other replacement deposits also chiefly extended in a horizontal direction. Only the lodes or fissure deposits had any great extent in depth; depth was the most important dimension of a fissure deposit.

He would rather like to emphasize this and other properties of the fissure deposit. It really did extend in depth more than most other deposits. It was a tabular deposit, and estimates of ore quantities were more easily made than with most other deposits. It lent itself more readily to systematical sampling. Leaving out of consideration ore-beds and coal seams, fissure deposits were the most regular of all deposits.

It was interesting next to consider those particular deposits which the author had mentioned as having attained great depth, and to see to which classes they belonged. The Comstock lode was essentially a fissure deposit and it had reached, as the author said, a depth of 3,500 ft. But that was the extreme depth. It was not payable to that depth; it was payable, probably, only to a depth of something a little above 2,000 ft. It was a Tertiary lode. The Pzribram lode was worked to 5,640 ft. That was a Lower Silurian deposit, or rather, it occurred in Lower Silurian strata. The Bendigo lodes also occurred in Palæozoic strata. The Bendigo and the Pzribram lodes were fissure deposits. The Lake Superior copper deposits were bedded deposits; they had been followed to an extreme vertical depth of 5,368 ft., and occurred in the Palæozoic formation. The Witwatersrand was also a bedded deposit which had been followed to an extreme depth of 4,780 ft. and occurred in Palæozoic strata. The lodes in the Kolar goldfield were fissure deposits, occurring in Archæan schist. The cases mentioned by the author therefore showed that fissure and bedded deposits no more than any others where responsible for the cases where considerable depth had been reached.

The author implied that secondary enrichment had a wider significance than that usually claimed for the term. He implied that lodes from the date of their primary formation to the time when they are worked by man were continually becoming enriched, not necessarily by meteoric waters but also by ascending or horizontally moving waters. To his (the speaker's) mind it that were an important factor in the genesis of ore deposits there would be greater evidence of it. When a lode after its original formation had subsequently been reopened, such reopening could be detected in many ways; new crusts of ore were added to those already there; brecciation of the pre existing ore took place; movement along the lode plane was marked by triturated material. With these evidences of reopening and further deposition there was as a rule little evidence that the pre-existing ore had been chemically deformed or enriched. Where there had been no brecciation or trituration, but only the quiet deposition of another crust, there was, in the fact of the unaltered condition of the pre-existing crust, evidence that the new solutions had only time or property for deposition—theirs was no long-continued operation.

When, therefore, in the lode-material there was no evidence of reopening it might be taken that no enrichment had taken place from depth. That would be the case with most lodes, and consequently, exclusive of alteration by secondary enrichment in the ordinary sense of the term, lodes now found were much in the condition in which they were originally formed. The view therefore held good that a deposit connected with one of the earlier eruptive epochs depended chiefly upon that eruption for its filling and only subordinately, if at all, upon any subsequent events.

To argue otherwise would be to argue that a pronounced tendency to uniformity was a prevailing characteristic of lodes. There was no such uniformity; lodes differed in their types. The tin-copper lodes and the lead-zinc lodes of Cornwall were practically all due to the intrusion of the granite there; that was to say, they were of the same age. In their case, therefore, time had not levelled or destroyed the characteristics given them at their birth.

This difference between lodes was particularly noticeable in the case of the gold lodes belonging to what might be termed the old gold group and the gold-silver lodes of the Tertiary group.

The old gold lodes were represented by those in the Kolar, Bendigo, and Minas Prietas and other districts. They usually occurred in ancient schistose rocks and in association with deep-seated extensive eruptives. The walls were slickensided, and there was an absence of crusted structure. The quartz was coarsely crystalline and often occurred in lenses or pleats. One sulphide was usually predominant, were it pyrite, arseno-pyrite or stibnite. Gold was the important metal, and it generally occurred very coarse. An interesting fact was the oft-mentioned occurrence of graphite. These lodes relatively often persisted to great depths, throughout which they maintained their characteristics.

The young lodes on the other hand were represented by those in the Comstock, the

Werlite, the Pachuca, the Sumatra and other districts. They usually occurred in Tertiary eruptives of local extent. The walls were usually highly and extensively propylitized, and crusted structure was characteristic. There was generally little evidence of movement, except such as that in response to a bursting pressure. The quartz was to a great extent cryptocrystalline. The sulphides were complex. Gold and silver, both finely divided, were the important metals. These lodes, as mentioned before, did not continue to great depth; their wealth was relatively near the surface.

From these considerations, from considerations of lode intersections, and from considerations of general geological position, it might be accepted that ore deposits were not of necessity geologically young nor, apart from secondary enrichment, were they generally young, but of all ages. Moreover, the old lodes were the roots of more extensive original deposits of similar character, part of which had surrendered to erosion.

Experience, nevertheless, showed that non-persistence in depth was normal, though great depths might be reached. Further, theoretical reasoning argued a gradual impoverishment and a final disappearance of any particular orebody in depth, with a chance, however, that deeper still, so long as the fissure continued and the mineralizing eruptive had not been entered, the deposit might take on another character. There appeared nothing, however, against the possibility of certain deposits continuing to greater depths than any yet reached. By certain deposits were meant those where deposition had resulted from pneumatolysis, from the precipitating action of the lode walls or of magmatic carbonaceous material, etc.

Nor did the acceptance of eventual impoverishment in depth raise the corollary that search for further ore was in general best undertaken laterally rather than in depth. The author had given four specific instances where lateral search gave abundant results, though deeper search had given nothing. These instances, however, were all Tertiary lodes connected with andesitic eruptions, and, as already mentioned, one of the characteristics of this class of deposit was that its members did not extend to any great depth. Each deposit must in itself contain the indices to the direction in which further ore was most probable. With some, lateral exploration would offer better chances, with others, greater depth.

In conclusion, the author had rendered a great service to the Institution in putting on record within its *Transactions* the expression of an opinion so sound upon the question of impoverishment in depth. There was no doubt that to have this record would be of considerable assistance to mining engineers generally, and especially to those who had to combat the unwarranted optimism of other parties. So long, however, as these parties were able to point to the fact that the profession made no effective demur to paying full income tax upon mining dividends as though such dividends contained no proportion of returned capital, so long would they be able to say that mining engineers tacitly endorsed the view that ore-deposits persisted indefinitely.

Mr. H. S. Denny said that the present discussion, more than any other in which he had had the privilege of taking part, hinged on the accepted meaning of certain terms, and it seemed impossible to get within clinching distance without some definite understanding on those points.

This was his suggestion: 'In any given deposit "ore" is that particular portion with which observation, experience and expediency associates in varying degree the occurrence and expectation of the particular mineral or minerals sought.'

Of necessity 'depth' must be a relative term, and the only practical measure was that fixed by the limit of exploratory operation. With improved methods of working and with the depletion of the shallower deposits, that boundary would be extended, but nothing could ever be a better standard of comparison.

For all practical purposes, he would assume that a mine was only a mine when it had proved itself a legitimate profit-earner—in other words, when it satisfactorily showed its ability to replace the capital invested in it, plus a margin over and above a given minimum rate of interest. Of such mines, taking those which were on authentic record as having existed in the past, coupled with those which were alive to-day, there were many hundreds which might be taken into calculation. Basing his conclusion on his own experience in Australia, South Africa, Europe and America, he ventured to assert that the vast majority of them died out, or were guilty of non-persistence, at or above the 2,000 ft. horizon; an appreciable aggregate number, though but a very small percen-

tage of the whole, continued perhaps to 3,000 ft.; and, lastly, an insignificant fraction had been successfully exploited to 5,000 ft.

Surely there was nothing surprising in that statement if it were simultaneously conceded that superficial concentration and secondary enrichment had played a very important part in the constitution of so many of the productive 'ore' bodies of the shallower horizon, and if it were conceded also that excessive temperature and pressure, the concomitants of depth, were unfavourable to the precipitation of the minerals which went to form those 'ore' bodies. Once the broad principle of those fundamental planks was accepted, impoverishment in depth became as necessary a corollary to the average 'ore' body as thin milk to its superimposed rich layer of cream.

For instance, in the Pachuca camp, in Mexico, the upper portions of the veins had been leached of their gold and silver contents for several hundreds of feet, in some places to as low a depth as 1,200 ft. Below that, down to say 3,000 ft., there was a middle zone of secondary enrichment. First was a case which Mr. McDermott would describe as leaving the poor end stuck up in the air. Below the zone of secondary enrichment a return to the primary 'ore' condition was anticipated, and with it a considerable impoverishment. In the circumstances it would be unreasonable and illogical to expect it to be otherwise. The crux of the question really lay in the truth or falsity of the assumption that the conditions surrounding the deposition, precipitation, or aggregation of the minerals forming the 'ore' bodies became less favourable with increase of depth. If that were so and the evidence in that direction was remarkably convincing then the assumption became an actuality. That did not preclude the possibility of finding occasional rich 'ore' bodies at a depth of even 10,000 ft., because the varying condition of porosity and permeability in various districts might easily introduce local factors which would permit of such isolated occurrences.

The average depth could be gauged by a study of past records, and without hazarding any definite figure it might be safely said that non-persistence in the average case would take place at a level well above the limits of practical working; and after all the average was what counted, not the exceptions.

Moreover, it was a most fortunate thing from every point of view that 'ore' bodies were more persistent in their longitudinal than in their vertical range. Imagine the Rand as one shoot, 1,000 ft. long and 80 miles deep! Climbing out of that mine when the skip was off would be a tragedy. As it was, the Rand had proved itself the most wonderful of all gold deposits, and he was sure that on the question of persistence in depth it had their united best wishes.

He would not join issue with his friend Mr. Horwood. The Rand, perforce, would speak for itself as the 'ore' in the shallower horizons became more and more depleted, and he who ran might read. He firmly believed that they might count upon the Rand to produce more than one instance of very considerable persistence; and as Mr. Horwood had pointed out, 'as "some" of the outcrop sections were unpayable, why not "some" of the deeper ones?' Like the author's term 'numerous,' the word 'some' must be given for the time being a certain elasticity.

Mr. J. Morrow Campbell said that the author in his paper had given a prominence to the asserted diminution of value in depth on the Rand to an extent that seemed somewhat inadvisable. The origin of the gold in those sedimentary beds was a matter of dispute, and he ventured to say that in view of the almost unique nature of those deposits it would have tended to simplify the question if he had confined himself to the consideration of what they all accepted as veins in the strictest sense of the term.

As every prospector knew, there were multitudes of veins which yielded commercially valuable material at or near the surface. That the vast majority of those became unremunerative, or even disappeared entirely, within the first few hundred feet was a fact; therefore it was simply banal for anyone to make the assertion of non-persistence of ore in depth in reference to the vast majority of veins outcropping on the surface.

Apart from the enrichment arising from secondary causes, which in many cases got credit for having converted gangue into ore, they had not much reason for saying that a horizon parallel to the present surface and, say, 500 ft. below would present fewer outcrops than were now found exposed.

Exploring for small valuable lenticular veins at 500 ft. was too expensive a matter for any engineer to recommend as a prac-

ticable scheme. When the lens of ore exposed on the surface had been worked out, the claim was usually abandoned. There was abundance of reason for believing that such lenses repeated themselves many times, not immediately below one another, but in the near vicinity: geological structure and movements of the surrounding country often gave a clue as to the probable direction in which search should be conducted.

There was, however, another class of veins in which he held there was every reason to believe ore would be found beyond practicable working depths. In certain mines in the Lake Superior copper region at the greatest depth yet attained the ascending water was found carrying copper in solution. The author quoted also that in the St. John del Rey mine at 3,126 ft. water was found carrying in solution all the constituents of the lode. Such lodes were apparently still in course of formation. The source of the ascending water was not known, but it was probably at a depth which mining would never reach.

The whole question at issue was bound up with the theory of the origin of veins. That they should find on the surface a multiplicity of, say, auriferous lenses of quartz such as occurred in Western Australia or on the Gold Coast, and that at, say, 500 ft. they should not also find them, seemed illogical. A negative must not be asserted simply because a positive had not yet been proved. Was it not much more probable that those outcrops represented, so to speak, the ends of branches, the result of the splitting up of an upward current of auriferous solution, which at some depth would be represented by veins of a more substantial nature? The theory of lateral secretion was only a make-shift, and there were the best of reasons for knowing that many veins had been filled from below.

Another class of mineral veins came under the name of magmatic segregations. It sounded very nice and learned, no doubt, to talk of the molten magma and eutectics, but their knowledge would require to be increased enormously before they could explain rationally the puzzling phenomena which nature presented in that direction. For example, in what condition was the magma which permitted the formation of crystals of orthoclase over 1 ft. in diameter and some 3 ft. long in a pegmatite vein of less than 3 ft. in width?

The theory of the origin of veins was in its infancy, and he was persuaded that they did not know enough to permit them legitimately to generalize as the author sought to do.

Honesty of purpose in mining must always be aimed at, but was it to be attained by rash assumptions? The practical mining man had enough worry and work as it was without the present threat of condign punishment if he ventured to doubt that the bottom of a lenticular deposit was the point at which the rocks at each side refused to exude any more gold.

The mining geologist was a man whose services in the future would become of ever-increasing value. Let them search for the truth, and not by means of rash postulates discourage legitimate enterprise and damage a great profession.

The sceptic in religion had been the main factor in advancement, though his theories usually carried him to extremes. Should they not say that the author's scepticism, while it would tend toward the advancement of mining, had forced him to unduly pessimistic conclusions?

Dr. J. W. Evans thought that the views of a geologist on the question under discussion might be interesting. He was very glad to see that the theory of the magmatic origin of fissure-vein deposits was so strongly supported in the mining world.

The deposition of ore depended upon the existence of favourable conditions. These were, the amount of material arising from below; the temperature, the pressure, and local circumstances such as the nature of the country. It seemed to him that the most important was temperature. So long as the surrounding rocks were sufficiently hot there would be no deposit, however rich the materials which were ascending, and this established a lower limit of deposition. As the conditions became more favourable, it increased and reached a maximum, above which it gradually decreased and came to an end, when the material was exhausted. The limits of deposition depended, therefore, largely on temperature, and the lower the heat gradient the greater would be the range in depth of deposition, though this might be somewhat extended if the amount of material were greater, and the distribution of the deposit might also be modified by the character of the country.

As a general rule, therefore, if one could examine the region shortly after the

material was deposited, when there had been no subsequent earth movements, one would find the deposits in neighbouring lodes practically at the same level.

If, however, as the result of folding or faulting, one portion of a metalliferous region were depressed relatively to another, this would no longer be the case and the level at which the deposits commenced, reached their maximum and came to an end would vary from point to point. If, under these circumstances, the surface were worn down to a plain of erosion, there would be a number of different cases. Some deposits would be intersected by the surface near their lower terminations. These would be of small account event at the surface and soon pinch out, when they were followed downward. In other places there would be lodes that were intersected where the deposit was at a maximum, and in these it would extend some distance down in gradually diminishing amount; while a third class would exist in which the deposit near the surface appeared to be quite insignificant, but, when followed down, increased in value for some distance before, having reached its maximum, it diminished and finally disappeared.

If that were all, it would be possible, by determining what earth movements had occurred since the deposits of a metalliferous area were accumulated, and where the rocks had been let down, to point to comparatively insignificant deposits which there was good reason to believe would show an increase in value in depth.

But, unfortunately, other considerations came in. He believed that many deposits were of very vast antiquity. The idea that most metalliferous deposits were of recent date was entirely unfounded, and in the long ages that had elapsed since their formation many changes had occurred.

Everyone was familiar with the process of secondary enrichment due to the action of surface water containing oxygen. That caused an increase of value at comparatively shallow depths, so that there was a probability of poverty lower down. But, wherever there was a circulation of water, however slow, whether it was surface water containing oxygen or not, there was redissolved solution and deposition as the conditions varied. This was specially characteristic of auriferous quartz, though it also occurred in the case of other metallic deposits. As a result of a well-known physical principle, there was always a tendency for solution to take place in regions of high pressure, and deposition at regions of low pressure.

There were many examples. He believed that the auriferous quartz of the Bendigo deposits represented original placer deposits. When these were thrown into isoclinal folds, solution took place in the straight portions or limbs, where the pressure was at a maximum, and deposition in the saddles where it was least. The gold, too, of the Rand was, he believed, originally scattered through the beds now represented by quartzites and conglomerates, but, subsequently, as the result of the alternate solution and deposition, it was largely collected in the intervals between the pebbles which were then points of minimum pressure. In the case of metalliferous lodes, the region of least pressure was usually that nearest the surface. The deposits would therefore tend to creep upward, and there would be a gradual enrichment near the surface at the expense of the lower points of the lodes.

His verdict, therefore, would be that from the geologist's point of view there was a very great amount of truth in the author's position that the scales were heavily weighted in favour of the region near the surface; but that, nevertheless, especially in areas where the deposits were comparatively young, a man who had studied the geology of the district should bear in mind the possibility, that, where there had been a depression or sagging of the rocks since the deposition of the ores, there might be a considerable downward increase of deposits represented by minor surface exposures. But that was the exception, the very great exception, he believed, and not the rule.

Mr. T. A. Rickard, in reply, said: Mr. President and gentlemen, I appreciate the introductory remarks of the President. There is no doubt that the title of the paper might have been 'The Impoverishment of Ore in Depth,' or perhaps even better, it being presented to the Institution of Mining and Metallurgy, 'The Continuity of Mining in Depth,' because, after all, I consider ore as the objective of mining. I do not consider it from the point of view of the mineralogist or the chemist; I consider it wholly from the economic point of view, and I have said so again and again.

May I express our general thanks to Mr. Marriott for showing us that longitudinal section, and giving us so much interesting information. I hope that my action in eliciting that information will not be con-

sidered an undue trespass on his courtesy. I do not care to argue with him on the matter. He was kind enough to let me see that diagram, but, as I said at the last meeting, I nevertheless came to a contrary conclusion. I am sorry Mr. Marriott is not here, but I have only this to say in addition, that the red that you saw on that map indicated a given grade of ore. He did not wish to state, and, although I know it, I shall not state, what it was. That section does not indicate ore richer than a figure considerably below the average of the richest mine: it only indicates variation from that point down. Of course, for the purpose of this argument we want the qualitative record up as well as down.

Permit me to thank you for the kindly manner in which you have received my effort to throw light upon an obscure problem. I desire more particularly to express appreciation for the contributions to the discussion. The greatest compliment that can come to an author is thoughtful criticism. I shall endeavour to return the compliment in kind.

Huxley said: "Spencer's idea of a tragedy is a deduction killed by a fact." Gentlemen, you may not be aware of it, but you have been present at the obsequies of a pathetic fallacy. The *coup de grace* was a sabre-thrust of Irish wit delivered by one of our most honoured members. We have reached the point where we recognise the beneficence of Providence in placing the better parts of ore deposits within easy reach of man.

Several members have demurred to the definition of ore given in my paper. My honourable friends, Messrs. McCarthy and Trewartha-Jones, have expressed a not surprising preference for their own definitions. But, I submit to you, it was necessary for me to define the meaning within which I proposed to discuss the question of persistence. In regard either to metalliferous substances attractive to the miner or to elements required for the world's use (I do not quote my critics verbatim), I can only say that the persistence of such material does not interest me; that involves a problem much too academic. Mineral aggregates that are attractive or that contain useful elements are likely to persist to a depth far beyond the operations of such eminently practical engineers as the two members mentioned. Frankly, I had to state my case before we could begin this discussion, and my case concerned only the limited portion of mineral deposits that can be exploited to economic advantage.

Economic advantage usually entails pecuniary profit, but not always: Governments may find it necessary, as in Hungary, to operate mines in order to find employment for their people; a manager may find it advantageous to break poor stuff, and, having broken it, to treat it as ore, so as to reach better ground. This not infrequently is the case in alluvial mining. But the exceptions only emphasize the rule, which is that the miner's ultimate purpose is to make money by the exploitation of such mineral aggregates as yield more than the total cost of the necessary operations.

While objection is made to my limitation to the meaning of "ore", it is suggested that I have failed to give a sufficient limitation to "depth". Here, again, it was a part of my case that "depth" cannot be stated in feet, simply because the lower limit of every orebody has not yet been reached. Prophecy is the test of a scientific theory, it is true; but I offer no theory: all that I have done is to collect facts and to sift them with a view of proving that the idea of general persistence is a false generalization. Bossuet said that the greatest aberration of the mind consists in believing a thing because it is desirable. That is what the operations of mines have done, while engineers have stood aside in polite acquiescence. "Depth" is a relative term: when it ceases to be so, as it will in the future, we shall be able to replace it by a measurement in feet. Meanwhile, I accept such an example as the Ivanhoe mine at Kalgoorlie and say that while Dr. Maclaren may have ascertained that a particular intrusive rock is favourable to the enrichment of the Great Boulder lodes, my view is that the congenial character of the rock is likely to prove a less decisive factor as further depth is attained. To put it plainly, the lode is more likely to be productive when traversing this particular rock at 1,000 ft. from the present surface than when traversing the same rock at 4,000 ft. The factor of depth, in a relative sense, has entered the problem.

The reference to the Flat lode in Cornwall is useful. According to Mr. Trewartha-James this lode was poor down to about 900 ft. in depth; it was rich from 900 to 1,500 ft.; and now at 2,000 ft. it shows "diminished productivity". I beg to demur to the statement that this is "a good sound instance of remarkable enrichment in

depth". When a mine has gone down to 2,000 ft., its workings between 900 and 1,500 cease to be "deep". Using depth in the sense postulated in my paper, the ore in this lode has given signs of impoverishment in depth.

The Britannia, I presume, is an example of secondary copper enrichment, and it was probably the recognition—this is a mere conjecture—of the possibility of such enrichment, which is not related to great depth, that led to the successful development described by Mr. Trewartha-James.

The limited number of mines worked to "any appreciable depth," as depth is considered to-day—that is, to 4,000 or 5,000 ft. vertically—should, it is true, render us cautious in coming to a definite conclusion, as Mr. McCarthy suggests; but I submit that the particular conclusion to which we should not come hurriedly is that mines generally can be exploited advantageously to such a depth as the greatest yet attained.

We are, it seems to me, justified by the small number of survivors in concluding that the survival is unusual. In an exceptionally rich mineral region, like Colorado for example, only one location in 500 warrants patenting, and I venture to say that out of 100 prospects strong enough to justify the expense of patenting only one becomes a mine of pronounced vitality, that is, productive to, say, over 1,000 ft. in depth. That means that 1 out of 50,000 claims covers an orebody relatively persistent.

It is true, as Mr. McCarthy says, that thirty years ago the probability of the Mysore or the St. John del Rey mines going down to their present depth was slight; nevertheless, was it not a fact that such probability was imputed not only to them but to a thousand others now abandoned and forgotten? Of course, in attempting to destroy a generalization, it appears as if a counter-generalization were being formulated. However, the denial of an old one need not require the affirmation of a new one. That you will, I trust, grant, even it my argument may seem to have led me from one extreme to the other.

An indication of non-persistence is afforded by the discontinuity of outcrops, as is suggested both by Mr. McCarthy and by Mr. Terrell. Yes, the best proof of indefinite persistence would be a circumterranean outcrop. Failing anything of the kind, we may well pause to consider how discontinuous are the superficial exposures of ore along the strike of a lode. Any attempt, however, to establish a ratio between length along the strike and depth along the dip is extremely hazardous. The depth of a crack may be in direct ratio to its length in a homogeneous rock, but orebodies in their extent are not limited merely by the fissures of shear-zones they occupy, but by the penetrative power of chemical solutions and other factors of a complex character.

The Main Reef series of the Witwatersrand may be mined continuously for 55 miles, but any inference therefrom that the banket can be exploited profitably to a depth, say, one-fifth of the length, is rendered ridiculous by the facts already available from workings barely one-tenth of the depth suggested.

It is true that the Rand engineers claim, as Mr. Olver has done in this discussion, that the ore deposits of the Rand are *hors concours* and must not be judged by observation elsewhere. That, I can say from wide travel, has been the mental attitude of intelligent persons in many other celebrated mining districts. It is an attitude stultified by experience.

Professor Frecheville thinks that I have given a one-sided view. That is fair comment. I think it is likely that I have. I was combating one interpretation of the problem and naturally inclined to its opposite. As regards my failure to offset the Village Deep with the City Deep, that was not done by inadvertence. The Professor may have information not accessible to me, but the information received by me from Johannesburg includes the following data:

Level	Depth.	Average Assay.	Average Width.	Inch Dwt.
		Dwt.	In.	
No. 2 ...	2,137	44·9	21·6	970
No. 6 ...	2,725	13·8	30·9	422
No. 10 ..	3,360	11·2	35·8	401

The outcrop mines above the City Deep are the Meyer & Charlton, Wolhuter, and New Goch, in order eastward. It appears to me that a comparison between the richness of the City Deep workings and those of the three mines mentioned does not warrant the idea that there has been an enrichment in depth. If it can be proved, it were well to do so.

As regards the Champion Reef mine, the longitudinal section of the stopes is quantitative, not qualitative. According to my argument, an improvement in the richness of the vein at 4,000 ft. is not so likely as a similar improvement at 2,000 ft. in depth. My reason for thinking so is that experience indicates that a mine's chances of recovery decrease with depth, as, by analogy, a man's chance of recovery from serious illness becomes less as he grows older.

In regard to the North Star, the production per square foot has increased, but this is due in part, as stated by me, to better metallurgical treatment. The statement that "the miner has never failed yet to bottom any ore deposit he has found" is true, because the few orebodies still persisting to great depth have not been abandoned by the miner, and every reasons exists for believing that he will work them to their lowest limit. Professor Frechéville misunderstood me, that is all

Mr. Turner and Dr. Cullis note that I have made no reference to deposits of magmatic origin. Theoretically it may be granted that the mode of origin of such deposits does suggest a persistence deeper than that of most other types of mineral aggregate. Yet the information available concerning contact deposits, as elucidated by Professor Kemp, for example, is significant. However, I am not discussing the persistence of mineral masses of magmatic or of pneumatolytic origin: I am discussing the persistence of those relatively restricted portions of them that have been concentrated by thermal solutions, more particularly the small number that have commercial value and come under the definition of 'ore' postulated in my paper.

As I listened to the clear exposition of the Professor of Economic Mineralogy I felt almost persuaded that he had placed his index finger on a type of ore deposit that belonged to the super-Methuselah class, but, on passing from under the spell of his scholarly speech, I remembered that no orebody of known magmatic origin had as yet been traced to the barosphere and that he was dealing with gay potentialities, not grim actualities. When you define 'ore' in terms of economics, you clip the wings of the geologist's constructive imagination.

The decreasing width of the banket beds of the Rand is an interesting point, brought forward by Mr. Horwood. Having regard to his intimate knowledge of the Main Reef series, I may be permitted to express sympathy with Mr. Horwood in his courageous insistence that the Rand is a unique goldfield; but I demur to his argument that the past production, or even the continued productivity, of the Rand, as stated by him in general terms, is evidence on the point at issue.

The fact that the Rand may, and I hope will, yield at least as large a total output of gold in the future as it has in the past does not throw direct light on the problem as to whether the banket lodes are richer or poorer in gold with increasing depth. The low-grade productivity of a mine usually lasts longer than its bonanza period and often yields more metal, but at less profit. It do not care to press my contention—in which I am supported by several engineers of the highest reputation—that the banket lodes give signs of impoverishment with depth: in this disputation I can claim, like the armies of the Allies, that General Time is on my side. *Qui vivra, verra.*

To Mr. Mennell I proffer sincere condolence on the spoiling of his frank and outspoken utterance by transcription from the first to the third persons. The sort of plain speaking in which he most properly indulged receives no literary justice by being recorded in the conventional phrasing of a skilful reporter. Scientific evidence ought to be given with the egotism of a witness. However, I have a delightful recollection of Mr. Mennell's delivery of his opinions, based upon wide and penetrating observation, and I am grateful to him for enriching the discussion by a contribution so much to the point. Certainly, as Mr. Mennell suggests, the great mines considered in detail in my paper represent notable exceptions: it would have been cruel to record the unhappy stories of the much larger number that would have proved my contention overwhelmingly.

Incidentally, I may say I have not mentioned railway tunnels, and yet most of you know that some of the most productive mineral regions have been penetrated at depth by tunnels, in the Alps, the Rocky Mountains, the Andes, and so forth, and among the rare cases of discovery of ore in railway construction I can only recall one at Cobalt and another at Copper Rock in Colorado.

I ask you to recognize that my treatment of the subject, at least as regards notable illustrations, has been kindly, not corrosive. But before I close these remarks I shall trespass on your geographical knowledge,

on that geographical knowledge that in its exuberance has been able to imagine the transfer of Russian troops from the White Sea to France by way of Scotland, and has conceived the transport of Japanese reinforcements from Yokohama to Warsaw; I shall ask you to go with me in retrospective imagination to famous points of vantage from which whole mining districts can be seen almost at a glance. Come with me then to Dolliver mountain, in Nova Scotia, and to Red mountain, in British Columbia; linger a few minutes on Carbonate Hill, Leadville, or Bull Hill, Cripple Creek. Proceed thence to Columbia Mountain, commanding a view of Goldfield, or Mount Davidson, overlooking the mines of the Comstock lode; go southward to the top of the Sierra de Santa Rosa, or linger on the Cerro de Proaño, both in Mexico; continue southward until you can look at the old mines at the foot of the Cerro di Potosi, or those visible from the Cima de Chorolque; cross the seas to the famous Victoria Hill, at Bendigo, or Bunninyong, near Ballarat; turn homeward and pause on the Snowberg or the Rammelsberg; alight on the Puy-de-Dome or the Chalanches, and then end your circumspection by resting on the top of Carn Brea or Carn Marth; look carefully at the gaunt engine-houses that punctuate the landscape like a big cemetery of bygone industrial life! I ask you to consider what it all means, and then I turn and say to you: *Si argumentum requiris, circumspice*.

CONTRIBUTED REMARKS.

Mr. C. Baring Horwood: Before further discussing Mr. Rickard's statements, I would like to take this opportunity of recording my great appreciation of his valuable paper. The free discussion among mining engineers, of the important subject with which it deals can hardly fail to exert a beneficial influence on the profession.

At the meeting held on November 1911, Mr. Rickard stated that the Adalbert shaft at Przibram, in Bohemia, had reached a depth of 3,640 ft. and that the mine had been idle for many years; that work had been discontinued when the mine became unprofitable; that the ore followed fissure-faults associated with diorite dikes in Cambrian sandstone; and that the ore in the deepest workings showed the same composition as in the upper levels, only it was poorer.

At that meeting I mentioned that this mine was still being worked when I visited it in the early part of 1911, and at a greater depth than that mentioned by Mr. Rickard, but the ore had become considerably poorer, the galena was more interminglcd with quartz, and the percentage of silver was less than formerly. As I wished at the time to discuss only the portion of Mr. Rickard's paper dealing with the Rand, I promised to refer later to the Przibram mines.

The ore in the Przibram mines has not the same composition in the lower as in the upper levels. In fact, one of the most remarkable features of these mines is the difference between the ore in the upper and lower levels. This difference becomes more accentuated as greater depth is attained. For the first 500 m. from the surface it consisted of galena, associated with a little zinc blende and some siderite, quartz and calcite; it contained some 20 oz. and upwards of lead, and the silver contents varied up to about 0·5 %. Below this depth the character of the ore gradually changed, the quartz filling became more pronounced, and the galena more and more interminglcd with quartz, until the ore assumed the nature of a compact granular, intimate, mixture of quartz and galena.

This latter ore is of a light to dark grey colour, of a fine-grained granular texture, and of a quartzitic nature, containing from about 42 to 48 % of silica. The galena occurs in it in finely disseminated granules, and is associated with much siderite. Boulangerite, zinc blende, arsenical pyrite, iron pyrite, and sometimes a little calcite, chalcopyrite, tetrahedrite, diaphorite and pyrargyrite occur scattered through the matrix. Pure silver, stephanite and bournonite are also said to have been found. It is interesting to note that traces of tin are present: seven samples taken from the 17th, 22nd, 27th, 29th, 30th, 31st and 32nd levels of the Adalbert mine gave 0·02 to 0·2 % of tin.

To distinguish it from what may be conveniently termed the 'galena ore' of the upper parts of the lodes, the name *durrerz*[1] has been locally given to the quartzitic ore of the lower portions. This *durrerz* contains about 12 to 17 % of lead and about 0·1 to 0·3 % of silver.

Thus, as compared with the 'galena ore' of the upper levels, the most striking features of the *durrerz* of the lower levels are: the decrease in the proportion of galena and

[1] "Ueber Durrerze von Przibram," by A. Hofmann and Slavik, p. 10.

[1] The literal meaning of this is 'dry ore.'

silver; the high percentage of quartz; the presence of much siderite; the granular character of the matrix and of the galena; the fine dissemination of the latter through the matrix and its intimate association with the quartz; the increase in the amount of sulphides present, other than galena, such as blende, and iron, arsenical, and copper pyrite; and the number of antimony-sulphide compounds of copper, silver, lead, and also of lead and silver.

The larger and more important ore-veins of the district strike north and south and occur *in diabase* dikes; the smaller veins trend north-west and south-east, and are in the Cambrian quartzite. The chief lode of the district is known as the 'Adalbert Hauptgang.' Among other important ones might be mentioned 'Fundgrubnergang,' which occurs on the east side of the first-mentioned vein. The north and south veins sometimes leave the igneous rock for short distances and enter the quartzite. The lodes vary in width from 6 in. to 4 ft. and upwards. The average thickness of the 'Adalbert Hauptgang' is about 1·2 metres (about 4 ft.) and driving has been done on it for a distance of about 1½ kilometres (4,920 ft.). The southern 500 metres are not so good as the 1,000 metres to the north. The northern end is cut off by the main structural fault of the district, which is known as the 'Letten-kluft' (clay 'fissure).

As regards the depth of these mines, there are five principal shafts, the Maria, Adalbert, Prokop, Anna and Franz Josef; and at the time of my visit, in March, 1911, the last two were down to the 32nd level, or 1,100 metres (3,609 ft.) below the surface; the Prokop had reached the 31st level; and the first two were down to the 33rd level, that is 1,150 metres (3,773 ft.)* below the collar of the Prokop shaft.

At the Adalbert shaft, which is the particular section of these mines to which Mr. Rickard referred, work was being done from the 25th to the 33rd level; but none above the 25th level. I was given to understand that in two years from then this shaft would have reached a depth of 1,200 metres (3,937 ft.).

The mines were being worked, although there is considerable impoverishment in depth; and Capt. A. Wentworth Forbes, British Consul for Prague, informs me that the mines were still being worked when he left Prague on the 23rd of last June, shortly before the outbreak of war.

They furnish a good example of considerable persistence of ore in depth, when the extent along the strike of profitable outcrop is great.

Mr. Rickard expresses sympathy with me for what he terms my courageous insistence that the Rand is a unique goldfield. Mr. Rickard knows that the point on which I have so frequently insisted is that the Rand auriferous conglomerate, or banket, differs in no important *essentials* from ordinary gold-quartz lodes—its uniqueness depends on the great extent of profitable ore along the strike and the great average extent of profitable ore along the dip; and on the fact that ascending solutions selected, for the deposition of their mineral contents, channels afforded by well-defined pebble beds, instead of, as is usually the case, ordinary fissures.

Mr. Rickard adds: 'I demur to his argument that the past production, or even the continued productivity of the Rand, as stated by him in general terms, is evidence on the point at issue'; and he goes on to express the hope that my forecast will be fulfilled that the Rand will yield at least as large a total output of gold in the future as it has done in the past; and he then says that the fact that it may do so does not throw direct light on the problem whether the deposits are richer or poorer with increasing depth. This suggests that Mr. Rickard is under the impression that I consider there is no impoverishment in depth. I would therefore refer him to the opening remarks of my previous discussion on his paper, in which I expressed surprise that there should be any need for a thesis to champion the opinion, which I regarded as no mere theory but as an established fact, that there is impoverishment with depth in the case of veins having an origin similar to that of ordinary gold-quartz lodes. I not only consider the Rand deposits come under this category, but also that they must be studied in the light of analogy with others, in other parts of the world, that also fall under the same heading.

In a book of mine published in 1905, I stated that 'it is found that the average gold values of the Main Reef series for the whole of the Witwatersrand decrease slightly in depth,' and I went on to explain that in these deposits, formed by ascending deep-seated mineralizers after the manner

* The level of the Mediterranean Sea is 550 metres (1804 ft.) below the collar of the Prokop shaft; so the 33rd level is 1969 ft. below this sea level.

of ordinary gold-quartz veins, such decrease was 'only to be expected.' Also recently, in 1913, referring to the Rand banket, I stated that 'as in all deposits from mineral solutions (either liquid or gaseous) that have ascended directly from great depths, or that have been given off as hot primitive solutions by cooling igneous intrusions, a fall in the grade of the ore with depth is to be expected.'

It is on quite another matter that I joined issued with Mr. Rickard—his remarks about the Rand were so worded, evidently intentionally, as to suggest that the limit in depth of profitable ore had already been reached. I showed that this was not proved merely by the fact that, owing to native labour shortage and high working costs, the two deepest mines had temporarily suspended operations. These two mines are situated in admittedly poor areas of the Rand, and I pointed out that Mr. Rickard had omitted to mention that even certain outcrop mines had for the time, and for similar reasons, ceased working.

On the Rand (although in development work samples are taken at 5 ft. intervals), however good the ground, it is usual, when a drive is sampled, for some of the samples to yield assay results that are below the pay limit. Similarly, at any depth, occasional stretches of ground can be found along that horizon that are either too poor ever to be worked, or too poor to be worked under the particular conditions existing at the time. Yet, whenever mining has been sufficiently extensive, it has been found that the average value of the remaining ground is above the pay limit, and that the horizon, considered as a whole, is profitable.

From this it follows that Mr. Rickard's implied conclusion is not based on sound premises; and one has good reason to believe, as I previously maintained, that the economic limit in depth has not yet been reached.

Mr. S. J. Speak: The question we are apparently asked to consider is why valuable orebodies appear not to persist to indefinite depths, to which I would reply by another question: why should they?

If we find valuable ore for a distance of 1,000 ft. at the surface, I contend that we have no sound reason for supposing that it will extend to enormous depths, but rather that its vertical dimension will not be greater than 1,000 ft. If we see a section of citron in a slice of cake, we must not feel certain that it will extend right through the slice. This analogy is not a good one, though not so far-fetched as it might seem at first sight.

Whenever the miner works an orebody for a metal we must recollect that in all probability there exists a considerably greater amount of the same metal in the neighbourhood that he cannot work profitably: he can only work for a metal where it occurs in an abnormally concentrated form. The author informs us that in Colorado, of 50,000 prospects discovered, only one became a mine.

I submit that we have no grounds for supposing that mineral concentrations of commercial value would occur otherwise than of a circumscribed character, more or less lenticular, until we have ascertained that particular geological conditions favourable to precipitation are present in one direction more than another.

It is important to notice that we have not under consideration whether mineral deposits in general persist in depth, but whether the rich portions thereof (or what the author calls 'ore') persist in depth. The chief question therefore seems to be—what is the cause of the valuable concentration of minerals in some mineral deposits?

Valuable ore is often found at depths of 5,000 ft. below the surface that existed when the deposit was first formed: such enrichment cannot, therefore, be purely a surface phenomenon. Though we know that many deposits that were rich at the surface became poorer in depth, we have for obvious reasons not an equal amount of evidence where deposits poor at the surface became richer in depth; notwithstanding which, we are not justified in supporting the author's conclusion that the surface is the main factor in producing valuable concentrations of minerals.

My points of view are (1) that though we find a valuable segregation of minerals at the surface, it by no means follows that it is a surface phenomenon further than that its proximity to the surface has enabled us to discover it; (2) that whether we discover a segregation at the surface or at great depths, we have no right to expect that it will persist in depth of equal value, but that, being an abnormal development, it must tend to verge into the normal.

It is not necessary for us to know in what manner valuable mineral deposits were produced to enable us to declare that the more abnormal they are in richness the

more quickly they will probably deteriorate and approach the normal. The more extensive they are at the surface the greater the prospects of their extension in depth if geological conditions do not alter.

Throughout the preceding remarks I have intentionally left out of consideration the oxidizing and leaching effects of surface waters in order to clear the main question from extraneous complications.

Mr. W. F. A. Thomae: Dr. Evans, in speaking of the saddle reefs of Bendigo, ascribed the deposition of ore at the apices of the saddles as due to release of pressure at those points.

It is well to be quite clear on this subject. Pressure is often loosely coupled with temperature, when speaking of ore deposition, as if it were of equal importance; as if reduction of pressure had a similar effect to reduction of temperature in facilitating deposition from solution.

Prof. Thorpe, in one of his books on chemistry, says (I am quoting my notes, not the text): 'If the solution of a salt is accompanied by expansion the solubility is decreased by pressure. If by contraction, the solubility is increased. The difference in all cases is, however, very small.' Seeing then that the effect of pressure on solubility varies in opposite directions according to the character of the solution, and in any case is insignificant, release of pressure cannot be considered to have any appreciable *direct* effect on ore deposition, such as lowering of temperature has. *Indirectly* it is of importance, as in this case at Bendigo, where it loosened and fractured the rock along the lines of bending of the saddles, making it more permeable to the ore-bearing solutions, and thus giving space for deposition, and assisting metasomasis.

While, therefore, lowering of temperature directly affects ore deposition, lowering of pressure can only be considered as an indirect agent by its effect on rock structure.

Mr. A. R. Weigall: Whilst I warmly appreciate the great merit contained in Mr. Rickard's paper on the 'Persistence of Ore in Depth,' yet the thought has occurred to me that perhaps in one feature this paper creates an erroneous impression, in so much as it suggests that the modern mining engineer is not fully conversant with the risk of the non-persistence of ore in depth, and in writing this portion of his paper Mr. Rickard has assumed a need for reform which does not exist.

Not only in mining reports, but also in the actual operation of mines, the usual practice of mining engineers proves that the risk of non-persistence is always taken into account. It is the almost invariable custom for mines to be exploited from levels only 100 ft. apart in depth. The advantage to be gained and the economy involved by increasing this dimension to, say, 1,000 ft. is fully apparent, but by only advancing the mine workings by successive depths of 100 ft. the mining engineer shows that the chance of the non-persistence of ore in depth is not only recognised but is likewise provided for in general practice, and, to say the least, this factor is certainly not as a rule 'either carelessly ignored or perversely misinterpreted.' If it be deliberately ignored by charlatans for dishonest purposes, or otherwise, it does not alter the fact that the mining profession as a whole is quite alive to the risk of non-persistence, as shown not only in their mining reports, but also in the manner in which it is their practice to lay out the workings of a mine.

Mr. H. F. Collins: Examples to confirm Mr. Rickard's view can be found by the dozen, instances to the contrary being quite exceptional.

As examples may be instanced all the important mines in the West of England, including amongst those still working, Dolcoath, Carn Brea, East Pool, Levant; and amongst those abandoned, Botallack, the United and Consolidated series of mines, Devon Great Consols, etc. All the mines of North Wales *without exception* have shown a marked falling off in value with even moderate depth, the same is true of Foxdale and Great Laxey, in the Isle of Man, the Greenside mines in Cumberland, Leadhills, and in fact all the well-known mines in the British Isles.

Most of the important metal mines in Spain illustrate the same experience. One may instance, in the case of lead mines, the Horcajo, San Quintin, Sierra Almagrera and others; also the Almaden quicksilver mines. The same is true of a majority of the great silver mines of Mexico, in Guanajuato, Zacatecas and Hidalgo, also of the only important group of gold mines in that country.

So many are the examples in favour of the view enunciated, and so few the exceptions, that but few competent and practised observers will be found to deny its truth as a general proposition.

I should like, however, to point out one direction in which the results of experience are likely to lead to an exaggerated view in regard to the impoverishment of lodes in depth. Just as the author argues that mineralization in a lode is less persistent or permanent than the lode itself, so the writer would emphasize the fact that ore-shoots are, generally speaking, less permanent than the mineralization of any given horizon. Upon this point I would venture to quote some remarks from a contribution of my own, made no less than 20 years ago to the discussion of a paper by Ed. Halse on 'Deep Mining in Mexico' (*Trans.* iii., pp. 141-2), as follows:

'Considering the accidental position with respect to mineral veins of the present surface produced by sub-aerial denudation, it seemed to him that (apart from surface enrichment) they were just as likely to find one vein, rich at surface, get poorer in depth as they were to find another, poor at surface, get rich in depth. Of course, those poor at surface would not be so likely to get worked, and therefore it might happen that in the majority of cases veins rich at surface would appear to become poorer in depth, merely because those which were evidently poor at surface did not get worked so much.'

Individual ore-shoots, sooner or later, do invariably become impoverished in depth, but cases are extremely common, and examples have occurred in the experience of every one of us, in which the shoot which was the mainstay of the output from a particular mine has pinched, and either at the same horizon or a little deeper another shoot which did not outcrop has come to take its place.

What does very often happen in the history of a mine is that the principal shoot is worked down to a point, another which does not outcrop is also worked down to a point, and perhaps a third similarly; and then some day or other, just about the time when a shoot has been worked out, there is an increase in the coming water; or the limits as regards depth of the existing pumping installation have been reached; or the price of metals is abnormally low; or the funds of the company have been depleted in keeping up good dividends for a few years by dint of forcing the production and 'picking out the eyes'; or it turns out that development has been neglected by the management for some time previously in order to keep down costs or to increase production with limited hoisting facilities, or for some other reason not directly connected with the permanent interests of the general body of shareholders; and no funds are available or obtainable to develop the property laterally by long drives along the lode from its short bottom levels, which for several lifts have been driven only the length of the shoot or shoots actually being worked.

In hundreds of such cases mines are stopped and abandoned because the working shoot has become impoverished, without any adequate attempt being made to drive along the course of the lode in the deeper levels to look for other shoots. In such instances, and they are very many, although it is quite accurate to state that the ore-shoots followed down have become impoverished, it is hardly fair to argue from the evidence available that the lode itself is, as a whole, less mineralized in depth.

Apart, however, from impoverishment of individual ore-shoots, the view of the author of the paper as to general impoverishment of lodes in depth is nevertheless undoubtedly true as a general proposition; and all I wish to point out in these remarks is that from the very nature of the evidence available—namely, the working of individual shoots down to a point—it is easy to get an exaggerated idea of the extent to which this theory finds support in the abandonment of a very large proportion of the mines which are no longer working.

With what is, from the practical miner's point of view, the author's principal corollary; namely, that for exploration purposes cross-cutting and driving are far more important than shaft-sinking, the writer is absolutely in agreement, having held the same view for many years. A number of arguments in favour of this view are adduced by the author; many more can be cited from the experience of every mining engineer who has had practical experience in a variety of districts.

The East Pool mine had become impoverished in the bottom many years past, the main shaft having been put down a succession of lifts, each of which turned out poorer than its predecessors. A long cross-cut had been advocated by Mr. J. H. Collins for the past ten years with the object of cutting a faulted section of another lode which was rich in the shallow working some 40 or 50 years ago. That cross-cut has been at last driven, and has cut a large and valuable lode at a comparatively shallow depth.

The Pozo Ancho mine, in Spain, had shown for years a gradual falling off in output: the main shoot had been worked to a point, as those of most abandoned Cornish mines were worked, and three successive lifts of the main shaft had shown practically no ground worth working, the property being finally sold for a song by the English company which owned it.

Under the management of the Spanish buyers, side-lodes in shallow ground were rapidly opened up, with the result that the purchase price was covered by the clear profits made in only three months' working, and since then during the past three years the output has averaged more than double as much as that realized by the English company during the last few years of its life, expenses being at the same time much lower and the divisible profits out of proportion greater. Such instances can be paralleled from almost every mining district.

Mr. J. H. Collins: Mr. H. F. Collins has referred to the comparatively poor results realized by moderately deep sinking at East Pool during the past ten or a dozen years. The sinking referred to has been on a series of secondary lodes underlying to the south which, except in small patches, have never been great ore-producers, except for copper and at depths not exceeding 150 fathoms. The Great Lode, which underlies north and which has yielded enormous quantities of tin, wolfram and arsenic, has been successively and normally faulted by these secondary lodes, 'thrown up' as the miners say. It is probably one of these thrown sections which has been intersected by the 240 fathom cross-cut, where we have lode matter about 20 ft. wide, averaging 2 or more of tin oxide. A continuance of this cross-cut will be necessary to determine whether there is still another 'heave'; in the meantime driving, rising and sinking on the lode now discovered seems likely to lead to a renewal of highly profitable working.

At Dolcoath, the great ore-shoot, which has been the chief support of the mine for so many years, while perhaps pitching away eastward towards and beneath Cook's Kitchen, seems to be failing in Dolcoath itself. Deep lateral driving westward along the line of the lode may perhaps cut into another deep-seated shoot hitherto unknown. But in such a rich 'sett' traversed by so many known lodes, or branches of the main lode, chances of important discoveries of rich junctions by cross-cuts at comparatively shallow depths would seem to have great probabilities of success, although what little has been done in this direction hitherto has not been too encouraging.

Several speakers have referred to the, perhaps inevitable, vagueness of Mr. Rickard's title—'Persistence of Ore in Depth.' As regards the West of England, something like precision is possible. Hundreds of copper lodes have been worked down to depths from surface of 100 to 300 fath. Some of the best have then been abandoned on account of poverty at depths rarely so much as 200 fath. below sea-level; others have been followed down as tin lodes, at first poor, then richer, then poorer again. How deep the payable tin zone may be expected to continue has not yet been determined, though some think that it has been reached, or nearly so, at Dolcoath, at a vertical depth of less than 3,000 ft. below sea-level.

Mr. Stephen J. Lett: Many instances could be given of what I may be permitted to refer to as the romance of prospecting, where mines, supposed to have been worked out, have been abandoned only to be re-opened and subsequently worked at a good profit. Take the case of the Great Fingall mine, at Day Dawn, which was nearly abandoned at 150 ft., when the first lens of ore cut out; fortunately, fresh capital was obtained, sinking continued, and a fresh lens cut at 400 ft., which persisted to a depth of 1,500 ft. and yielded about $1\frac{3}{4}$ millions sterling in dividends.

As another instance there is the Viking mine, Norseman District, Kalgoorlie, referred to in the *Mining and Scientific Press*, 9th August, 1913. This mine was abandoned at 571 ft. after having produced £223,000 in gold and silver, chiefly gold, from 56,000 tons of ore. The lease was restaked in 1906 and over £80,000 was won from 11,000 tons of ore taken from shallow depths, whilst on sinking to 600 ft. phenomenally rich ore was obtained. This latter example possibly serves to emphasize Mr. Rickard's plea for more enterprise in lateral prospecting, but is none the less valuable on that account.

The number of abandoned 'prospects' that have subsequently become payable mines is legion, and if the Viking mine had only been eroded a further 575 ft. it might have been added to the number.

It is generally admitted that ore deposits cannot persist indefinitely in depth, but, on

the other hand, there is no doubt that some primary deposits improve in depth, and, to paraphrase one of Mr. Walter McDermott's witty remarks, it is a merciful dispensation of Providence that there are men alive to the fact that mines may have their poor ends uppermost.

I would urge them that, after reading Mr. Rickard's able thesis and hearing the remarks of the various speakers, also assuming experience in the research for orebodies and a fair knowledge of the literature of prospecting otherwise known as economic geology, the only reasonable conclusions we can come to are these: no ore deposits can persist indefinitely in depth; but many primary ore deposits improve in depth, many become impoverished in depth, and many have recurring zones of relative impoverishment and enrichment; and, without being unduly hopeful, we may look forward to a little experience elsewhere.

Mr. E Homersham: As one notable example of gradual, but none the less certain, impoverishment in depth, the author gives us the mines of the Witwatersrand. Certain gentlemen considered that the Rand was such an exceptional deposit that it should not have been brought into the discussion. Here I beg to differ, as, if vertical depth is a deciding factor, you have on the Rand an admirable example, and one not complicated by any change in the nature of the enclosing rocks. In any case, honest discussion and consideration of known facts can do no manner of harm to any field, and certainly not to one so well established as the Rand.

The case of the Rand after its twenty-five years of strenuous life with still a long and unknown future before it is a point in favour of the argument that a geologic change in the enclosing rocks is a cause for more anxiety than mere depth alone. The author, however, makes out a strong case in favour of his contention that here, too, a gradual impoverishment in depth has and is taking place. He has Mr. Webb of the Goldfields openly on his side, and he is able to point to poorer returns from most of the deep levels when comparing them with their immediate outcrops.

Mr. Marriott was able, however, to affirm the faith that is still in him, and the diagram he showed us was most interesting; it is a pity we had not a longer time to study it. The results of current development on the Rand are not published in sufficient detail to enable many of us to form a definite opinion as to how far the lower returns are due to legitimate mining of more lower-grade ore than formerly, and how much, if any, must be put down to an actual impoverishment in depth. Fortunately or unfortunately whichever way you like to look at it, the two gentlemen in an exceptional position to offer a valuable opinion, owing to their having access to information not generally available, have come to diametrically opposite conclusions.

Professor Henry Louis: Mr. Rickard's main thesis is that the persistence of ore in depth is an exceptional phenomenon. He very properly commences the discussion by a definition of ore, and it is greatly to be desired that everyone who initiates such a discussion would follow his thoroughly logical example in this respect. My own definition, which I have given my students now for many years, differs from his in one important particular, being as follows:—'An ore is a metalliferous portion of the earth's crust which may possess intrinsic economic value *under favourable conditions.*'

Upon the whole I am inclined to give a somewhat doubtful assent to Mr. Rickard's main proposition, and I should have much less hesitation in fully agreeing with him were it not for a difficulty that has been a stumbling block with me for many years, and which his present paper has done nothing to remove.

It is possible to dispose without much difficulty of two out of the three great classes of mineral deposits and to narrow the discussion to the persistence in depth of mineral veins and of their metalliferous contents. Again, some types of veins can be disposed of with similar ease. For example, the lead veins of Derbyshire and of the dales of Yorkshire and Durham only carry lead when they traverse the mountain limestone, and, as a rule, only when they traverse certain beds of this formation. Thus, in Durham, they rarely carry lead except when they traverse limestone, and one particular bed, the Great limestone alone, has produced more lead than all the rest of the formation taken together. This bed is near the top of the formation, and although it is quite as productive in the deepest places where it has as yet been struck as it was where it was shallow, its occurrence at such depths as we are here discussing is quite out of the question, so

that, with regard to this particular class of veins, it is possible to assert definitely that they will not carry ore at great depths.

As regards other mineral veins, upon the mineral contents of which the influence of the country is less clearly marked, the evidence brought forward by Mr. Rickard shows that many have become poorer in depth. Of course, I am ignoring here the subject of secondary enrichment or impoverishment as foreign to the real matter under discussion. Genetic considerations, too, rather support Mr. Rickard's thesis. Heim considered it impossible for fissures to form in the earth's crust at depths below 15,000 ft., owing to the great pressure and to the plasticity of the rocks at this depth. None of the accepted theories of the filling of fissures by mineral matter implies that the filling of the lower portions is likely to be richer in metalliferous content than the upper, and indeed my own interpretation of them would lead me to expect the richer portions to be above rather than below the medium line, so that I should not normally expect to find mineral veins rich below say 8,000 ft.

So far, then, my own views tend to corroborate Mr. Rickard's thesis, and I now come to the difficulty that I have found in fully accepting this view.

We know that in following along a mineral vein horizontally we traverse alternations of richer with poorer and nearly or quite barren portions. If we assume that the same form of distribution holds vertically also, we should expect to find precisely the same set of phenomena as Mr. Rickard has chronicled. If the veins were first struck (disregarding secondary changes) in a richer portion, we should expect to find the deposit get poorer slowly or rapidly in depth, and finally become unproductive, when the miner would stop his operations.

At St. John del Rey the rich belt has been 5,000 ft. deep. Supposing that it now gave out, will anyone sink another 5,000 ft. through poor or barren veinstuff in order to see whether payable ore puts in again? And if this is not done, how is the theory of vertical alternations of rich and poor zones to be discovered.

It is equally evident that if a vein were first cut in a poor zone, no one is going to sink some thousands of feet in order to see whether there is or is not a richer zone to be met with in depth. In practice, when a miner cuts a poor reef he may sink 50 ft., or if it is a known rich region possibly 100 ft. upon it, and then he gives it up as a worthless prospect, but he does not go on sinking indefinitely.

This, then, is my difficulty. I do not see that the possible correctness of this theory of alternating horizontal zones has ever been disproved, nor that it is ever likely to be disproved. I can see no argument that shows it to be untenable; but, until it is definitely disposed of, I feel that it prevents my full acceptance of the thesis that the ore in veins must necessarily give out in depth.

Mr. J. H. Goodchild: Mining being regarded as a sampling of the earth's crust at abnormal places, the paper leads up to the pertinent question, does anything persist in depth. In other words, does geology, the science of 'rocks,' apply in depth: have the geologists had a fair chance to sample the crust of the earth? If the outcrops of so-called ore deposits are so misleading, why not other outcrops?

In any case we have to examine the meanings of the term 'rock' in connection with the paper, which is admittedly a departure from the 'fissure' idea, and a part of that notion was the immutability or relative stability of the 'rock' as distinguished from the 'ore'. Acquaintance with shallow mines leads to a special point of view. The fissure idea recedes to the background, and one becomes aware of its influence in geological terminology.

Such sentences as that quoted by Mr. Rickard from Professor Lindgren, 'Deposits in which the metal is of later origin than the surrounding rock,' are ambiguous. We do not know whether rock-body or rock-substance is meant. The fallacy runs throughout the whole of geological literature. It is partly the outcome of the fissure idea in observation, and on it is based the whole fissure-intrusion philosophy. I am not putting this forward as new and must apologize for tediousness, but the facts as to the migration of lime, silica and other materials are well known, and yet they continue to be ignored in the expressions we use, or are smothered up in a word like 'metamorphism'.

If we carry the above distinction in our minds in dealing with pockets, gash veins, disseminated deposits and so on, a great many difficulties are got over. The whole rock is or may be looked upon as the lode. Applied in the case of those little stringers in Nova Scotia referred to, it renders ridi-

eulons the idea that they are the stumps of veins which once extended several miles upward. It is begging the whole question to assume that erosion proceeds on those lines, without a change in the substance of the whole rock during which the veins might segregate.

In many cases 'beds' are no more beds than a piece of silicified wood is a tree. That being granted, any apparent cutting across the strata or layers does not constitute evidence of a fissure, or, in the case of a dyke, of an intrusion. This way of viewing the matter, of course lends support to Mr. Rickard's belief that all ore deposits are relatively young and that the everlasting flow of substance is more rapid and extensive than is generally conceded.

Outcrops afford evidence of processes which are left out of account altogether in the assumptions which are made when the bodies of schistose rock are spoken of as beds or the crystalline rocks as intrusions. Colloid chemistry is gradually throwing light on those processes, and beyond that is giving us images unthought of in nineteenth century geology which is founded on distinct ideas as to mechanics and chemistry. It is not necessary to prove that rocks contain gels before going on to speculate whether a large body of rock may not behave like one, and exhibit phenomena analogous to those attributed to selective absorption for instance.

Mr. Rickard states very roundly that ore deposits are deposited from solution. But if the whole mass of rock is the 'solution' or the medium, then there is no necessity for thinking about fissures or channels or pores in the now-existing rock, for after the precipitation has taken place, the substance of the encasing rock is not the same as it was before.

In any case, returning to Dr. Maclaren's exception, the pre-Cambrian rocks, it seems to me that the whole area round the St. John del Rey mine can with profit be looked upon as a kind of huge disseminated deposit, before we divide it up into 'rocks', before we take apparent superposition to imply difference of age, before we assume that schistosity has anything to do with bedding, before we assume that the crystalline rocks are the igneous agents of metamorphism rather than its extreme phase, and, consequently, before we assume anything about fissures in connection with the St. John del Rey lode.

Dr. Maclaren has, of course, made a far more systematic study of these rocks than I have, and I certainly do not suppose that he makes those assumptions holus-bolus, but the system by which he is, so to speak, obliged to work is vitiated by assumptions which are no longer obvious. Surface mines belie those assumptions and colloid chemistry is showing them to be no longer necessary.

Many people are doing excellent work in attempting to apply colloid chemistry to ore deposits. We had Messrs. Simon and Hatschek's paper on 'Gels', and last year Liesegang published his book on 'Geological Diffusion', in which many remarkable experiments are described. The latter author, however, does not seem to have grasped the full significance of his own work. Although he has something to say about schists and rhythmical precipitation, he still in other places makes distinctions between ore and country-rock which seem to show that he does not realize how the new conceptions apply equally well to rock and, consequently, how our whole faith in the igneo-aqueous system is shaken.

The new conceptions provided alike by geological observation and physical chemistry are still hampered in their application by the persistence of old terms, invented by men whose ideas of geology were of the grandiose architectural order. Their world was built of practically immutable 'rocks' permeated by homogeneous solutions. What Dr. Derby has humorously called 'mud-geology' is replacing it by a world consisting not of rocks *and* solutions, but of a solution-like medium. I am not able to discuss 'rhythms' or 'growths' or 'nuclear effects', but the range of what is usually called segregation in geology seems to be widening.

In such conceptions the hard, visible rock substance is not ever present. In the zone of weathering where the conditions can be fairly well ascertained we find veins, bands, contact deposits, layers, nodules, etc., all of which are local phases of extensive change of the rock substance. To those accustomed to these phenomena the permanency and immutability of rock are not so insistent in their thoughts, there is no need to suppose fissures, the distinction between rock and ore is lost, there is no need to suppose a source of energy from below to account for the changes in form and substance.

Why, therefore, immediately on passing below the water level change the whole point of view? Of course there are fissures and hot springs, but why go right away down below or to the nebular hypothesis to account for the energy? If, as many already hold, the problem of the igneous rocks and ore deposits is one problem and not two, the connecting link being hot solutions, why go outside the sediments in every case for the energy?

If silicate rocks can segregate from sediments, and if volcanic activity is an accident in this process, the classification of all such silicate rocks as 'igneous' is a classification by accident. It is here that the facts as to the persistence of ore become suggestive. They suggest that since the ore deposits peter out in depth and in length below the water level just as those above do, the whole array of rocks known to us must be looked upon more than ever as the outermost skin of the earth, and that it is a long way to down below.

Mr. George E. Collins: I think Mr. Rickard does not sufficiently distinguish between the geological evidence as to persistence of *ores* in depth, and the economic evidence as to persistence of *mines* in depth. The question is not one of the continuity of individual ore-shoots, but of the continuity of the conditions which render the existence of ore-shoots probable.

As each commercial orebody has certainly some limit in every direction, and as mines are usually started only on known orebodies, it is obvious that all mines must, sooner or later, become bottomed. That experience has confirmed this, and has further shown that the average grade of ore produced by a mine tends to become lower, does not settle the question. It is for Mr. Rickard to show that, at a given depth, there may not exist an equal number of orebodies which, owing to the lack of a natural section exposed as at surface, are never found. This particular phase of the question was outlined so clearly by H. F. Collins in the discussion of a paper by E. Halse (*Trans.* iii., p. 412) that his statement is very much to the purpose today:

"He could not believe in any continuous or general impoverishment of fissure veins in depth, either in Mexico or anywhere else, any more than he could believe in any continuous enrichment of veins in depth. Considering the accidental position with respect to the mineral veins of the present surface, produced by sub-aerial denudation, it seemed to him that (apart from surface enrichment) they were just as likely to find one vein rich at surface get poorer in depth as they were to find another poor at surface get rich in depth. Of course, those poor at surface would not be so likely to be worked, and therefore it might happen that in the majority of cases veins rich at surface would appear to become poorer at depth; it was, however, merely because the ones evidently poor at surface did not get worked so much."

If Mr. Rickard merely means to prove that individual mines eventually become worked out, or that individual orebodies eventually end, few will deny it. But it was hardly worth while to expend so much time and space on the proof.

In this entire discussion he seems to assume that all ore deposits should be considered together, regardless of differences in nature or origin. I think we may with advantage confine discussion to gold mines, from which most of his illustrations are drawn. The conditions in the case of the more soluble ores, such as those of copper and silver, are so different that, in my opinion, no conclusions drawn from them can be safely applied to gold ores. Confining myself for the present to the latter—I strongly doubt whether secondary enrichment has played any large part in the formation of bonanzas in veins. I see no evidence for the idea which Mr. Rickard has so often expressed—that bonanzas in gold veins are the result of repeated reworking of their original constituents. I refer to what he says on p. 36: 'The evidence, especially that gathered during recent years in connection with secondary enrichment, all goes to show that the orebodies found by man to-day represent the temporary balance between solution and precipitation, between dissemination and concentration.'

To my mind, the facts suggest that the vast majority of mineral deposits in veins owe their origin to relatively strong mineralizing solutions, acting through relatively short periods of time; and that the vast majority of (gold) bonanzas are primary. The effect of long-continued chemical processes on orebodies is more likely to be in the direction of dispersion than of enrichment. This may parallel Mr. Rickard's notion that 'most orebodies . . . . are

geologically young.' If not because of dispersion, what has become of the orebodies formed in remote ages? Are we to suppose that the processes which formed so many orebodies in the Tertiary were inactive earlier? There does not seem to be any essential relation between geological age and permanence in depth of orebodies; excepting that, conceivably, which arises from secondary alteration, in which case it may be that only those deposits (such as gold ores) which are least affected by reworking agencies have survived the change of long geological time.

The changes which occur in the oxidized zone form a separate branch of the question, which hardly enters into the present discussion.

The structural conditions cited by Mr. Rickard as influencing the localization of ore deposits do not necessarily depend on depth. Within ordinary depths they may change unfavourably or favourably.

On the other hand, the conditions of heat and pressure which theoretically govern the deposition of ores depend essentially on depth: and if our present theories are correct, they limit the ultimate depths at which ores can occur. But there is an essential distinction which should be drawn between ordinary mining depths and these theoretical ultimate depths. There is certainly a considerable vertical range within which these conditions affecting ore deposition are practically uniform; and I doubt whether this range is exceeded by the depths ordinarily attainable by mines.

Mr. Rickard has heretofore done good service in combating a popular generalization (as to veins improving with depth) which was not in accordance with fact. I feel, however, that his tendency is to 'stand up so straight that he leans backwards'; and to found, on admitted facts as to the limitations of individual orebodies, a theory of universal impoverishment in depth which, carried to extremes, may be almost as harmful and almost as far from the truth.

From I.M.M., Bulletin No. 125, Feb. 11, 1915.

Mr. A. G. Charleton: In a contribution, presented to the American Institute of Mining Engineers in 1907, by Geo. J. Bancroft, entitled: 'The Formation and Enrichment of Ore Bearing Veins,' he advanced the following propositions:

1. That the majority of mineralized veins are the product of expiring vulcanism.

2. That most of these veins are primarily mineralized by comparatively rich solutions in comparatively short periods of time.

3. That the solutions gained their metal-values from a comparatively rich source.

4. That there is a 'barysphere' containing large amounts of the useful metals.

5. That eruptions spring from various depths and bring various kind of magma towards the surface.

6. That only those eruptions which disturb the 'barysphere,' and bring a magma rich in metals sufficiently near to the surface to be leached by vein-making solutions, are productive of valuable ore deposits, other eruptions producing barren veins.

Ore deposits due to magmatic segregation were not included in this general survey.

And in a supplementary paper, with the same title, published in the same *Transactions*, September, 1909, Mr. Bancroft remarked, that as a result of considerable further study, he had modified his views in some respects, whilst in others he felt more than ever sure of the ground he had taken previously; frankly admitting, however, that his fifth and sixth propositions have not been as well established as the others; and he explains the grounds for his belief, into which I cannot attempt to go in detail. Those who wish to do so can refer to the original.

Two admirable papers, one by Professor J. Kemp, entitled 'Influence of Depth on the Character of Metalliferous Deposits'; and the other by Dr. Malcolm Maclaren, entitled: 'Persistence of Ore in Depth,' both read in August, 1913, at the International Geological Congress in Canada, which I had the honour of attending, deal, I think, very fairly and clearly with the obscure subject upon which Mr. Rickard has focussed our attention; and he has brought to light in his very instructive paper a number of extremely important points that certainly demand consideration.

But, speaking in a general way, it appears to me that the geologists to whom I have referred, who one must bear in mind are all very keen observers, and two of whom certainly possess international reputations; go just about as far as it is safe to generalize, with our present knowledge; and in summarizing their papers, they state the case broadly, in my opinion, very fairly. Dr. Kemp[x] expressed his conclusions as follows:

[x] Compte Rendu de la XIIe Session, Congres Geologique International, 1913.

"As regards the continuation of ore in depth, we can only say:

1. While there seems to be nothing to prevent precipitation at greater depths than we have yet reached, yet conditions seem to be specially favourable in those portions which lie between the surface and 2,000 ft. to 4,000 ft. in depth.

2. Secondary enrichment has increased the yield of those portions of many veins which are above 1,000 ft. in depth, the vertical extent of the action being limited to a relatively short stretch below the 'groundwater' level."

These ideas, it may be observed, agree with views that I advocated in an article written in July, 1913; my own conclusions being formed quite independently of those that Dr. Kemp arrived at.

Again, Dr. Maclaren[3] says:

"A study of the recent literature of ore deposits invariably forces the conclusion that workers in this branch of geology are endeavouring in their zeal for the advancement of knowledge to wrest from the scanty data available more than the simple facts warrant."

And he sums his views up thus:

"Where auriferous ore-bodies have been deposited by the influence of meteoric oxidizing waters or by cooling on approach to the Earth's surface, they may reasonably be expected to diminish in tenor with increasing depth, and finally to disappear.

"The deposits to which this generalization appear to apply are those of the Tertiary Andesitic group."

And Dr. Maclaren goes on to say:

"Even for many of these non-persistence of ore is more often a function of geological structure than of increase in depth.

"For all other deposits, and especially those of the pre-Cambrian group, ore found in strong well defined fissures may be expected to persist unchanged (apart from local horizontal variations) in 'depth', provided the rock of the lode walls is homogeneous and that the ore-bearing fissure does not pass out of that rock.

"In all these, therefore, geological structure and not depth is the factor controlling the presence of ore.

"'Depth' exercises only an indirect control, inasmuch as the greater the depth the more is the likelihood of geological change."

I cannot speak from personal examination of any greater depth than about 5,500 ft., and that in one case only, but my general knowledge, based upon the views of different authorities, and my own observations, lead me to endorse the above-named conclusions.

Consequently, if Mr. Rickard had limited his observations to those parts of the crust of the Earth about which more is actually known, without going into speculations that are practically fruitless, at the moment, regarding unknown deeper regions—about which nothing positive can be said one way or other—I should have felt nothing but appreciation for the paper he has presented to us; and I should have confined myself to criticism of some minor matters of detail, which are of secondary importance, and, comparatively speaking, unimportant.

But Mr. Rickard's contentions carry us a very considerable step, it seems to me, beyond the position that both Dr. Kemp and Dr. Maclaren at any rate felt justified in taking up in regard to this important and big question in August, 1913, and I am not personally convinced, upon the evidence since presented by the author, carefully as it has been prepared and worked out, that there is anything at present to warrant a further speculative advance into the unknown 'barysphere,' or lower 'lithosphere'.

Rightly or wrongly, however, I hold that there is only one method by which an abstract theory of this kind can be satisfactorily put to the proof and settled, viz., by *extensive* exploration, at greater depths than we have yet reached. It is possible, of course, that the examination of what are known to have once been deep-seated rocks may yield further valuable evidence, in support or otherwise, of Mr. Rickard's theory; where such rocks and rock-masses have been elevated by geological disturbances, subsequent to the formation of lodes they may contain; in places where such rock-formations have been partially exposed at surface, through the planing-down of later strata, once superimposed upon them, subsequent to the formation of the lodes that they may carry; in localities where mining is or may be hereafter more or less extensively pursued.

And this line of investigation, if followed systematically and thoroughly, is doubtless calculated to throw fresh light, in time to come, upon the nature and occurrence of ore deposits at greater depth than mines, in similar rock-formations, have perhaps

[3] Compte-Rendue de la XIIe Session, Congrès Geologique International, 1913.

reached elsewhere. Professor J. H. Vogts in his 'paper' entitled 'Problems in the Geology of Ore-deposits,' *Trans. Amer. Inst.*, Feb., 1901, p. 657, said:

"We may . . . . recall the fact that ore veins often continue to a great depth. As will be shown later on, mining is carried on in many places at a depth of not merely 0·75 to 1·25 kilom., but, in fact, as referred to the *original* surface, 3, 4, 5, perhaps 6 kilom."

Accepting this statement as a fact, how does the author harmonize it with his general theory of the poverty in metal contents and of the non-permeability of the deep-seated rocks?

Is it certain after all that the 'Old Men,' as the author appears to believe, were entirely unwarranted in *honestly* thinking that lodes 'held in depth,' and even in assuming that they were likely to go richer as they were followed down? An idea that might have appeared more ridiculous than it was, before we had gained the actual knowledge we possess to-day.

We know, of course, unfortunately, that the reverse is frequently the case in practice; what fortunes might not mining-men make if it were otherwise! But the original grounds for the belief above mentioned strike me as being tolerably clear and strictly logical from a purely scientific standpoint.

Science tells us that the mean density of the earth may be put at between 5 and 6; whilst that of the rocks of its crust is about 2·5.

And the mean density of the crust, to the mean density of the whole earth, may be calculated at between 5 and 11. I believe. At 250 miles in depth its density has been figured at 3·1; 2,000 miles (half way down) at 7·5, and at 4,000 miles (the centre) 9·8.*

I am not making myself responsible, of course, for the absolute correctness of these figures, but I think they are generally accepted by modern scientists as being near the truth.

Moreover, it does not appear that pressure alone can account for such increased densities in depth of the world's old crust; consequently the existence of a more or less highly metalliferous core doubtless grew into an honest and naturally popular belief amongst miners, and some very good engineers, too; nor have I ever seen any practical or scientific evidence yet presented, proving that the composition of the core of the earth is essentially non-metallic.

At any rate scientific and modern geologists, who have accepted and sustained the opposite theory, have accounted for the high mean density of the earth by supposing that the interior contains a much larger percentage of heavy metals than the crust.

It may be chiefly iron, or consist of various metals. Who can tell us? But let us assume, at any rate for the moment, that the foregoing general proposition is true! Is it most likely, if so, that these metals are *evenly* distributed throughout the 'barysphere', or that they are concentrated in workable zones in parts of it?

Then, if you accept the views of such modern geological authorities as the late Professor Stelzner, the late Professor Franz Posepny, C. R. Van Hise, G. W. Bancroft, the late Dr. Emmons, and even Mr. Rickard himself, may there not be many yet undiscovered 'bonanzas 'awaiting development, at much greater depths than these to which we have yet penetrated, *i.e.*, below 5,000 ft. from the present surface? May this region not contain yet many pleasant as well as unpleasant surprises for us all?

Let me call attention to some observations by C. R. Van Hise, in discussing 'Some principles controlling the Deposition of Ores,' which seem to me to lend strong rational scientific support to this view of the case.

Van Hise observes* in that magnificent collection of 'papers', 'The Genesis of Ore Deposits,' published by the American Institute of Mining Engineers in 1902:

"Two fundamental points which Lindgren, Vogt and Beck emphasize are, that the main source of the metallic ores is the igneous rocks, and that the heat of the igneous rocks has been instrumental in their production.

"The original source of much of the material for the metalliferous deposits may indeed be largely the 'centro-sphere' or the lower part of the lithosphere; for from these sources vast masses of volcanic rocks are injected into the zone of fracture or brought to the surface.

"While I hold that the main source of the water depositing ores is meteoric, I recognise that another source of such water is the igneous rocks.

* Prof. J. Le Conte places it much higher. viz., 11.

* "Some Principles Controlling the Deposition of Ores," p. 764.

"I say also, through the agency of vulcanism, water occluded in magma is transferred from the zone of rock-flowage, or even possibly from *the centrosphere* (the italics are mine) to the zone of rock-fracture."

Notwithstanding the arguments and examples so admirably presented by Mr. Rickard, I cannot see any obvious reason myself why in certain cases, *occurring sufficiently frequently to be of great economic importance*, there should not be *extensive* barren-zones and workable orebodies recurring below 5,000 ft., just as they have alternated nearer the surface.

At any rate, I think, this view can be equally as well upheld, geologically, upon the *scientific* grounds I have mentioned, as the opposite proposition, which the author advances with, I fear, undue assurance.

My own personal experience it is true is, that individual ore-shoots and orebodies near the surface have shown an unfortunate tendency in many instances to grow shorter in length or cut-out in depth, new ones sometimes coming in to take their place at lower levels; but I do not yet see any sufficient *conclusive* evidence to satisfy me that lodes and mineral deposits almost invariably tend to grow substantially poorer in depth.

Mr. J. B. Tyrrell: It is unfortunate that the class of orebody to be discussed has been left uncertain. It would hardly be worth while for anyone to write a paper with the object of proving that orebodies formed by secondary enrichment do not persist in depth, for every miner at all conversant with his subject knows that they do not. On the contrary, there are few who would have the hardihood to assert that magmatic segregations, such as the nickel-copper orebodies at Sudbury, do not continue to great depths, for the acknowledged method of their formation would indicate persistence downwards with the margin of the norite laccolith with which they are associated, and no proof has yet been offered that would indicate a contrary set of conditions.

The title of the paper does not state whether the author intends to use the surface of the earth as the starting point from which to measure the depth to which ore persists, but the whole tone of his paper would indicate that this is his intention. Now, the present surface is an accidental or fortuitous section through the rocks which compose the earth's crust, and on account of either sedimentation or erosion, which is constantly in progress, is a different plane from the surface at the time when any orebody was formed. The difference may be slight or it may be great, but it is different. If the surface at the time had been the controlling factor in the deposition of the ore, or had been the *datum plane* from which the ore was deposited, that *datum plane* is now either above or below the position which it occupied at the time of ore deposition. In the case of most secondary enrichments, where the surface has exercised a controlling influence in their deposition, and where that surface has since been subject to erosion, the controlling *datum plane* was above the present surface. In the case of many primary deposits formed in fissure veins by ascending solutions, the cause controlling deposition was at an unknown but often great distance beneath the surface, and this controlling cause or *datum plane* may have had no relationship whatever to the surface as it then existed, and may have been far beneath it.

In some cases the *datum plane*, while beneath the surface at the time of the formation of the deposit, is now partly beneath and partly above the present surface.

An excellent example of this condition may be seen at Cobalt, in Northern Ontario, where a sill of coarse diabase, about 500 ft. thick, was intruded into pre-Cambrian rocks of Keewatin and Huronian age. Both the lower and upper surfaces of this sill were planes from which the silver-bearing veins extended more or less vertically at right angles to it, so that if the sill with the dependent veins could be lifted up these veins would be seen hanging as series of fringes from its lower surface.

Erosive agencies have removed the rock which formerly covered this sill over large areas, have bevelled off the sill itself to an edge, and have removed it completely with the over-lying rock over other large areas. Most of the veins which have been worked up to the present time are in the area from which the sill was completely removed, though they are sufficiently near its edge to indicate that they must have been close to its lower surface before its removal. Veins so situated were easily discovered, but the discovery of veins which are still beneath the sill, and have not been exposed by the removal of the sill through processes of erosion, is a much more difficult matter, because it necessitates the driving of expensive tunnels and cross-cuts through the hard underlying rock. However, some

such veins have been discovered. The first one that came under my observation was in a mine just east of the Nipissing property. It terminated upwards at the lower surface of the diabase sill, and extended thence downwards into the underlying rock.

In the south-eastern portion of the Cobalt area the surface has not been eroded down to the diabase sill, and there the ore was found to extend upwards from the diabase sill towards the surface, but not to the surface. In this district very rich ore has been found at a depth of 650 ft. below the present surface, which is considerably deeper than the 400 ft. recorded by Mr. Rickard.

Considered generally, the mode of occurrence of silver ore at Cobalt is typical of rich veins which have been formed beneath the surface, but, as far as is known, quite independent of surface agencies. The *datum planes* to which the ore must be referred are the upper and lower surfaces of the diabase sill, and not the present surface, and as the ore does not persist for more than a few hundred feet from these *datum planes*, where these *planes* are close to the surface, the ore-bearing veins naturally do not extend to great depths. Whether similar veins would be found at great depths, if the upper or lower surface of the sill were followed where it dips under the overlying rock, is not known.

The author's method of sweeping aside all evidence and simply asserting his belief that most ore deposits are of recent origin will hardly carry conviction to thoughtful mining engineers. Instances in which the gold, or other valuable material for which a mine may be worked, has been determined to have been deposited at but a slightly later date than the gangue of the vein in which it occurs must be known to everyone, and it is not necessary to enumerate details of evidence here. However, I might mention one of many instances. In the pre-Cambrian rocks of the Porcupine district of Northern Ontario, gold-bearing quartz veins occur in basaltic greenstones and sericitic schists. The most recent rock in this pre-Cambrian complex is a basic syenite which, in the form of a dyke, cuts across the schist with its contained gold-bearing veins, and is evidently later than the introduction of gold into the vein, showing that enrichment was confined entirely to pre-Cambrian times previous to the intrusion of this dyke.

From I.M.M., Bulletin No. 126, March 11, 1915

Mr. W. H. Goodchild: The theoretical section of the paper seems to me to be open to serious criticism. Describing ore as 'metal-yielding rock that can be exploited to advantage,' the general impression conveyed seems to be that such rock is limited in its occurrence to an outer shell of, say, 6,000 ft. in thickness. Mr. Rickard points out that 'as mining is based on the application of scientific principles, it is the aim of the intelligent miner to eliminate error, more especially the veiled falsity that masquerades in the guise of an attractive generalization.' Later he states that 'deposition of ore is the result of precipitation from solution,' an unmistakable generalization which, however attractive it may seem to inventors of theories of orogenesis, the intelligent and practical miner would, I think, do well to regard, in the present state of oro-genetic science, as an unfortunate example of 'veiled falsity'.

Mr. Rickard concisely summarizes the state of contemporary opinion in regard to the origin of some selected types of such ore deposits. The view that deposits belonging to these types when encountered again will probably have a similarly restricted vertical extension and diminishing richness with increasing depth, is without doubt an economically sound one for the mining engineer and shareholder to adopt, until additional evidence is forthcoming to require its modification.

To the serious student of ore deposits, however, descriptions and theoretical discussions purporting to tell us how nature has furnished the top story can carry no firm conviction that the lower stores are 'To Let, unfurnished'. I have it in mind, for instance, that matter sinks in the cupola and that in a well-conducted smelter one would have to treat a vast amount of slag with watery washes if one sought to recover but a tithe of the amount of wealth contained in the bottoms. The human metallurgist extracts or concentrates quite a limited proportion of his metals by solution and precipitation processes. What evidence have we, indeed, that the eternal metallurgist, nature, has depended merely on solution and precipitation phenomena for the concentration of ores when she has a plethora of other means at her disposal for concentrating the minerals we prize as sources of our metals?

In this connection I think the miner may well be encouraged by the vague indefinite-

ness of oro-genetic theories in general. The number of missing links in the chain of evidence, particularly in physico-chemical detail, is an outstanding feature of the present-day literature on ore deposits, to which Mr. Rickard's paper is no exception. Until many more of these gaps are filled in by the advance of the physical sciences it seems to me that the miner would be well advised to attach far more weight to detailed description of ore occurrence than to theories based on necessarily scanty and frequently quite inconclusive evidence.

Mr. Rickard refers to the work of F. D. Adams on the crushing strength of rocks, but the figures given by Mr. Rickard, and the conclusions he draws in regard to the depth at which the only openings in rock are of subcapillary size, are apparently so very wide of the mark that I prefer to quote Adams himself in this connection:

(1) The calculations which have been made as to the depths below the earth's surface at which all cavities in the earth's crust would be closed by plastic flow, based on the crushing strength of rocks at the earth's surface, are erroneous.

(2) At ordinary temperatures, but under the conditions of hydrostatic pressure or cubic compression which exist within the earth's crust, granite will sustain a load of nearly 100 tons to the square inch, that is to say, a load rather more than seven times as great as that which will crush it at the surface of the earth under the conditions of the usual laboratory test.

(3) Under the conditions of temperature and pressure which are believed to obtain within the earth's crust, empty cavities may exist in granite to a depth of at least 11 miles. These may extend to still greater depths, and, if filled with water, gas, or vapour, will certainly do so, owing to the pressure exerted by such fluids or gases upon the inner surfaces of such cavities or fissures.

Presumably Mr. Rickard has inadvertently taken the earlier figures referred to by Adams* as erroneous.

Mr. B. I. Collings: We are compelled to recognize the fact that in the vast majority of cases mines lose their values with depth, although the accepted theories to account for this phenomenon are in some respects vague, incomplete, and unsatisfying. There are, with Rhodesian mines, a number of accepted facts, which are not easy to reconcile with current theories of ore deposition and its lack of persistence in depth. The genesis of the gold reefs in Rhodesia, is, without exception, closely connected with acid igneous rocks.

The orebodies in nearly all cases skirt the margins of the granite masses, which outcrop over large areas, and are invariably associated with felsite, quartz, porphyry and allied intrusive rocks.

These dikes are generally regarded as apophyses of the main granite masses, and the period of injection of the gold and accompanying minerals, with resultant formation of the orebodies, undoubtedly synchronized with the later stages of the cooling of the granite.

All these conclusions are in general accord with present-day theories of the formation of the type of mineral deposit all the world over. The granite, with which these deposits are genetically connected, belongs unquestionably to a very early geological period, and it therefore follows that the orebodies themselves are of great age. Since their formation, denudation must have removed many thousands of feet of overlying rock, and the present outcrops must have been a corresponding distance below the surface.

I maintain that in most instances mines give out in depth, for the same reasons that they give out on the strike, and as a proof of this I suggest that the longitudinal section of any worked-out mine be taken and tilted through an angle of 90°, so that its line of strike becomes its line of dip. Then, after eliminating the effect of any surface concentrating influence, it will be found that the conditions of a normal mine still obtain.

In other words, the behaviour of an orebody on the limit of its strike will be found similar to its behaviour on the limit of its depth. It is probable that most metalliferous veins have been filled from below, but the channel whence the filling has come may well have been very small.

On this theory, the Rand orebodies, with their extraordinarily long strike, might be expected to extend also for a very great distance on the dip.

This is found to be the case, their very gradual impoverishment in depth being attributable entirely to the factors influenced by increasing depth.

* F. D. Adams, *Journ. of Geology*, Vol. 20, 1912, pp. 97-118, and Duly, "Igneous Rocks and their Origin," pp. 180-181.

THE THEORY OF TUBE MILLING.

Printed in February Journal, 1915.)

By Mr. H. A. White, (Member of Council).

DISCUSSION.

Mr. W. R. Dowling *(Past-President):* In discussing the problem of efficient tube-milling, the value of safety cone classifiers after the usual tube-mill cone classifiers for the best classification, crushing and amalgamation results, is possibly not fully recognized on the Rand. The first published reference I know of recommending the installation of a sand return cone after the tube-mill classifiers is by Walter Neal, one of our corresponding members of Council, in an article published in the *Mining and Scientific Press* on 2nd April, 1910. In June, 1910, J. E. Thomas recommended the installation of return sand cones to prevent coarse sand overflowing the tube mill classifiers, reaching the cyanide works (*Journal*, C.M. and M. Society, Vol. X., p. 153, June, 1910). Also in the *Journal* of this Society, in a paper "The Amalgamation of Gold in Banket Ore" by the writer, an illustration is given showing a safety or return cone (Vol. XI., p. 121, March, 1911). These safety or return cones are preferably built of the same size as the usual tube mill classifying cones, and should be about one-third in number of the battery of tube mill classifying cones. As the number of safety cones is smaller than the tube-mill classifying cones and a much greater volume of pulp overflows each safety cone, the peripheral launders of the former should be larger and the cone operated with a considerable depth of fluid pulp. The safety cones should be grouped at some point after the several streams overflowing the tube-mill classifying cones have been brought together in a common launder near either the amalgamating plates or the tube mill return pumps so that the underflow may be either led down a steep grade to plates or flowed to the pump pit. By this means any coarse particles overflowing the tube mill cones through irregular or excessive pulp inflow are returned into the tube mill circuit.

When fitted with diaphragms and automatic regulators, the thick underflow of the safety cones may be reduced to the minimum volume, thus saving elevating costs. Where the necessary fall of 30 ft. for the cyclone underflow launder is not available, a supply of water turned on and off by an attachment to the regulator may be resorted to, as has been done by J. M. Dixon, the mill manager of the Robinson Deep. This necessary extra water only flows when the cone is discharging an underflow of thick sand pulp.

The system of safety cones as sketched above is preferable to the individual safety cone for each tube-mill classifier, as being more efficient and as there are fewer units to require attention, less volume of pulp is elevated by the tube-mill pumps, and there is less danger of chokes with the large underflow apertures of large cones than in the necessarily smaller underflow aperture of smaller cones with fluid underflow.

A more important advantage of a reliable safety cone installation is that the necessity for even distribution of initial pulp to tube-mill classifiers is not so great. Even and equal distribution of pulp to classifiers has always been difficult and costly, and the most successful of the many devices tried involve greater elevation of pulp and high capital and maintenance charges. With an ordinary distribution of pulp as obtained from a reasonably well arranged set of launders there is no danger, given an adequate crushing plant, of coarse sand reaching the cyanide works when safety cones such as described are installed.

In some of the older plants it has been impossible to install the ordinary safety cones owing to lack of fall both in overflow and underflow launders. In these cases a rectangular box 9 ft. long by 3 ft. wide by 5 ft. deep has been installed lengthwise across the total pulp stream overflowing the tube-mill classifier cones, and is run filled with sand. The box is provided with three openings, one in the centre of each square yard of the bottom, permitting nozzles up to 2½ in. internal diameter being used, but the actual working size is usually ¾ in. to 1 in. Over each opening inside the box is a 12 in. diaphragm is supported 12 in. from the bottom on three legs which are screwed at the top to allow of adjustment of the diaphragm. The top of the box is covered with 1 in. mesh screen set just below the pulp surface level, and this prevents the incoming stream disturbing the settled sand. A continuous thick underflow of pulp is obtained, which gravitates into an adjacent tube-mill at the Simmer and Jack, and very little attention is required. From 10 to 60 tons of sand flow from each opening with a mois-

ture content of 27 and a grading of 21·7 +60; 11·3 −60 · 90; 37·0 −90. Though not so efficient as regular safety cones owing to limited overflow breadth of only 9 ft., compared with over double this figure for the periphery of one 6 ft. diameter cone, this safety diaphragm classifier is cheap to instal and of considerable utility, as has been found at the Simmer & Jack and Simmer Deep plants.

The President: It is very disappointing there is so little discussion on the papers by Mr. White and Mr. Nettleton, which are before you for final discussion to-night.

I might point out that next month will be the last opportunity for dealing with Mr. Dowling's and Mr. Cullen's papers. So far we have only had experiences of one particular group with regard to tube-milling. There are many metallurgists on the mines of other groups who should be able to come forward with experiences and discussion on this paper; and I hope they will do so. As there is no further business before the meeting, we will adjourn.

The meeting then terminated.

The following paper will be submitted for discussion at the meeting of the Society to be held on Saturday, the 15th May, 1915.

NOTES ON THE PRACTICAL TESTING OF WORKING CYANIDE SOLUTIONS

By Edward H. Croghan, B.A. (*Member*).

The purpose of these notes is to illustrate, firstly, the great diversity of results that may be obtained in testing working cyanide solutions, and secondly, to promote a discussion as to the most suitable uniform procedure which should be adopted on this field. Before doing so, it would be advisable to define certain terms, practically as stated by Clennell.

Free cyanide is the equivalent, in terms of potassium cyanide, of all the cyanogen which is present in the solution as simple cyanides of the alkalis and alkaline earth metals, such as KCy, NaCy, NH_4Cy, $CaCy_2$ and $BaCy_2$. It does not include cyanogen present in the form of hydrocyanic acid or double cyanides.

Total cyanide is the equivalent, in terms of potassium cyanide, of all the cyanogen which is present in the solution as simple cyanides of the alkali and alkaline earth metals, hydrocyanic acid, and zinc or copper double cyanide. Such cyanide is rendered "free" by the addition of caustic alkali, whereas the cyanogen which exists in the form of ferrocyanide and sulphocyanide is not set "free" by caustic alkali.

Protective alkali may be defined as the equivalent to the alkaline hydrates plus half the mono-carbonates. The term is based on the assumption that the presence of free alkali in an ordinary working solution can be wholly or partially neutralized, on addition of a dilute mineral acid, or carbonic acid, before any decomposition of cyanide occurs. As a matter of fact, as it is well known, a cyanide solution undergoes gradual decomposition, with evolution of hydrocyanic acid, even when an excess of caustic alkali is present, so that the protection afforded is only partial and temporary, and not absolute.

As you know, on working solutions, and by that is chiefly meant, with zinc precipitation in vogue, solutions containing both free cyanide and cyanide existing as double zinc cyanide, there are in titrating with silver nitrate in the presence of potassium iodide as indicator, two "end" points that can be observed: (1) the first appearance of a distinct opalescence or permanent flocculent whitish precipitate; (2) following on (1) the appearance of a distinct permanent yellowish tint.

The first point is presumed to approximately give the figures for free cyanide content, and the second depending chiefly upon whether there is present or not sufficient free alkali and the dilution, the whole or a part of the total cyanide. Subsequent references to the above two finishing points will be briefly termed the white and yellow points, respectively.

Now, there does not seem to be uniformity in the matter of testing. It is in consequence interesting to note that supposing a test were made on, say, a strong solution to the yellow point and 0·42 KCN found the

more correct result for *free* KCN, if the titration had been conducted to the white point, might only be 0·06 .

On slime solutions the same sort of values could occur, a 0·015 KCN according to the yellow point might not be on the white more than 0·008 free KCN, and possibly less.

In the course of testing some working solutions for free cyanide, I took the opportunity of gathering further comparative data by noting the percentage of so-called free cyanide according to the white or yellow points previously referred to, and continuing the determination for protective alkali on the same solution which had been titrated by silver nitrate with potassium iodide as indicator.

As another interesting test, I adopted a method which I have designated "special", as given in Chennell's *The Chemistry of Cyanide Solutions* (1910 Edition, p. 25, b). The end point stated for this method is the white one, but I also estimated the results by continuing titrations up to the yellow point. Further, the total cyanide was ascertained by the usual standard method. The description of the analytical methods is given as an appendix, and under Table I. the results of the different procedures are entered.

It will be noted that the protective alkali is reported in terms of lime (CaO), which is open to question, especially on solutions after precipitation, but as it seems to be customary, I have for these notes not departed from the practice. It is a well known fact that the alkalinity of solutions increases whilst passing through a zinc extractor box. This increase is obviously not due to the lime factor, but for working data and in order to get comparative results I can see no serious objection to the alkalinity being reported in terms of lime. After all, it is on a par with the fact that we still report the cyanide in terms of potassium cyanide when it is understood that the salt being used is sodium cyanide.

From a general survey of the preceding tabulated figures, the following is obvious:

KCN.

(1) The white point results of both methods are much lower than those of their respective yellow.

(2) The yellow point data of the ordinary method are a good deal higher than those of the yellow under the special process; in fact, the former almost correspond to the total cyanide found, particularly on the weak solutions.

3) The white point percentages of the ordinary method fall interestingly between those of the white and yellow of the special procedure, with two minor exceptions (Nos. 16 and 19). As shewing this, I have under Table II. entered the average of the two results given by the white and yellow end points of the special method (where 0·0005 occurs as a fourth decimal, it is taken as 0·001) and the individual percentages of the white point of the ordinary procedure.

The average of the differences stated under Table II. is about ±0·002. (See p. 273.)

From this small number of analyses it is indicated that neither the white nor the yellow point data of the special method can lay claim to being in closer agreement with those of the white point of the ordinary procedure. I do not wish to convey the impression that I am trying to establish any case under the above, this is far from my purpose. I have simply tabulated what seemed to me an interesting comparison, which subsequent work might prove to be only a curiously close coincidence of results.

PROTECTIVE ALKALI.

(1) Comparing the protective alkali results obtained by the ordinary method on solutions which had been titrated for free cyanide, *i.e.*, to the white point, with those of the special method, on the weak cyanide solutions the agreement on the whole is quite close, but on the stronger the former tends to be higher.

(2) The protective alkali figures found under the ordinary method on the solutions titrated to the yellow point with silver nitrate, are on the average distinctly lower than those secured on the white point of the ordinary, and seemingly, though to a lesser extent, lower compared with the data yielded by the special method.

The estimation of protective alkali on a yellow point solution is very unsatisfactory owing to the disappearance and slow faint reappearance of the purple-red of the phenolphthalein indicator, in fact any result might be obtained on a quick titration and not looking out for this recurrence. My titrations were slowly carried out until after about a quarter of a minute no trace of colour developed. If in the titration with silver nitrate in the first instance the yellow point be overdone, and it is fairly easy to do so in a bad light, the protective alkali figures will be found to be much lower.

Table accompanying "**Notes on the Practical Testing of Working Cyanide Solutions,**" by E. H. Croghan, B.A. (Member).

TABLE I.—Illustrating chiefly the various results obtainable according to the end point taken in testing working cyanide solutions.

Sample No.	1		2		3		4		5		6		7		8		9		10		11		12		13		14		15		16		17		18		19		20		21		22	
	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO	KCN	CaO
Ordinary Method. *End point taken to distinct permanent*																																												
(*a*) Flocculent whitish precipitate	0·175	0·011	0·155	0·017	0·144	0·015	0·125	0·019	0·050	0·007	0·045	0·012	0·040	0·009	0·020	0·008	0·015	0·006	0·015	0·009	0·015	0·007	0·015	0·009	0·011	0·009	0·011	0·006	0·011	0·003	0·009	0·004	0·008	0·003	0·008	0·004	0·008	0·005	0·007	0·005	0·007	0·001	0·006	0·004
(*b*) Yellowish tinted precipitate	0·225	0·012	0·202	0·012	0·190	0·010	0·190	0·011	0·059	0·004	0·083	0·006	0·073	0·001	0·048	0·003	0·037	0·002	0·042	0·006	0·030	0·004	0·035	0·007	0·041	0·006	0·028	0·003	0·021	0·002	0·042	0·004	0·017	0·003	0·011	0·003	0·010	0·005	0·012	0·004	0·011	0·002	0·008	0·003
Special Method. *End point taken to distinct permanent*	Per cent. of KCN																						Per cent. of KCN																					
(*a*) Flocculent whitish precipitate	0·156		0·145		0·133		0·115		0·035		0·031		0·030		0·012		0·009		0·010		0·006		0·006		0·007		0·006		0·006		0·003		0·001		0·003		0·003		0·006		0·004		0·003	
(*b*) Yellowish tinted precipitate	0·180		0·165		0·155		0·137		0·051		0·060		0·060		0·030		0·025		0·025		0·016		0·018		0·021		0·011		0·017		0·007		0·012		0·008		0·006		0·009		0·010		0·006	
	Per cent. of Protective Alkali (CaO)																						Per cent. of Protective Alkali (CaO)																					
With addition of silver nitrate and potassium ferrocyanide	0·013		0·012		0·012		0·017		0·004		0·008		0·006		0·006		0·005		0·008		0·007		0·009		0·008		0·006		0·002		0·004		0·004		0·004		0·006		0·005		0·004		0·004	
Standard Method	Per cent. of Total KCN																						Per cent. of Total KCN																					
Total cyanide—free KCN + KCN of $K_2Zn(CN)_4$	0·220		0·215		0·205		0·198		0·088		0·097		0·100		0·056		0·046		0·045		0·031		0·035		0·043		0·029		0·036		0·012		0·018		0·011		0·010		0·012		0·015		0·012	

TABLE II.

Comparison of free cyanide white point results of ordinary method, and the average of the two results under the white and yellow points of the special method.

Sample No.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Ordinary Method Distinct permanent flocculent whitish precipitate	0·175	0·155	0·141	0·125	0·050	0·045	0·040	0·020	0·005	0·005	0·005	0·005	0·004	0·004	0·011	0·009	0·008	0·008	0·008	0·007	0·007	0·006
Special Method Average results of white and yellow points ...	0·168	0·155	0·144	0·126	0·045	0·046	0·045	0·021	0·007	0·008	0·011	0·012	0·014	0·011	0·012	0·005	0·008	0·006	0·005	0·008	0·007	0·005
Difference ...	0·007	Nil	Nil	0·001	0·005	0·001	0·005	0·001	0·002	0·003	0·004	0·003	Nil	0·003	0·001	0·004	Nil	0·002	0·003	0·001	Nil	0·001

One of the theories advanced is that the lower alkalinity estimated may be due to the formation of zinc nitrate, which in turn would react with the alkali, and so lessen its amount (*vide* Clennell, *The Chemistry of Cyanide Solutions*, 1910, pp. 64-65). Under the special method of determining the protective alkali, this difficulty is obviated chiefly by the addition of a slight excess of potassium ferrocyanide, which regenerates the alkali.

In touching on the subject of the finishing point that might be adopted in the testing of working cyanide solutions for *free* cyanide, I should first like to quote from Clennell's *The Cyanide Handbook* ". In the 1910 edition, on page 438, under a heading " Working test for free cyanide ", for which standard silver nitrate with potassium iodide as indicator is utilised as usual, the following is stated:—

" The finishing point is usually taken as the first appearance of a permanent *yellowish* turbidity, disregarding a slight white cloudiness which may appear earlier. This probably gives slightly too high a result in presence of K_2ZnCy_4 and its analogues, as some of the cyanogen of these compounds may also be indicated when free alkali is present. This is not, however, of much consequence if the test be made in the same manner for all solutions. In absence of KI, the end point (white turbidity) is reached sooner, but is generally uncertain and indefinite. An intermediate result, and one which, so far as tested, appears to correspond with the actual working strength of the solution, is obtained by neutralizing the protective alkali and then adding KI and titrating with $AgNO_3$ in the usual way, taking as the end-point the appearance of a distinct permanent white turbidity."

The word " yellowish " in italics is as printed in the book.

Now, I wish to offer some criticism of this paragraph, because I think it is misleading on application to local working solutions, and it must be based on other conditions. Referring to the end point " usually taken as the first appearance of a permanent *yellowish* turbidity, disregarding a slight white cloudiness which may appear earlier ", I should like to state that as the result of my experience I have found on local working solutions that before the yellow point develops or can be seen there is not a slight white cloudiness, but a very strong turbidity or precipitate, or as it might be termed, an exceedingly overdone titration for free cyanide if the first distinct permanent white turbidity had been taken as the end point.

On turning to page 140 of the same book, I see that in stating the end point for the estimation of total cyanide, the same wording is practically used, viz., "permanent yellow turbidity." In titrating local solutions, using potassium iodide as indicator, I have not noticed any marked difference in the appearance of the end point, whether the titration be for free cyanide, or whether it be after the addition of alkali for total cyanide.

On looking at the figures shown under Table I. for so-called free cyanide according to the yellow point of the ordinary method, and comparing them with those of the total cyanide found, with which they almost correspond, especially on weak solutions, and from my further experience up to the present, the general statement that the permanent yellowish turbidity probably gives slightly too high a result in presence of $K_2Zn(Cy)_4$, etc., can hardly be applied to local working solutions, as there seems to be sufficient free alkali, dilution, etc., to give a result under the yellow point of almost total cyanide.

As to the point that this does not matter if the test be made in the same manner for all solutions, this may be so from a works point of view, but if we are talking of the strengths of working solutions in cyanide, and presumably *free* cyanide would be generally understood, it is apt to be misleading. This is well indicated under Table I. by a comparison of the data of either the ordinary or special method where the titration has been conducted to the white point with those of the yellow of the ordinary method.

I hope Mr. Clennell, if he should happen to read these notes, will not misinterpret the trend of my criticism, which is not uttered as a criticism of his own views, which I believe from a study of his writings are more in harmony with my own. The method referred to in the last portion of the preceding abstracted statement is the special one of these notes, and it is interesting to observe that Mr. Clennell has so far found it as agreeing with the actual working strengths of solutions.

Mr. Clennell, in referring to this method of first neutralizing the alkali before titrating for free cyanide, states in his book on *The Chemistry of Cyanide Solutions* (1910 edition, p. 25) as follows:

"In solutions containing no free alkali, the first appearance of a flocculent precipitate, in presence of potassium iodide, gives a reading which is, in general, slightly higher than the theoretical contents of the solution in free cyanide. The point is fairly definite, and although not absolutely satisfactory, appears to be the most reliable method so far described for determining this quantity in presence of zinc.

"It is possible that an actual dissociation of the K_2ZnCy_4 into KCy and $ZnCy_2$ may take place to some extent, and that the test correctly indicates the free cyanide. In some solutions the numbers correspond closely with the theoretical values."

As remarked on before, towards the end of the paragraph abstracted in the first instance from his *Cyanide Handbook*, Mr. Clennell again there states that on tests so far conducted, this method appears to correspond with the actual working strengths of the solution.

Now, I do not propose that this method should be adopted in practice, though it has certain advantages. To carry it out it means first determining the total cyanide, a figure which might be of value at times as indicating the state of our solutions in double zinc cyanide, which can be arrived at by a simple calculation.

The $K_2Zn(Cy)_4$ would be the difference between the total cyanide and the free cyanide multiplied by 1·9. Having found the total cyanide, the protective alkali is estimated on a fresh portion of the solution by the addition of slightly more than the amount of silver nitrate found to be required for the determination of the total cyanide above and a slight excess of potassium ferrocyanide. Phenolphthalein indicator is added and the turbid mixture (without filtering) titrated as usual with standard acid. This portion of the method is credited to L. M. Green (*Proceedings*, Inst. Min. and Met., October, 1901), and for protective alkali it is supposed to obviate the difficulties due to the presence of zinc. Mr. Clennell questions this.

The free cyanide is then determined on a fresh portion by neutralizing the alkali as found above and titrating with standard silver nitrate in the presence of potassium iodide to the permanent flocculent whitish precipitate end point.

Though the above method is stated to give slightly higher results than are demanded by theory, and the results under Table I. for free cyanide as ascertained by titration to the white point, under the ordinary method show yet higher

data, yet from the point of view of practical convenience and simple procedure in my opinion, if the *free* cyanide is the factor to be recorded, then all tests should be only carried out in the presence of potassium iodide as indicator to the first distinct opalescence, or, as I have so far noted, to what seems a definite end point that cannot be mistaken, *i.e.*, the distinct appearance of a flocculent whitish precipitate which after agitation remains permanent for about half-a-minute. In doubtful cases, if the solution is more or less at rest, it can be usually seen by shading, with the hand, the flask standing on a black or brown surface or simply shading against not too strong light. There probably exist exceptions, but up to the present I have found that as a final court of appeal it is very satisfactory, and it should do away a good deal with the personal equation. It tends to give slightly higher results (about 0·1 cc. more) than the first faint opalescence, which end-point, speaking for myself, I am not always sure of, and I suppose is a matter of having a very sharp vision gained by experience for detection.

For the determination of the protective alkali, it is interesting to note under Table I. the close agreement on dilute solutions of the results obtained on the solution which has been titrated to the white point as above, compared with those of the special method, though on strong solutions, the former tends to be higher. This may be partly explained on the assumption that a portion of the potassium zinc cyanide has been included as protective alkali. The late Mr. Bettel, in his researches given in Vol. I. of our *Journal*, for devising some scheme for the rapid technical analyses of cyanide working solutions, showed that with phenolphthalein as indicator, the alkalinity of K_2ZnCy_4, as tested by n 10 acid, is equal to 19·5 parts of KCy out of a possible 130·2 parts. Neglecting this factor, for practical purposes, for the estimation of protective alkali, continue as usual on the same solution which has been titrated for free cyanide by silver nitrate to the white point.

Before concluding I should like to remark that in the determination of total cyanide, it has so far been always noted that on the addition of the extra alkali required for the estimation, a whitish looking precipitate appears, which becomes quite heavy with the further addition of alkali (NaOH or KOH, not CaO, which does not seem to develop it).

I have not had time to go into any detailed examination, but the following will serve as an approximate description of a qualitative investigation:—

To 1,000 cc. of a strong working solution which had passed through the zinc boxes was added a concentrated solution of purest KOH until precipitation seemed complete, filtered, and the precipitate well washed in the cold about 12 times.

It was then washed off the filter paper into a 2,000 cc. flask and made up to the mark with distilled water.

The bulk of the precipitate immediately dissolved. The mixture was shaken off and on during a couple of days, but a fair amount of a distinct flocculent precipitate still remained. In solution was only lime, assuming potassium or sodium salts to be absent.

On collecting this final smaller amount of flocculent precipitate on a filter I was surprised to see that it was distinctly coloured blue. It was washed until no free alkali showed in the filtrate (tested with phenolphthalein).

Practically the whole of the precipitate freely dissolved in the cold in dilute hydrochloric acid with much effervescence of CO_2 (no odour of HCN was perceptible), and as bases were chiefly found lime and zinc, with a small quantity of copper. No magnesia was evident, and sodium or potassium were not looked for.

The very small amount that did not dissolve in the hydrochloric acid was flocculent, and gave as a reaction a minute trace of cyanide. Owing to the little quantity available it was not tested for bases, and is possibly zinc cyanide. In the original final precipitate no cyanide was detectable, the prussian blue test being used in each case.

Mr. G. W. Williams, in his excellent paper on "The Determination of Constants in Working Cyanide Solutions", contained in Vol. iv. of our *Journal*, states on p. 186, in a reference to the troublesome white precipitate of hydrated zinc oxide which appears in the weak boxes, "This precipitate is only formed when an excess of an alkaline hydrate is present in solution, or rather when the ratio of sodium hydrate to 'total' cyanide exceeds 1NaOH : 5KCN."

In the estimation of total cyanide this ratio is exceeded.

The blue colour may be accepted as due to the copper which is possibly present as carbonate.

I have only examined this one precipitate, and so it must not be assumed that they are all bluish or of the same character.

In conclusion, I think that the subject of standardizing the testing of working cyanide solutions, the recording of free cyanide in terms of, say CN (cyanogen) and the protective alkali, say, as OH (hydroxyl), though the latter terms, which may be more correct, are of little consequence from a practical standpoint as long as for comparative purposes certain definite terms are decided on, are matters that should be fully discussed in our Society, and it is with the idea of promoting that discussion that I have submitted these few notes.

APPENDIX.

Analytical Methods employed by the Author.

Standard silver nitrate 6·7523 grammes per litre. Per 50 cc. sample for testing 1 cc. standard silver nitrate 0·01 KCN.

ORDINARY METHOD.

For *Free Cyanide:*

To 50 cc. sample add 5 drops of 10 % solution of potassium iodide. Titrate with standard silver nitrate to first distinct flocculent whitish precipitate, which after agitation remains permanent for about half-a-minute.

For *Protective Alkali:*

To the solution used above for free cyanide, add 8 to 10 drops of 0·5 % neutral alcoholic solution of phenolphthalein and titrate with standard acid as usual until the colour has quite disappeared.

For the yellow point results proceed on similar lines to the preceding.

Total Cyanide:

To 50 cc. sample add 5 cc. to 10 cc. of 4 % solution of sodium hydrate containing 1 % potassium iodide, and titrate with standard silver nitrate to yellow tint.

SPECIAL METHOD.

(Clennell's *Chemistry of Cyanide Solutions*, 1910 edition, p. 25, n.)

Free Cyanide:

(1) Determine the total cyanide as above.

(2) To another lot of 50 cc. sample, add slightly more (about 0·2 cc.) than the amount of silver nitrate found to be required for the total cyanide, together with a slight excess of potassium ferrocyanide. Then 8 to 10 drops of phenolphthalein are added to the turbid liquid and the mixture titrated (without filtering) with standard acid (N/10 sulphuric acid) until the pink just disappears.

(3) To a third 50 cc. of the original sample add gradually with agitation the amount of acid required under (2). Five drops of 10 % potassium iodide solution are then added and the mixture titrated with silver nitrate until a distinct flocculent whitish precipitate remains permanent for about half-a-minute.

Protective Alkali:

This is the result as found under (2).

Contributions and Correspondence.

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

I would like to be permitted to discuss further Prof. Stanley's reply to my criticism on his paper on "Hydraulic Classifiers and Classification".

I did not refer to jigs because these are obviously too closely related to hydraulic classifiers to be rationally employed on classifier products.

The products of machines which divide material according to the rate at which particles fall in water should be treated on machines which operate on some *radically different principle*, and it is the general custom to subject such classified products to the rolling action of water flowing down an inclined plane.

So far, doubtless, we are agreed; but I have set this out explicitly lest Prof. Stanley's expression "overlooked jigs" might be taken to imply an inaccountable omission instead of a very natural one on my part.

I cannot agree that in maintaining that efficiently classified particles are more easily concentrated than closely sized particles, I am "in conflict with the general view", if by "general view" one is permitted to understand the general view of those who combine with thorough practical experience sufficient theoretical knowledge to express their observations clearly to others.

I only remember seeing the contrary expressed by one other such person, and he, like Prof. Stanley, had the courage of his convictions and invented a submerged sieve for dealing with finely divided material on these lines. I do not know if it has come into practical use.

The only excuse for such procedure is that in the past it has been found possible to attain fairly efficient sizing through sieves, whereas such hydraulic classification as has been carried out on a continuous

working scale has in general been most inefficient, especially where the attempt has been made to produce more than two classes.

Thoroughly efficiently screened products may be better, I admit, than *badly classified* ones, but efficient sieving of finely divided material is costly, whilst hydraulic classification efficiently carried out offers the prospect of economy, and at the same time supplies the particles of concentrate and waste in ideally related sizes for rapid and complete separation by the rolling action of water down an inclined plane.

Prof. Stanley appears to be under an unfortunate misapprehension about this action, which I maintain is the chief factor in the removal of the waste on all machines of the Frue vanner and Wilfley table type, as also on the buddle, the revolving slime frame and the "dead" frame.

He stated that (starting with two particles of the same size, but of different specific gravity, of which of course the lighter will be removed the more easily) "as the size of the particle of heavier mineral *decreases a point will be reached at which it will be moved with equal readiness.*"

I maintain, and indeed know from oft-repeated observation and experience, that this is just the reverse of what really happens, for (assuming that the particles are completely immersed as is essential) the *smaller* a particle is *the more slowly it is rolled along* by the current, and the more difficult it is therefore to remove it with the waste until one comes to a particle so small that it has no chance of settling out of the feed water, but is kept in complete suspension. Such a particle even may drift into a quiet space amongst the larger ones and settle down, in which case whether of heavy mineral or light it will be likely to reach the concentrate bin.

In the foregoing I have passed from the single layer condition to the more practical one of a bed several particles deep, which introduces some complications, such as the settling out of excessively fine particles just mentioned, which cannot be very efficiently obviated by any motion given to the table.

It is surely an error to say that stratification on a Wilfley table or vanner takes place "in accordance to a great extent with 'hindered settling laws', for that is to confuse the action taking place in the thin bed on these machines with that in the bed of a jigging sieve or amongst all but the lowest layers of a *deep* bed of wet material (*e.g.*, in the "tossing kieve" employed in Cornwall for removing the last traces of fine light waste from the concentrate before it is ready for sale) in which further settlement is taking place. In both the latter cases there is either a continuous or an intermittent upward flow of water between the particles sufficient to carry up with it much of the finest and lightest material present, but that condition is only maintained among the particles on a shaking table or a Vanner for a very limited period of time, after which the residual water in the lower layers of the bed only acts as a lubricant keeping it slightly mobile and permitting some of the finer specifically light particles if present to settle down among the heavy particles which found their way quickly to the bottom when "hindered settling" conditions were momentarily prevalent. Of course, if the feed consisted of efficiently classified particles no such fine specifically light particles would be present, and this difficulty would not arise.

Let it be clearly understood as a fact which may be demonstrated by the careful examination of the second quality concentrate from any shaking table, that the most difficult light waste to remove from heavy mineral is considerably finer, and not coarser than the latter. A little work with the vanning shovel (which is a scientific instrument, somewhat maligned of recent years by those who do not understand all its capabilities, but of the utmost value in the examination of mixed minerals) will quickly demonstrate the same thing.

My contention should now be clearly understood that *with efficiently classified particles*, the tendency to settle or otherwise has little bearing on the action of the concentrator, because all the particles act very nearly alike in that respect, and provided the angle of slope and the volume of water be correct, none will be in free suspension after the first few moments on the table or belt. Henceforward the rolling action of the water becomes all important in removing the gangue, which is coarser and stands up in the swifter part of the current, whilst the friction of the specifically heavier particles on the moving bed is the all important factor in their transfer to the concentrate bin.

T. Graham Mabyn,
A.R.S.M., M.Inst.M.M.

March 20, 1915.

Prof. Stanley's Reply.

In again replying to Mr. Martyn, although the matter is rather beyond the scope of the original paper, may I point out that my remark, to which he takes exception obviously covers the whole range of concentrating appliances, from jigs to slime tables, and that I only said it *may* be a disadvantage, and that *on theoretical grounds*. But, as a case in point, since jigs work by hindered settling, classification by hindered settling would be a disadvantage.

Perhaps if I had used the word "pieces" instead of particles in the second sentence to which Mr. Martyn objects, my meaning will be clearer in conjunction with the explanation that I was dealing with the whole range of sizes dealt with by classifiers, and not the smallest sizes only.

I am well aware that there has been much controversy in the past between the advocates of sized *versus* classified and unsized feeds for jigs and tables, and that in practice sizing by screens for the finer sizes is generally probably more inefficient than hydraulic classification, and that consequently classified feed generally gives better results, while even unsized or natural feed may in many cases give good commercial results.

Mr. Martyn is, however, apparently of opinion that jigs should not be employed on any classifier products, yet he must have seen them in use in such a manner, as I have in many mills, besides the very numerous instances given in published descriptions. They are used, too, down to as fine a limit of size of particle as possible because of their efficient performance, and are usually fed with feed sized by trommel screens down to the size at which screening becomes inefficient and below that with classifier products (*i.e.*, underflow), because of the impracticability of working finer mesh screens.

If, *on theoretical grounds*, classification is preferable before concentration, why should there be such demand for, and such activity in the invention of fine screening appliances, such as the Wilfley Impact, Callow, King, Bunker Hill, etc., which are designed to carry on screening below the size limit at which ordinary trommels become ineffective?

I know, of course, that within limits, a large particle of gangue *may* be moved by water down a sloping table more easily than a smaller particle of heavy mineral: Mr. Martyn says my statement is the reverse of what really happens, but he quotes only a portion of what I stated, for the continuation was "as its size approximates to the depth of the assumed stationary film of water . . . . *eventually the case may be reversed*" and therefore, as I said before, at that point we are in agreement.

His remarks apply chiefly to the very small particles which are suitable for treatment on stationary buddles, frames, slime tables and similar appliances by a thin film of water and in a layer theoretically of one particle deep.

But with regard to the action of the Frue vanner and Wilfley table and similar machines, which are employed most suitably on particles too small for jigs and not small enough for those just mentioned; and which possess the essential features of working with some sort of agitation and with a layer several particles deep (Richards says about 10 deep); while I admit the action described by Mr. Martyn plays a considerable part, I differ as to its relative importance.

According to the foremost authority, Richards[1], stratification of the particles in accordance with their specific gravities is effected by the agitation, and he mentions the term "agitation ratio"[2] between the diameters of gangue and heavy mineral (similar to "free-falling" and "hindered-settling" ratios) and establishes it between quartz and pyrite as 10·7. Having been stratified, the travelling or jerking action of the table then transports the concentrate in one direction, while water transports the waste in another.

The action described by Mr. Martyn is given by Richards[3] in his explanation of the action of, not Wilfleys or vanners, but of what he classes as "film-sizing tables", comprising surface tables (slime tables), buddles, etc.

It seems to me that the action of agitation to cause settlement in a liquid pulp is sufficiently like that of a jig to give some justification for my assumption that the settlement is to a great extent in accordance with hindered settling laws: the particles are mixed together in a mobile mass, every one jostling its neighbours and trying to move downwards, so that the water which is present might be considered to have, relatively, an upward motion.

[1] Richards, "Ore-Dressing," Vol. II., pp. 664, 667; Vol. III., pp. 1164, 1165, 1195.
[2] Richards, "Ore-Dressing," Vol. II., p. 665.
[3] "Ore-Dressing," Vol. II. p. 685.

Mr. Martyn is correct in stating that the most difficult light waste to remove (on tables) is considerably finer than the coarse heavy mineral. My contention in this respect is that such light waste should have been removed by an efficient classifier (as also by screening).

May I conclude with a very apt quotation from another author[*]:

"In concentration the question of sizing is one which commands a very important position, and without proper attention to it the treatment of the ore is rendered very difficult. As a result of classification the ore particles have an extremely wide variance in the matter of size as has been shown, far exceeding the boundary lines of the range of sizes which would reasonably seem to give the best results. There is room for discussion on this point, as there seem to be equally honest advocates on both sides of the question. One point held is that, as the quartz particle is so much larger than the galena particle, by way of illustration, it extends further up into the flow of water and is more readily rolled. If the action of carrying off the quartz particle were that of rolling there might be some ground for this, but as the ore is stratified before any elimination of the quartz takes place, the quartz must necessarily rest on the stratum of the metallic particles."

And another[†]:—

"Amongst the great advances due to Rittinger was the recognition of the importance of sizing, and the demonstration of the fact that it is much easier to effect a separation amongst mineral particles of different density when they are all approximately the same size; his theory also indicated the degree of accuracy to which sizing should be carried. This is now looked upon as one of the fundamental principles of modern dressing, and the grading of the crushed mineral according to the size of the particles is a portion of the work of every dressing plant; it is performed by means of a large variety of screening and classifying appliances."

G. H. Stanley.

[This discussion is now closed.—*Ed. Com.*].

[*] E. A. Sperry, *Western Chemist and Metallurgist*, Sept., 1905.

[†] H. Louis, *Journal of the Society of Chemical Industry*, Dec., 1905.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Standard Tests for Lubricants.—The "Proposed Provisional Tests," of which standardized details are given, are: Viscosity test, by the Saybolt viscometer; specific gravity, by the hydrometer, Westphal balance, or Geissler pykmometer; flash and fire test, by the Cleveland open cup or Pensky Martens apparatus; soap test; saponification-value; free acid test; sulphur test; water test; precipitation test; microscopic examination; carbon residue test. In a minority report the retention of the Engler viscometer as an alternative is recommended. *Journ. Soc. Chem. Ind.*, Oct. 31, 1914, p. 1048. (J. A. W.)

Serious Corrosion of Boilers.—The feed water was very soft, having a total hardness of only 3.5° (German), the temporary hardness being only 1°. It contained 3 gm. SiO_2 and 3 gm. of nitrates per 100,000. Chlorides were present in very small amount. Boilers were worked at about 4 atmospheres pressure for many years without trouble, but when they were replaced by boilers working at 10 to 12 atmospheres, considerable corrosion was produced. After 2 months a hard crust, 6 mm. thick, had formed, and the water blown off was found to be strongly acid. The crust contained 80% Fe_2O_3 and 6% of alkali silicates. On the surface was a layer, 1 mm. thick, of calcium sulphate. The action was due to the decomposition under pressure of the chlorides and nitrates by the silicic acid. The liberated mineral acids attacked the iron, and the resulting salts becoming hydrolysed, a progressive rusting of the boiler took place. The trouble was overcome by adding sufficient sodium hydroxide to the feed water to keep the water in the boilers alkaline to phenolphthalein. This reaction between silicic acid and nitrates in boilers is considered to occur only at a pressure of nearly 10 atmospheres.—A. Heinzelmann.—*Journ. Soc. Chem. Ind.*, Nov. 30, 1914, p. 1079. (J. A. W.)

Action of Metals on Water.—The order of the metals when arranged according to their potential (referred to the hydrogen electrode) in normal solutions of their salts, is: Mg, Al, Mn, Zn, Cd, Fe, Co, Ni, Sn, Pb, H, Cu, Bi, Hg, Ag, Pt, Au; those preceding hydrogen being positive, and those following it being negative. The position of aluminium refers to the amalgamated metal; the position of the ordinary form is uncertain owing to its superficial coating of oxide. Aluminium keeps well in dry air, but oxidises superficially in moist air. Distilled water has little action upon it, but ordinary water may pit the surface; sea-water corrodes it strongly. Amalgamated aluminium decomposes dilute sulphuric acid, a contrast to amalgamated zinc. Zinc remains bright in distilled water free from gases, but in the presence of oxygen some hydroxide is formed which is sparingly soluble; rain-water dissolves it. Nickel is hardly attacked by air, moisture, sea-water and dilute hydrochloric and sulphuric acids. Pure tin is not attacked by water and dilute sulphuric acid so long as air is absent, but sea-water corrodes it in presence of air. The chloride and sulphate of lead being much more soluble than the carbonate, the presence of chlorides and sulphates in water increases the danger of lead

poisoning. The action of a potable water on lead and zinc cannot be predicted from the composition of the water. W. P. Jorissen. *Journ. Soc. Chem. Ind.*, Nov. 16, 1914, p. 1055. (J. A. W.)

METALLURGY.

Tube Mill Tonnage Calculation. It is often difficult to determine the tonnage actually being handled by tube-mills working in closed circuit with classifiers. Where either tube-mill feed or discharge can be deflected and weighed for short periods it is still a quite inexact determination, as an accurate moisture sample is essential; and where a portion of the pulp is cut out of the flow and weighed, if the amount is sufficient to accurately determine the weight of total flow, it is next to impossible to get a close sample for moisture. In any case, only by averaging a number of weighings and moisture determinations can data of any value be obtained, and as it is an awkward, sloppy test it is not run often enough to be of real value.

The following method is, I believe, original with myself, and allows a close determination of tonnage from four screen tests which are often a part of either daily or periodic routine. The four samples required for these screen tests are: (a) battery discharge, (b) tube-mill feed, (c) tube-mill discharge, and (d) classifier overflow.

The object of the calculation is to determine the ratio of dry tons tube-milled to dry battery tonnage. The calculation is based upon the fact that the minus 200-mesh material coming to the classifier is equal to the minus 200-mesh which leaves the classifier.

Let a = per cent minus 200 mesh in the battery discharge
b = per cent minus 200 mesh in the tube-mill feed
c = per cent minus 200 mesh in the tube-mill discharge
d = per cent minus 200 mesh in the classifier overflow
B = dry battery tonnage
T = dry tons tube-milled

Now the minus 200-mesh material coming to the classifier is the sum of that in the battery discharge (aB) plus that in the tube-mill discharge (cT); while the minus 200-mesh material leaving the classifier is the sum of that in the tube-mill feed (bT) plus that in the classifier overflow (dB), hence:

$$aB + cT = bT + dB$$

and

$$\frac{T}{B} = \frac{a - d}{c - b}$$

ratio dry tons tube-milled to dry battery tonnage.

Thus, knowing the dry tons passing through the battery daily, the dry tons actually tube-milled daily can be readily calculated from the routine screen tests.

It is interesting to study the limits of this ratio in practice. I should say in a general way that the minus 200-mesh in modern fine-grinding plants for the four screen tests entering into the above calculation would lie between the following limits:

Battery discharge	10	30
Tube-mill feed	2	8
Tube-mill discharge	35	60
Classifier overflow	60	90

The lowest value of $(d - a)$ within these limits would be 30 and the largest value of $(c - b)$ would be 58, hence within the above limits, tons tube-milled could never be less than half of battery tonnage. In the same way the highest value of $(d - a)$ within the limits above would be 80 and the lowest value of $(c - b)$ 27, or the maximum tonnage tube-milled within the above limits could not be over three battery tonnage. Generalizing, I should say that for mills where tonnage is being pushed, this ratio would be about 1·75 and never under 1. Noel Cunningham. *Metallurgical and Chemical Engineering*, Jan., 1915, p. 22. (G. H. S.)

Tube-Milling Tonnage Calculation and Notes on Tube-Milling.—"I have read with much interest Mr. Noel Cunningham's article on calculating tube-mill tonnage, and have carefully studied his method of calculation and the equation obtained. Mr. Cunningham has handled the subject in a novel manner, and has presented something in which all operators will be interested. A study of his calculation leads to some interesting speculation on the general subject of tube-milling, relating to current practice in all-slime cyanide practice.

In my opinion Mr. Cunningham's equation is correct for certain ideal conditions which, however, are not attained in practice. The necessary ideal conditions would be that the total tonnage be ground to minus 200-mesh, or, in other words, that the quantity of minus 200-mesh pulp overflowing the classifier be equivalent to the total battery tonnage. Of course this is not the case, as there is always a percentage of plus 200-mesh material in the classifier overflow.

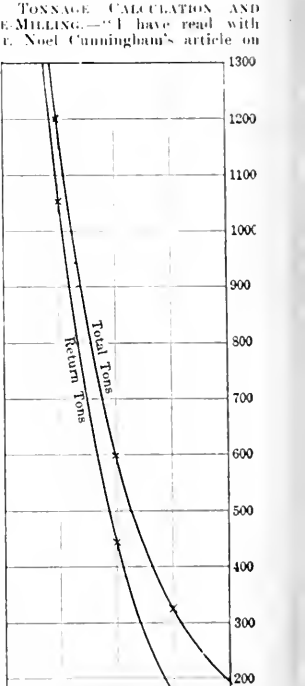


Fig. 1. Showing accumulation in tube feed due to insufficient number of tube-mills.

In order to insure having the classifier overflow all ground to minus 200-mesh, it would be necessary to have sufficient tube-mill capacity to grind all the battery oversize to this mesh, and, as it now seems to me, do this in one passage of the pulp through the tubes. Of course, if this were the case, there would be no need of running the tube-mills in closed circuit with classifiers.

All this leads to some interesting speculations on current tube-mill practice in cyaniding, with particular reference to (1) the conditions most favourable for obtaining the maximum fine grinding in a tube-mill; (2) cumulative effect of returning to a tube-mill a part of its discharge that has not been ground to the desired degree of fineness; and (3) the consequent propriety or impropriety of the "closed circuit." Mr. Pentland has touched on some of these points in his article on tube-milling, raising some of the same questions from the viewpoint of correct moisture content of tube-mill pulp.

If a tube-mill is unable to crush the entire oversize from its quota of stamps to minus 200-mesh, with one passage of the pulp through the mill, there will be a return of oversize from the classifier. This, in time, increases the load on the tube which, in turn, produces a diminishing percentage of minus 200-mesh material and an increasing percentage of oversize. If a certain amount of this oversize is allowed to overflow the classifier, equilibrium will be established; but if an attempt is made to hold the classifier oversize at minus 200-mesh, the accumulation in the tube-mill will ultimately reach a point where the machine will do little or no work. I am sure all cyanide operators have had such an experience.

Recently there was published a series of tables on tube-milling, some of which are pertinent to this discussion. One of them, given below, bears on the

TABLE I.

Relation of Feed per Mill of Tonnage of Return Feed.

Tube-Mills Operating	Tons Dry Sand per Day—Feed per Mill	Total Feed to All Mills	Total Return Feed
5	40	200	50
4	80	320	170
3	200	600	450
2	600	1200	1050

Note.—In each case the total initial feed of concentrator tailings passing to the tube-mills was 150 dry tons per 24 hours.

accumulation of oversize under closed-circuit conditions, and the number of tubes required to perform a certain duty.

A study of these figures, which are also plotted as a curve in Fig. 1, shows the accumulation in tube-mill feed due to the increasing quantity returned. When this accumulation begins to reach serious proportions, it naturally shows itself in the screen analysis of the classifier overflow; that is, by an increasing amount of plus 200-mesh material. Many cyanide operators (I have done the same thing in the past) will attempt to adjust this by changing the baffle on the classifier or by diluting the battery pulp, so that only the finest material will overflow the classifier. On second thought, however, it will appear that this merely makes matters worse and allows a greater accumulation before the inevitable equilibrium is again established by the overflow of more plus 200-mesh material from the classifier. When such a condition does arise there is no way of adjusting the classifier to prevent it, and the only solution lies in increasing tube-mill capacity. For this reason I consider any adjustment of the classifier as utterly useless. That machine cannot be made to control a condition that is created in the tube-mill and for which the latter is responsible.

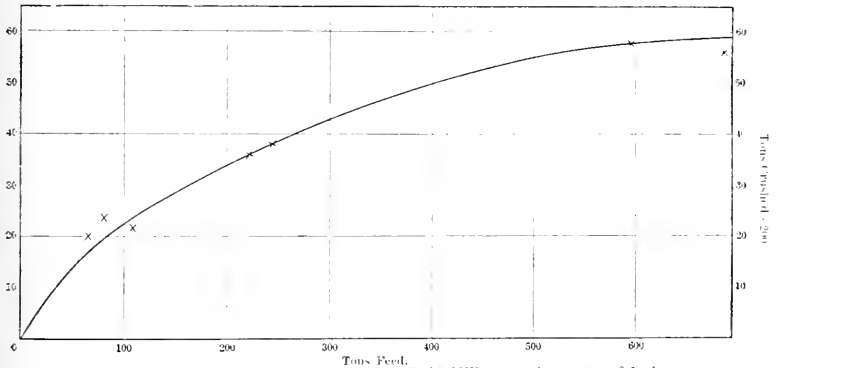


Fig. 2.—Tons crushed to minus 200-mesh in Tube-Mills at various rates of feed.

This brings up the matter of closed circuit, for if we increase tube-mill capacity to a point where it will grind the entire oversize as desired, we have then reached a point where there is no need of returning the discharge to the classifier, as the product from the tube-mill will then be practically the same as the classifier overflow.

From the foregoing it can be seen that in a plant containing, say, two tube-mills in parallel, neither of which will handle its tonnage so that the product does not need classifying, it may be possible, by putting the tube-mills in series, to obtain this result. In fact, this was proved to be the case at the Liberty Bell mill, where two tube-mills in series gave a finer product than three tube-mills in parallel in closed circuit.

Other tables appearing in the source already mentioned show the effect on the grinding obtained in tube-mills by varying the tonnage fed. I have plotted as a curve in Fig. 2 the data from these tables as relating to the minus 200-mesh pulp

produced, except that I have shown actual tons instead of percentages as originally given. This curve shows that, with an increase in the tonnage fed to the tube, the actual *quantity* of material ground to minus 200-mesh increases. But the *percentages* of that product is steadily decreasing, which shows that the feed is increasing faster than it is ground to the desired degree of fineness. In

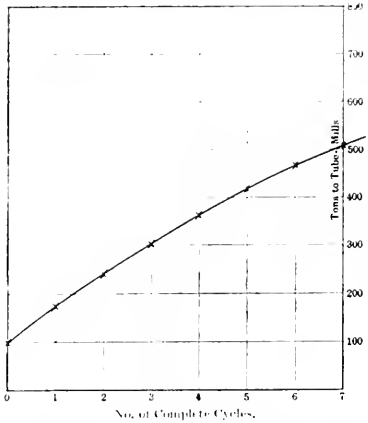


Fig. 3.—Showing accumulation of Pulp in Tube-Mill circuit with 100 tons per cycle of initial feed.

other words, the *per cent.* of minus 200-mesh material produced is approaching zero as the feed increases, and will ultimately reach a point where none of that product will be made. The following table gives the figures from which the curve is plotted.

TABLE II.

Effect of Varying Tonnage in Tube-Mills.

Tons feed 24 hours	67	89	100	217	239	570	693
Per cent. minus 200-mesh in discharge	30·03	30·15	20·01	16·56	16·00	20·28	8·09
Tons minus 200-mesh in discharge	20·12	24·10	21·84	35·93	38·21	58·59	56·06

The curve given in Fig. 3, which is derived from the data afforded by the curve in Fig. 2, also has a bearing on the subject of accumulation. Suppose we arbitrarily consider the flow of pulp as intermittent or occurring in cycles, and assume 100 tons of oversize coming from the batteries to the classifier in each cycle. Then in the initial cycle we will deliver 100 tons to the tube-mill, and of this (see Fig. 2) 23 tons is ground to −200 mesh, leaving 67 tons of oversize to be added to the next 100 tons of original feed, making a total of 167 tons in the second cycle. By continuing this calculation we get the results plotted in Fig. 3, showing the accumulation in the tube-mill circuit. The only way of preventing this accumulation is to provide sufficient tube mill capacity to grind the battery oversize in one passage through the tube, whereupon the curve of Fig. 3 would become a horizontal line. That this accumulation does not occur in practice is due to the fact that the classifier acts as an equaliser, maintaining equilibrium by overflow +200 mesh material.

The figures given in these tables and curves are not intended to represent general tube-milling practice, and the actual data given will vary widely from similar data on other ores and with other mills, but they illustrate the point I am trying to make.

The general accumulation of ore in the tube-mill circuit is enormous, and is not generally realized by tube-mill operators. As shown in Table 1, where two tube-mills were used they were ultimately grinding every twenty-four hours seven times the tonnage of the batteries; and I think I can safely say that this is no exception, as the tonnage usually ground in tube-mills is tremendous as compared with the total tonnage coming from the stamps.

While this discussion has not all been pertinent to Mr. Cunningham's article, it has been suggested by it and by Mr. Pentland's notes on tube-milling for all-slime cyanide practice. The meat of the entire matter resolves itself into the fact that with a certain design of classifier, tube-mill and stamps, working on a certain ore, the degree of fineness of classifier overflow becomes a constant when equilibrium has been established. Absolutely the only way to obtain a larger percentage of −200-mesh material in the final product is to increase tube-mill capacity, in which case the closed circuit will not be needed. To put this condition into a formula would require a separate formula for each tube-mill, and a constant due to the ease with which the ore is ground."—JUSTIN HAYNES. — *Metallurgical and Chemical Engineering*, Jan., 1915, p. 10. (G. H. S.)

NOTES ON TUBE MILLING IN ALL-SLIME CYANIDE PRACTICE.—"It is usually true that a pulp must contain 38% moisture in order to 'get it through' the tube, or to 'make the tube handle it,' but not necessarily grind it to the required degree of fineness. Under such conditions we discharge the pulp to a classifier so that the unground portion may be returned to the tube-mill, and this is done because more work is demanded of the tube-mill than it can do under the conditions prescribed. In this respect the tube mill is to be compared to a launder transporting pulp. If it is set practically level, as in the small-discharge-opening type, the more ore it has to handle, the more moisture must be supplied to transport the ore. The greater the inclination, or difference in fall between feed and discharge, the more ore can be passed as a thick pulp, or, in other words, the percentage of moisture required to carry it can be reduced.

Duty of the Tube-Mill.—It is a perfectly legitimate subject for debate whether the load going to the tube-mill should be in excess of the tube's capacity for grinding to the requisite degree of fineness, necessitating the return of a part of the pulp for regrinding, or to be so adjusted that all of the required grinding is done in one passage of the pulp through the tube. If we are depending on the tube-mill to act as a part of the agitation system, we may be compelled to return part of the pulp, but there are more economical agitators than tube-mills. If we can obtain proper grinding with one passage of the pulp through the tube, and also secure some agitation and dissolution of metal at the same time, we can well afford to build the subsequent agitators a little larger to secure dissolution of the metal that would have taken place in the tube-mill if we had returned part of the pulp to it.

The use of a closed circuit in tube-milling has become general practice, and this feature is closely related to the custom of grinding in 38% moisture, for under no conditions will pulp of that moisture

content pass through a tube-mill and be properly ground for all-slimings. At the same time we believe that there is no fixed and unalterable reason for returning part of the pulp to the tube-mill. It is the custom; just as it is custom, also, to use 38% moisture. The advocates of this practice claim that if thicker pulp be fed to a tube-mill the machine will become 'constipated' and will be unable to handle sufficient tonnage, and so for mechanical reasons it is necessary that the pulp be more dilute. As a matter of fact, this claim tells us exactly what is the matter with tube-mills of small discharge openings. It is admitted that such mills will not grind a sufficient tonnage of low-moisture pulp, and so we dilute it and circulate it through and through the mill until it is sufficiently ground.

An Experiment with Graded Pulp.—A simple experiment can be performed that will give some light on what takes place in a tube-mill. Take battery sand practically free from slime as it comes from the discharge of a classifier, and screen it into a number of sizes at intervals at 10 points between 10-mesh and 150-mesh. With each size made a pulp containing 38% moisture, and by agitation of some kind keep the solids in suspension so that the pulp shall be of uniform consistency throughout. Then, commencing with the coarsest grade, take a sample of each pulp on a spatula. A certain quantity of the coarsest grade can be raised and retained on the spatula. Of the succeeding sizes less and less can be raised until, in the case of the 150-mesh pulp, scarcely any can be secured because the accompanying moisture affords sufficient carrying medium to keep the solids in suspension. Or, if each of the pulps were to be dumped on a flat surface, we would find that almost all of the coarsest grade would remain on the plate. There would be successively smaller quantities of the finer grades, and as we approached 150 mesh we would find more and more of the ore carried away in suspension, even with 38% moisture.

We believe that this experiment indicates what occurs in a tube-mill. With the same percentage of moisture, finer sizes of sand are kept in suspension among the pebbles; and even though the whole mass of pebbles and pulp be rolling and grinding, the finer sands are still in suspension and sluicing around in the voids. The coarser sands are picked up by the pebbles and remain on their surfaces because the accompanying moisture is not sufficient to keep them in suspension. They are then in favourable position to be caught between pebbles and receive the requisite amount of grinding. We think that the pulp should be thick enough to permit the sand particles to cling to the pebbles as they rise with the mill. When the pulp is so thin that no particles cling to the pebbles, poor work is being done, and only the coarse sand is being satisfactorily ground.

It is readily seen that such thick pulp will not flow through a tube mill that has feed and discharge openings at practically the same level; but it will flow when the discharge is lower than the feed, according to the difference in levels. Tube mills of small discharge opening were responsible for the standard of 38% moisture; and, in turn, adherence to that standard is responsible for the present tendency toward shortening tube-mills, for 20 ft. or more, to 16 ft. With 38% moisture it is only of the coarser sands that the due proportion is ground, and that grinding takes place near the feed end of the mill. Then the remaining length of a long tube gives little service as a grinder and acts mainly as an agitator: therefore we shorten the tube, classify the ground pulp and return the oversize in the hope that it also will be duly ground in its next passage through the mill.

Diameter is an important factor in tube-mill design, but it is not 'the whole thing,' and increased diameter has the disadvantage of requiring more power for starting the mill. There is a tendency at present to regard 16 ft. as the most desirable length, on the assumption that practically no grinding is done beyond this point, making additional length useless. Some operators are inclined to believe that this is a mistaken idea, and that with thick pulps, or low moisture content, and a low discharge level, longer mills can be made to do satisfactory work. There is an advantage in using large units, but there is a limit to the practicable diameter, and greater capacity can then be had only by increasing the length.

Cones versus Drag Classifiers.—At one time it was customary to use cones for thickening tube-mill feed; but these devices gave trouble from the beginning, and only by unremitting care and attention could they be made to furnish pulp even as thick as 38% moisture. We believe, however, that as drag classifiers displace cones, and operators have thicker pulps to experiment with, they will regard them as more desirable. As tire-type mills come into greater use, the advantage of quicker discharge will be observed; and those who have condemned the long tube-mills will try discharging with the lifter device over the whole diameter of the tube, and will find that the long mills will grind a larger tonnage than the shorter ones.

Fresh battery sand is easier to grind to slime than is returned sand that passed through the tube mill and classifier. The former is sharp and angular; the latter is rounded. The former can be more readily picked up by the pebbles in the mill. This can be demonstrated practically by comparing the work of tube-mills on sand from old tailing dumps and fresh battery sand from the same ore.

Effect of Size of Feed on Capacity.—It has been observed that some operators deceive themselves as to the net work their tube-mills are doing. If they are treating 100 tons of ore per day, and the pulp going to the treatment vats contains 80% 200-mesh, they assume that the tube-mills are grinding 80 tons per day, forgetting that with 16-mesh screens on the battery some 38 tons to 40 tons was already ground to 200-mesh before passing to the tubes, so that the latter are really grinding but 40 tons to 42 tons to the required mesh. In this connection we recall a neighbouring operator who had 14 tube-mills under his direction, and had always estimated their work at 40 odd tons daily. On properly revising his estimates he found the figure reduced to about 28 tons daily. Feeling that the tube-mills were not doing as much as they should, he endeavoured to improve the service. By beating the mills with additional pebbles, feeding the thickest pulp his launders would carry, and substituting 6-mesh screens on the batteries for 16-mesh, he brought the tonnage up from 650 tons to 1,000 tons daily and still had 80% 200-mesh solids in his final pulp. This mill follows the practice of open circuit; the pulp does not return to classifiers after passing through the tubes, but flows through a revolving trommel that removes any coarse sand that may accidentally occur in the pulp. The trommel has a 16-mesh screen.

A personal experience may be cited. We had 200 stamps, cone classifiers and thickeners, two tube-mills 5 ft. 30 ft. and seven tube-mills 4 ft. × 22 ft. The ground was again classified in cones. Following

the dictum that the finer the sand going to the tubes the better the product, we used 35-mesh and 40-mesh screens on the batteries, yielding a product containing about 60% 200-mesh solids. From 730 tons to 750 tons could be treated, giving a final pulp of 80 200-mesh. By substituting 25-mesh screens on 100 stamps and 30-mesh or 35-mesh on the other hundred, we were enabled to raise the tonnage to from 830 tons to 850 tons, but this was the most we could get through the tube-mills. With coarse thickeners we had to give constant attention to keep the moisture down to 38 .

We then hung up 100 stamps, put 8-mesh and and 10-mesh screens on the remaining hundred, rearranged and improved the cone system of classification and thickening, and added three tube-mills 5 ft. 24 ft. This system enabled us to increase capacity to 1,050 tons daily; while battery, crusher and conveyor costs that used to total 35 cents went down to 17 cents.

Some of the tube-mills received battery pulp only; others received a mixture of battery pulp and oversize returned from classifiers; and still others received only return feed. The last was itself classified or graded by the arrangement of the distributing box at the top of the elevator, so that some mills received fine returned sand and others coarse. We had continuous opportunity to observe the difference in grinding various sizes of sand. Coarse sand dewaters easily and grinds better in the tube-mill; fine sand is difficult to dewater in cones, and is harder to grind. With coarse feeds we could get lower moisture, greater tonnage and better ground pulp; with the finer feeds it was almost impossible to get low moisture, the tonnage was reduced, and the product never contained as much 200-mesh solids as was secured from the coarser feeds.

Grinding Oxidized and Sulphide Ores. Our mill ore consisted of 950 tons of yellow, oxidized ore, assaying about 87 per ton, and 100 tons of blue, sulphide ore, assaying about 830 per ton. There was never much difference between the mine and mill superintendents regarding the tonnage or value of the yellow ore; but there was a continual difference between them as to the blue ore, as this factor entered strongly into the calculations at the end of the month. So we can safely assume that the figures for tonnage of blue ore, as agreed upon by the superintendents and given herewith, are correct.

Formerly the two classes of ore were mixed in the main bin, but on account of the more refractory character of the blue ore it was decided to give each separate treatment. Ten stamps were set aside for blue ore, and a special bin provided. Separate classifying and thickening cones also were added. Later a drag classifier was substituted for the cones. The tube-mill was 4 ft. × 22 ft. and had a discharge opening about 18 in. in diameter, being of the tire type.

Extensive experiments were conducted on kind and size of pebble load, and on moisture content of pulp, each experiment being under way for a number of days with the regular attendants on shift. Battery screens were 7, 8, 10 or 12 mesh and were varied to give the desired tonnages under different conditions.

With iron pebbles or grinders, cast iron ovals or odd bolts cut into 5 in. lengths, it was possible to grind 120 tons of blue ore daily and a get a final pulp containing from 86 to 88 200 mesh. The moisture was carried as low as 25 , though at this consistency the pulp was really too thick to work through the mill, and only about 100 tons could be handled. At 28 to 29 moisture, however, 120 tons could be treated—that is, a total of 120 tons through stamps and tube-mill. From 30 to 35 200-mesh solids was produced in the batteries, leaving only about 70 tons actually ground to 200-mesh in the tube-mill. The power requirement under these conditions was about 95 h.p., and as only a 75 h.p. motor was available, a run of a few days only was possible.

With a mixture of iron grinders and imported pebbles, or using mine rock for pebbles, the horsepower was reduced to 80 or 85, and the total tonnage treated to 100, still obtaining a final pulp with over 85 200-mesh solids.

The final conclusion was to treat 80 tons to 85 tons of blue ore daily, using mainly mine rock for grinders, with a small daily addition of imported pebbles. The moisture was maintained at 31% to 32 .

Summary of Comparison of Results.—While using 200 stamps with 25 and 35-mesh screens on the batteries, from 50 to 55 of the battery pulp was 200-mesh, and the final tube-mill pulp contained 80 200-mesh. Under these conditions each of nine tube-mills ground on an average 25 tons to 200-mesh. Since using only 100 stamps, with 8 to 10-mesh screen on the batteries, giving a battery pulp containing only 30 to 33 200-mesh, each of twelve tube-mills averages 42 tons net. Drag classifiers have now been installed throughout the mill and the cones are on the scrap heap, as also is 38 (and so many decimals) per cent. moisture for tube-mill feed.

Of these tube-mills, four were trunnion type with small discharge openings; eight were tire type with about 18 in. discharge openings. There were two pairs of 4 ft. 22 ft. tubes, each pair comprising one trunnion type and one tire type. There was one pair of trunnion type only. The remaining mills, two 5 ft. 30 ft., and one 4 ft. × 22 ft., all were of tire type with 18 in. discharge openings. With the first two pairs mentioned, it was quite convenient to compare the work of each type, and it was observed that the trunnion type never gave as good results in either quantity or quality as did the tire type.

Effect of Speed in Grinding. Grinding in tube-mills is done by the pebbles sliding or rubbing over or against one another, and not by falling against each other. Very little grinding is done between pebbles and the tube-mill liner. The revolution of the mill should be so regulated that centrifugal force will raise the load high enough to permit the pebbles to slide over each over, but it should never be sufficient to permit the pebbles to fall through clear space. The proper mean speed probably results in letting some of the upper pebbles fall clear while the main body slides. This result can be obtained better through centrifugal force than by liners adapted to carry the whole load up.

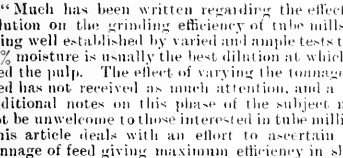
Irregular-shaped mine rock used for grinding probably does little work until it has been ground to round or oval shape, and tube-mill power for this purpose is expensive. The same amount of power expended on grinding with round pebbles would accomplish much useful work. Feeding pebbles through the scoop is slow work, and pebbles that miss the scoop and fall in the feed-box may cause the breaking of the scoop. On removing the door of a tube mill we usually find the pebbles evenly arranged throughout the mill, according to size, regardless of the method of feeding them.

When a tube-mill is grinding its full lode a distinct rise in temperature of the shell can be noted. If the pebble load is light, or the moisture content of the-

pulp so high that the increased temperature is not apparent, the mill is not doing the work it can be made to do.

Relining a Tube-Mill.—With El Oro liners a tube-mill can be relined in 14 to 16 hours. The new liners should be piled near the tube on the previous day. At 5 a.m. stop the mill, take off the man-hole plates and start the mill again, allowing the pebbles to discharge on the floor, then stop the mill again, loosen the bolts holding the liners, taking the nuts off and driving the bolts to the inside, where they are gathered up. Now start the mill again, and inside of half an hour the plates will have fallen and almost all of them will be lying on the floor beneath the mill. The new plates and pebbles can be replaced, and the mill will be in operation again by 8 p.m."—W. J. Pentland.—*Metallurgical and Chemical Engineering*, Dec., 1914, p. 750. (G. H. S.)

Relation of Feed to Tube Mill Efficiency.—"Much has been written regarding the effect of dilution on the grinding efficiency of tube mills, it being well established by varied and ample tests that 40% moisture is usually the best dilution at which to feed the pulp. The effect of varying the tonnage of feed has not received as much attention, and a few additional notes on this phase of the subject may not be unwelcome to those interested in tube milling. This article deals with an effort to ascertain the tonnage of feed giving maximum efficiency in short tube mills at the Lucky Tiger cyanide plant in Sonora, Mexico.



Relation of Tube Mill Feed to Efficiency.

The fine grinding equipment of the Lucky Tiger plant consists of five 5 ft. 14 ft. tube mills. They are lined with manganese steel liners, of a modified El Oro type, which reduce the effective dimensions to 4½ ft. × 13½ ft. The feed ends of the tube mills are equipped with spiral sand feeders, and the discharge ends with worm pebble feeders. Each mill is driven, through gearing, by a separate 60 h.p., slip-ring, induction motor. There are ammeter and watt-hour meter connections to each motor, making it possible to ascertain the power used by any mill. The speed of the mills is constant at 27 r.p.m.

Danish flint pebbles of the best grade are used, the pebble load being kept fairly constant at 7 tons. This is accomplished by regulating the feed of pebbles by the readings of the ammeter, an amperage of 58 being maintained. With a voltage of 450, and a power factor of 0·75, this indicates a horsepower of 45·5 per mill.

The tube mills receive from the concentrating plant and from old dumps, an average of 150 dry tons per day of clean, coarse sand. After passing through the tube mills, these sands are raised in bucket elevators, passed over Wilfley tables, and then distributed to four 16 ft. drag classifiers. The sand discharge from these machines is returned to the tube mills, and the slimed overflow is sent to agitation vats, thus being removed from the tube mill circuit. An average of 80 of the overflow passes a 200 mesh screen, the remaining 15 being finer than 150 mesh.

The experiments for determining the proper tonnage of feed to the tube mills were made on single mills running under actual working conditions. Special efforts were made to keep the character and dilution of feed constant throughout each series of experiments. Before beginning any test, the mills were run for 90 minutes on the same tonnage of feed to be used in the test. Sampling of feed and discharge was then commenced, the sampling continuing 90 minutes. At the end of this period the volume of feed was changed and the mill again allowed to run 90 minutes before sampling was commenced on the following test. Four times during each sampling period, the feed was diverted into a suitable vessel and the flow of wet pulp measured. Knowing the dilution and the specific gravity of the pulp, it was then possible to calculate the tonnage per hour of dry feed. The samples of feed and discharge were retained for moisture and screen analysis, and, together with the tonnage, form the basis for the comparison of efficiency.

Table 1.—*Example of Calculation of Work Done by Tube Mills.*

Mesh.	Average size of particle Inches.	R= Reciprocal of size.	Feed. % of various sizes.	Feed. R= 100 work units.	Discharge. % of various units.	Discharge. R= 100 work units.
20	0·0550	18	0·8	0·1	–	–
40	0·0245	41	3·6	1·5		
60	0·01485	81	18·8	15·8	0·4	0·3
80	0·00775	129	14·2	18·3	2·0	2·6
100	0·00615	163	15·0	24·4	6·0	9·8
150	0·00458	218	14·8	32·3	10·0	21·8
200	0·00330	303	11·2	34·0	11·6	35·1
200	0·00250	400	21·6	86·4	70·0	280·0
Work units per ton in feed ... 212·8						
Work units per ton in discharge 349·6						
Diff.= work done per ton of feed 136·8						
Tons × diff.= work done per day 6156						

Note.—Work-unit = work required to grind a ton of inch-size fragments of ore, to half-inch size.

The experiments for determining the proper tonnage of feed to the tube mills were made on single mills running under actual working conditions. Special efforts were made to keep the character and dilution of feed constant throughout each series of experiments. Before beginning any test, the mills were run for 90 minutes on the same tonnage of feed

Table 2.—*Effect of Varying Tonnage in Tube Mills.*
(Coarse Feed).

Test No.	1		2		3		4	
Tons dry sand per day	22		47		70		85	
Moisture, L:S ratio	1:1·1		0·9:1		0·8:1		0·86:1	
Per Cent.	Feed	Disch.	Feed	Disch	Feed	Disch	Feed	Disch
On 20 mesh	11·2	0·3	21·6	0·2	18·4	–	22·4	–
40 ,,	28·0	0·6	39·6	0·6	45·2	0·1	39·4	0·6
60 ,,	25·6	1·4	23·0	0·8	19·6	4·0	20·6	5·2
80 ,,	13·4	0·4	3·8	2·2	5·0	4·2	4·2	6·2
100 ,,	6·0	1·2	2·4	2·9	2·8	5·4	2·6	6·8
150 ,,	4·6	2·4	2·2	5·4	1·8	8·4	1·2	9·6
200 ,,	3·6	2·6	1·0	4·9	1·2	6·2	0·8	4·8
Through 200 mesh	7·6	91·1	6·1	83·0	6·0	71·4	8·8	66·8
Work units in feed	113		82		81		87	
,, ,, discharge	381		367		341		326	
Diff. = work done per ton	268		285		260		239	
Tons × diff. = work done per day	5896		13,395		18,200		20,315	
Power constant at 47 h.p.								

Four series of tests were run. Table 2 gives the results from the first series, consisting of four tests made on coarse concentrat or tailing, with no admixture of return feed. Table 3 shows the results of four tests run on dump sands, mixed with a small amount of return feed. Tables 4 and 5 give the results on tests run on a mixture of return feed with a relatively small amount of dump sand.

In order to reduce the screen analyses and tonnages to terms of grinding work, a modification of the method suggested by Algernon Del Mar has been

TABLE 3.—*Effect of Varying Tonnage in Tube Mills.*
(Mixed Coarse and Fine Feed.)

Test No.	1		2		3	
Tons dry sand per day	45		103		145	
L:S	0·44:1		0·46:1		0·57:1	
Per Cent.	Feed	Disch.	Feed	Disch.	Feed	Disch.
On 20 mesh	0·8		1·0		1·0	
40 ,,	3·6		8·4		5·4	
60 ,,	18·8	0·4	18·4	2·2	17·4	4·4
80 ,,	14·2	2·0	13·6	5·2	18·0	8·2
100 ,,	15·0	6·0	13·8	9·8	17·0	12·2
150 ,,	14·8	10·0	14·2	13·0	16·0	16·4
200 ,,	11·2	11·6	10·2	12·0	10·2	13·0
Through 200 mesh	21·6	70·0	20·4	57·8	15·0	45·8
Work units in feed	213		203		194	
,, discharge	350		320		293	
Diff. = work done per ton	137		117		99	
Tons × diff. = work done per day	6,165		12,051		14,355	
Power constant at 17 h.p.						

TABLE 4.—*Effect of Varying Tonnage in Tube Mills.*
(Fine Feed.)

Test No.	1		2		3	
Tons dry feed per day	109		217		570	
L:S	0·47:1		0·49:1		0·53:1	
Per Cent.	Feed	Disch.	Feed	Disch.	Feed	Disch.
On 20 mesh	0·52		0·76		0·42	
30 ,,	0·08	0·08	2·72	0·28	1·76	0·60
40 ,,	4·52	0·44	3·40	1·04	3·60	2·56
60 ,,	6·48	2·96	7·24	4·64	10·92	10·56
80 ,,	9·44	5·44	7·76	6·68	13·08	11·08
120 ,,	27·36	19·20	27·40	22·60	29·60	30·52
150 ,,	30·60	32·96	31·20	34·28	24·86	25·20
200 ,,	12·44	18·88	13·28	13·92	9·28	9·20
Through 200 ,,	8·56	20·04	8·24	16·56	6·18	10·28
Work units in feed	218		216		198	
,, discharge	264		248		212	
Diff. work done per ton	46		32		14	
Tons × diff. work done per day	5014		6944		7980	
Power constant at 45·5 h.p.						

employed. Table 1 furnishes an example of this system. The work units in Tables 2, 3, 4 and 5 were all calculated in the same manner as in Table 1, though the calculations have been omitted on account of space; only the final results being given. The work units in feed and discharge are obtained by multiplying the reciprocal of the average size (in inches) of the sand grains corresponding to each mesh, by the proportion (not the percentage) of the sample remaining on each mesh. The sum of all these products is the work units in 1 ton of the pulp sampled. The difference between the work units in the discharge and the work units in the feed gives

TABLE 5.—*Effect of Varying Tonnage in Tube Mills.*
(Fine Feed.)

Test No.	1		2		3		4	
Tons dry feed per day	67		80		239		683	
Moisture. L:S Ratio	0·45:1		0·44:1		0·46:1		0·55:1	
Per Cent.	Feed	Disch.	Feed	Disch.	Feed	Disch.	Feed	Disch.
On 20 mesh	0·32		—		0·51		0·99	0·19
30 ,,	0·76		0·96		1·48	0·40	2·25	1·14
40 ,,	1·72		2·12	0·40	2·21	1·00	4·52	3·12
60 ,,	7·20	1·10	7·80	2·87	6·43	4·65	11·41	10·84
80 ,,	8·56	2·87	12·46	4·25	12·17	9·05	12·78	12·90
120 ,,	25·78	14·13	28·40	19·60	30·17	27·80	27·98	28·50
150 ,,	30·20	35·78	27·18	29·30	24·98	27·65	24·92	24·63
200 ,,	12·82	15·10	12·62	14·35	9·50	10·26		
Through 200 mesh	10·50	30·03	8·34	30·15	8·68	16·00	5·51	8·09
Work units in feed	234		216		215		194	
,, discharge	293		281		241		205	
Diff. = work done per ton	59		65		27		11	
Tons × diff. = work done per day	3952		5200		6543		7823	
Power constant at 41·8 h.p.								

the work units performed upon 1 ton of the pulp in passing through the mill; while this difference multiplied by the tonnage per day gives the total work units performed by the mill in one day. The advantage of this system is that it furnishes a perfectly definite unit of grinding work; *i.e.*, the work required to reduce a ton of inch-sized fragments of ore to half-inch size. In Table 1, the work units per ton in the feed are shown to be 212·8. This means that in order to reduce the feed to the size shown by the screen analysis, it has required 212·8 times the amount of work which would be required to reduce a ton of inch-sized fragments of the same ore to half-inch size. This system has the same disadvantage as all other systems for determining the amount of grinding work done from screen analyses; it disregards the fact that different sands vary in hardness, and it is therefore accurate for comparing grinding efficiencies only when the material ground is of the same hardness in all cases compared.

That there are pronounced variations in the hardness of sands, even in the same plant, as shown by comparing Table 2 with Tables 4 and 5. At a given tonnage of feed to the mill, five times as much work is indicated in Table 2 as in Tables 4 and 5. Still the horsepower required is practically the same in each case. The explanation of this apparent anomaly is that in the first case the feed consists of comparatively soft fragments of the original ore, while in the last two cases the feed is composed of the harder fragments of the original ore and of particles from the flint pebbles, which were too hard to be pulverised in the first passage through the mill.

The principal lessons, however, to be gleaned from the tables and diagram are, first, that the power consumption is independent of the tonnage of sands fed to the mills, and second, that the proportion of pulverised sands in the discharge is highest for lower tonnages, and decreases as the tonnage is increased. At low tonnages (20 tons per day), coarse sand is

slimed in a single passage through the mill—the discharge containing more than 90% of 200 mesh product. On the other hand, when tonnages in excess of 200 tons per day are fed to the mill, the discharge is only slightly finer than the feed. The third point is that when we consider tonnage, multiplying the tons passed through the mills by the grinding work done per ton, we see that without a single exception the total grinding work, or "efficiency," increases as the tonnage of feed is increased.

A large feed has the same advantage over a small feed that a short tube mill has over a long one: *i.e.*, the pulp passes oftener through the classifiers, the pulverised sands being promptly removed from the circuit, and the work of the tube mill being expended more upon grinding coarse sand than upon grinding sand already sufficiently fine.

Extremely large tonnages, however, have two important drawbacks: first, the gain in efficiency is small; second, the proportion of coarse sands in the discharge is greatly increased. The small gain in efficiency for large tonnages is best shown by reference to the diagram: above a feed of 200 tons of dry sand per day, the curves flatten, indicating practically no gain in efficiency. The increase of coarse sand in the discharge is shown in Tables 2, 3, 4 and 5, and is also indicated in Table 6. This last table shows the increase of return feed when the number of tube mills in operation is reduced. As the tonnage is crowded into fewer mills, the amount of return feed increases out of all proportion to the tonnage fed to each mill. This excess of return feed must be elevated, concentrated, classified and returned to the tube mills. The extra cost of performing this work, when the tonnage of feed per mill exceeds 200 tons per day, is easily in excess of the slight reduction in cost due to increased grinding efficiency. Hence an average daily feed of 200 tons of dry sand per mill has been selected as the tonnage giving best results. This 200 ton feed includes 50 tons per day of concentrator tailing ("initial feed") and 150 tons of sand returned from the classifiers after having passed one or more times through the tube mills.

TABLE 6.—*Relation of Feed per Mill to Tonnage of Return Feed.*

Tube Mills Operating	Tons Dry Sand per Day: Feed per Mill	Total Feed to all Mills	Total Return Feed
5	40	200	50
4	80	320	170
3	200	600	450
2	600	1,200	1,050

Note.—In each case the total initial feed of concentrator tailings, passing to the tube mills, was 150 dry tons per 24 hours.

When operations were started at the Lucky Tiger plant, all five mills were run as much of the time as possible. It was soon noticed, however, that the mills grinding the return sand were receiving very little feed, and one of these mills was stopped. Later, three mills were forced to do all the grinding when the feed was light, and a fourth mill run during heavy feeds. Recently (after completing the above tests), a greatly increased initial feed was crowded into three mills, the fourth mill being run only during exceptionally heavy feeds. By this means the total daily feed (initial and return feed) has been increased to 200 tons per mill, which is the tonnage shown by the tests to give maximum efficiency. This crowding of the feed through the tube-mills has resulted in practically doubling the capacity per mill. When five mills were being run, the average daily tonnage of initial sand from the concentrating plant was barely 26 tons per tube mill. Now, with three mills in operation, the average tonnage of initial sand per mill is 50 tons per day. This increase in capacity has occasioned no decrease in the grinding work done, the screen analyses of the final overflow from the classifiers still averaging 80 through 200 mesh, as it did during the time five mills were used.

TABLE 7.—*Effect upon Pebble Consumption and Cost Per Ton of Increasing Feed to Tube Mills.*

(From Records at Lucky Tiger Mill.)

	Average Daily Tonnage of Initial Sand per Tube Mill.	Approximate Total Tonnage of Initial and Return Sand per Tube Mill.	Pounds of Quartz Pebbles per Ton of Initial Sand.	Pounds of Imported Pebbles per Ton of Initial Sand.	Cost in Imported Pebbles per Ton of Initial Sand.	Cost in Pebbles per Ton of Initial Sand, Milling, Classifying and Elevating.
First 3 months of operation (data not complete).						
Next 6 months of operation	26·1	35	10·0	17·9	$1.88	
Next 9 months of operation	40·3	80	none	15·8	1.01	
Next 12 months of operation	42·0	85	none	15·3	0.94	
Last 3 months of operation	50·0	200	none	8·2	0.68	

The power required to grind a ton of sand has decreased in inverse ratio as the capacity of the mills has increased. It now requires 22 h.p.-hr. to grind a ton concentrator tailing to such a fineness that 80 passes a 200 mesh screen. This is half what was required when the mills were run on a light feed.

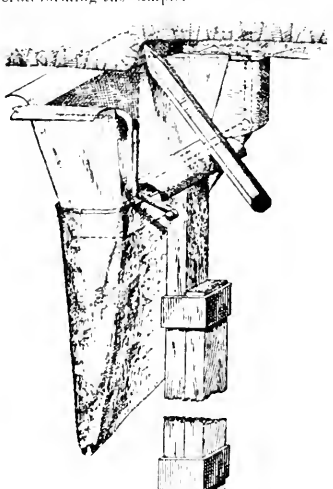
The decrease in pebble consumption has been even more noticeable. With a feed of 26 tons of initial sand per day, it was necessary to charge 2,860 lb. of quarry rock and 467 lb. of imported pebbles per day into each mill. This is at the rate of 110 lb. of quarry rock and 17·9 lb. of imported pebbles per ton of sand ground. When the feed per mill was increased to 41 tons of initial sand per day (80 tons per day of combined initial and return feed), the imported pebbles charged to each mill amounted to 640 lb. per day, or 15·5 lb. per ton of initial coarse sand. No quarry rock was used. When the tonnage was further increased to 50 tons of initial sand, or 200 tons of combined initial and return sand per day, the daily consumption of imported pebbles dropped to 410 lb. for each mill, or 8·2 lb. per ton of initial sand fed.

The cost of tube milling has been reduced in proportion to the reduction in power and pebbles consumption. With a light feed, the cost of grinding, elevating and classifying, per ton of initial sand fed, was $1.88 U.S. currency. With a medium feed, the cost per ton was reduced 97c. Now, with a heavy feed, the cost has dropped to 68c. per ton."—R. T. MISHLER. *Engineering and Mining Journal*, Sept. 12, 1914, p. 469. (H. A. W.)

MINING.

SAMPLE CATCHER FOR BACK.—"In the course of sampling the back of a level or other working, there is also a certain amount of difficulty in insuring that all of the material broken from the cut is collected without admixture from extraneous sources or loss from splintering. After experimenting with

numerous methods and devices, I finally arrived at the simple arrangement shown in the illustration, for simultaneously collecting and bagging the material forming the sample.



Catcher on Use on Stoff.

It consists essentially of a two pronged fork, the upper extremities of the prongs being bent backward, parallel to each other and at right angles to the original plane, as shown. On these bent portions of the prongs is hung a canvas or soft leather, square-sectioned tunnel, which is open at the top and bottom and also has one side partly cut out. On the lower part of this tunnel are four hooks from which the sample bag is hung. The upper part of the tunnel has wings attached which are stiffened around their free edges by being sewn over pieces of copper wire. The metal part of the fork is attached rigidly to a wooden handle made in two pieces, arranged to slide over each other and so permit of adjustment for height.

The fork when in use is held firmly up against the back." — A. LIVINGSTONE OKE. *Engineering and Mining Journal*, Dec. 19, 1914, p. 1096. (G. H. S.)

Abstracts of Patent Applications.

(C.) 508/13. Anthony William Richard Schieder (1), William Spence Wilson (2), George Innes Smith (3). Improvement in apparatus for a portable chute or stope box for use in underground workings of mines and the like.

This application has reference to improvements in apparatus for a portable chute or stope box for use in underground workings of mines or the like.

(C.) 180/14. W. A. Caldecott. Improvements in the treatment of acid or corrosive water. 27.4.14.

This application relates to means for neutralizing acidity in mine water, and consists in adding finely powdered carbonate of lime, etc., preferably in the form of a pulp.

(C.) 153/14. C. F. Wienand and William Dowsett. Improvements in linings for tube-mills and the like. 8.4.14.

This application relates to improvements in tube-mill linings, and consists in providing a liner composed of bars set on the shell of the mill, in contact with each other, of a suitable thickness of say between 1 in. and 2 in. At suitable equal distances a "higher" or projecting bar is placed to arrest the slip of load.

(C.) 369/14. W. Mauss. Improvements in centrifugal separators of the planetary type. 31.8.14.

This specification relates to centrifugal separators suitable for separating solids and liquids, the said separators being of the planetary type, in which separating drums or vessels are arranged to be whirled about a main axis to effect the separation of the material treated and are also rotated relatively slowly about their own axes to permit removal of the solid material deposited on their inner surface.

(C.) 100/14. A. V. Park. An improved mill for treating ores and the like. 14.10.14.

This invention relates to the pulverization of ores by means of a single apparatus as distinct from the several stage crushing machines commonly used for that purpose. As compared with the gravity stamp, it is a highly finished mechanical contrivance. It consists of an inner and an outer disintegrator and grinder, each having a percussive and a continuous rotary action with means for feeding and discharging the ore.

(C.) 5/15. C. G. Osgood. Improvements in or relating to filtering apparatus. 11.1.15.

This application is for a filtering apparatus in which the filter is composed of a series of spike-shaped and grooved cores covered with filtering medium, which are separately connected to the suction header. These filter-spikes on immersion in the pulp and application of vacuum form cylindrical cakes, which the inventor claims are less liable to develop cracks, and therefore enable more perfect washing to be effected.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

DOWSETT, C. W., *to* Brakpan: Dome Mine, Porcupine Dist., Ontario, Canada.

GAUERTATH, N. M., *to* Nevada, U.S.A.: *c/o* Union Jack Mine, Gravelotte, Transvaal.

JACKSON, G. T., *to* Thane, Alaska, U.S.A.

LISTH, H. M., *to* P.O. Box 3906, Johannesburg.

RICHARDSON, A., *to* Sussex: Royal School of Mines, S. Kensington, London, S.W.

STOKES, CORPL. R., *now* Lieut. R. Stokes, 174th Coy. Royal Engineers, 6th Division, British Expeditionary Force.

UNIVERSITY OF TORONTO.

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Vol. XV. MAY 1915. No. 10

Proceedings

AT

Ordinary General Meeting, May 22nd, 1915.

The Ordinary General Meeting of the Society was held [illegible] South African School of Mines [illegible] on Saturday, May 22nd, 1915, [illegible] G. H. Stanley, President [illegible]. There were also present [illegible]

16 Members: [illegible] J. Gr[illegible] J. W[illegible] [illegible] cott, W. R. [illegible] H. R. Adam, [illegible] F. H. Croghan, J. M. [illegible] D. M[illegible] Nettleton, J. A. [illegible] Thomas, W. E. Thorpe, and M. [illegible]

6 Associate [illegible] Messrs. A. King, G. Lu[illegible] M[illegible] S. E. Willows-Munn, W. J. [illegible] and W. Waters.

3 Visitors and Fred Rowland, Secretary.

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

OBITUARY.

SIR GEORGE FARRAR.

The President: [illegible] be fitting if we proceeded with the [illegible] business of the Society without [illegible] reference to the loss [illegible] the Society have sustained [illegible] death of Sir George Farrar. I [illegible] nothing to what [illegible] papers in the last few days. [illegible] English paper used a very [illegible] reference to him "A very [illegible] man". That is how we all think of Sir George Farrar. He commanded the respect of everybody. He was a real man, and the community can ill afford to lose him. I propose a sincere vote of condolence with Lady Farrar and family.

The vote was agreed to by all present standing.

ANNUAL GENERAL MEETING AND REMOVAL OF OFFICES.

The Secretary: I would like to draw the attention of members to the fact that the [illegible] meeting, which will be held on the last Saturday in June—the 26th June—will be the Annual Meeting. Members are reminded that [illegible] forms must be [illegible] on the 26th inst.

I [illegible] also to [illegible] members that the [illegible] will be removed this [illegible] to the Old Municipal buildings, which adjoin this building. The entrance is in De Villiers Street, opposite the Railway Offices.

GENERAL BUSINESS.

SPECIMEN OF SOFT SANDSTONE FROM SANDFILLING.

Dr. W. A. Caldecott (*Past-President*): [illegible] which has been sent [illegible] O. P. Powell, the Manager of [illegible] Jack. It illustrates a very [illegible] form [illegible] in the [illegible] sandstone from sand-filling, which Mr. Powell [illegible] about two and a half [illegible] in the [illegible] of his mine. He [illegible] not been [illegible] to any [illegible] rock pressure, but at the same [illegible] will see on [illegible] it has been very considerably compacted and with material of this description even under pressure the liability to causing lateral pressure against barriers is small. The compacting is apparently due to the pressure of the deposited sand itself, which is practically free from slime, and to the cementing effect of the partially oxidized iron salts and the acid leachings from the weathering of the pyrite in the sand. Probably in the course of a

few years the sand in the filled stopes will be even harder than it is at present. After this meeting the samples will be presented to the Geological Department of the School of Mines.

The President: I think it will be a very useful example of the formation of rock by consolidation of sediment. Probably when it is examined under the microscope something interesting will be seen in between the sand grains.

SCRUTINEERS FOR ANNUAL BALLOT.

The President: We have to appoint scrutineers to-night for the election of Councillors and officers for next year, and I shall be glad to receive nominations.

The following were nominated and declared appointed to act as scrutineers for the ballot for the annual election of Councillors and officers:—

Messrs. A. Thomas, S. Newton, P. T. Morrisby, E. A. Osterloh, H. R. Adam, F. J. Pooler.

NOTES ON THE PRACTICAL TESTING OF WORKING CYANIDE SOLUTIONS.

By Edward H. Croghan (Member).

(*Printed in Journal, April, 1915.*)

DISCUSSION.

Mr. Edward H. Croghan (*Member*) briefly introduced his paper, which was printed in the Journal for April.

Mr. H. A. White (*Member of Council*): I should like to propose a very hearty vote of thanks for this paper, which will be very interesting to all practical workers who have to make the determinations dealt with by the author.

The attitude taken up by those responsible is one of welcome to outside criticism, especially where experimental details are used in support of arguments brought forward, as has been done in this case; it is fully realized that the subject of analysis of working cyanides has not yet been exhausted, though practical control is now fairly secure. It would possibly be a gain, as the author suggests, if our records were kept in terms of cyanidion and hydroxidion; in the latter case most mines use Na_2HO as the base for alkalinity records, but others use the less convenient CaO, so that there is room for improvement in the direction of uniformity.

Mr John Watson (*Member of Council*): In seconding the vote of thanks to Mr. Croghan, it is a pleasure to have a paper, once more, from the author, who is a former member of our Council. His object is to attain more uniform procedure. History repeats itself, in these matters, and when I first joined the Society of Chemical Industry, we heard a lot, in those days, about uniform methods of sampling and testing. An influential committee was appointed at the Annual General Meeting, held at Newcastle-on-Tyne, in 1881, to consider how to carry into practical effect the resolutions of the various local sections upon uniform methods of analysis and testing. I have the bound volumes of the Society for that year and the following ten years; but I have looked in vain for any report on the subject. I distinctly remember, however, that many leading chemists were opposed to uniform methods as tending to cripple the spirit of research. I agree with Mr. Croghan that where comparative results are to circulate within, say, a large group of mines, it is desirable that samples should be taken properly and tested as accurately as possible. In Germany something was done, as Drs. Lunge and Hunter have pointed out (see preface to 1st Ed. *The Alkali-Makers' Handbook*). A committee of seven owners and managers of high repute was appointed and Dr. Lunge collected and sifted the material for such a manual. The result was the publication of the *Alkali-Makers' Pocket-Book*; the second edition of which, under the title *Handbook*, appeared in 1891. Dr. Hunter died several years ago; but Vol. III. of the *Technical Methods of Chemical Analysis*, by Drs. Lunge and Keane, has recently been published.

Protective alkali. To return to Mr. Croghan's paper, in defining terms, he should, I think, have inserted the word lime before the word equivalent, or he might have said, "stated in terms of lime," after the word mono-carbonates. This is a mere slip on the author's part, as later perusal of his paper shows.

CN tests.—The author shows the very different strength denoted by carrying the titration to the first white, or to the yellow point. It was customary a few years back, amongst cyanide chemists to designate the first as "free cyanide" and the second with KI added) as "total cyanide." Clennell, in his *Chemistry of Cyanide Solutions* (1910), gives three methods, by which the CN of the double cyanides of zinc, silver and

mercury would be set free and determined. My advice to all responsible for the successful running of a cyanide plant is to pin their faith to the free cyanide, indicated by the first permanent white. Titrate over a black surface and the white opalescence will be seen all the better. The extra CN contained in the total cyanide may be available under proper conditions of alkalinity, etc., but a cyanide working solution is a very complex proposition, and, no doubt, on different mines the combined CN is there in a great variety of forms.

Protective alkali.—Mr. McArthur Johnston, in *Rand Metallurgical Practice*, p. 326, says no satisfactory method has yet been introduced for the accurate determination of this factor; on the whole, I agree with him. Even after having added potassium ferrocyanide, as recommended by Green, the pink colour has a way of sometimes returning. Clennell says this is because too little silver nitrate or too little ferrocyanide has been added. These tests are usually made by the man on shift, and he, as a rule, has not time to carry out a series of investigations into secondary causes.

The Author's conclusions.—I cannot agree with Mr. Croghan's suggestion that free cyanide in solutions should be stated in terms of CN (cyanogen), but I think that commercial cyanide should be sold on its contents of cyanogen, and the merchant should be compelled by law to state whether he is selling potassium or sodium cyanide. Whilst potassium cyanide cannot by theory contain more than 39·93 cyanogen (CN)$_2$ by weight; sodium cyanide on the same basis may contain CN 53·06 . It is absurd that we chemists have now to test a sample of sodium cyanide and state our result in terms of potassium cyanide. On the present system a theoretically pure sample of sodium cyanide may contain CN 132·85 KCN .

The recording of protective alkali as OH (hydroxyl) is suggested by the author. So long as lime is the material added to give the cyanider his protective alkali, it seems to me most rational to state that alkali in terms of lime (CaO), although as stated by Bloxam (1913) "Alkaline properties are identical with the presence of hydroxyl ions." Possibly, in the future, our cyanide workers, from the consulting metallurgist and reduction officer down to the junior learner, may all have the ionic theory at their finger-ends. I fear me, by that time, the main reef series of the Witwatersrand may be about worked out.

Standard solution, etc., employed.—Mr. Croghan is out as an advocate of uniform procedure; but I notice that his standard silver nitrate solution is just half the strength of that employed in probably 90 % of the Rand extractor houses. Also, instead of working on the usual 100 cc., he takes only 50 cc. The first change makes for greater accuracy: the second leaves you, relatively, where you were before.

Mr. Jas. Gray, F.I.C. (*Member of Council*): Since the comprehensive contribution by G. W. Williams (this *Journal*, 1904, pp. 288-298, 412-419) on "The determination of constants in working cyanide solutions," no discussion has been initiated before the Society on this important branch of the gold industry, and it was in the hope of finding some further investigations which might increase our knowledge of the subject that I read Mr. Croghan's contribution. Unfortunately, I have found difficulty in following the author's remarks, because in many instances the sentences appear to be somewhat involved.

Mr. Croghan first draws attention to the two end points observed in titrating cyanide solutions with silver nitrate, using KI as an indicator, which he terms the "white" and "yellow" points, respectively, but not until the end of his contribution do we find an opinion expressed that "if the free cyanide is the factor to be recorded, then all tests should be only carried out in the presence of potassium iodide as indicator to the first distinct opalescence" . . . .

Williams, "Determination of Constants in Working Cyanide Solutions" (p. 112), says: "The first opalescence indicates the 'end-point', and further points out that the presence of calcium salts causes a precipitate previous to the true opalescence."

Mr. Croghan does not mention any difficulty in the determination of protective alkali, although Williams, who carried out a large amount of research on the subject, was of opinion that the methods of Green and Clennell were inaccurate. The method given by the author (appendix) is admitted by Clennell to be inaccurate in the presence of zinc, and Williams states that Clennell's method is inaccurate even when no zinc is present, and also points out that the addition of ferrocyanide when zinc is present, although an improvement, also fails to give strictly accurate results.

It would add to the interest of Mr. Croghan's reply if he would supply results for protective alkali obtained by the method suggested by Williams (p. 11D).

I would point out that the reference given to Williams by Mr. Croghan is to be found on p. 295, not 186 as stated.

I hope Mr. Croghan will understand that these criticisms are offered in no carping spirit, but I feel that had a closer study of Williams' paper on the same subject been made, that some results which would constitute a real advance in the estimation of the three constants cited would have been brought before us.

The President: I sympathise with Mr. Croghan in his endeavour to promote the use of a standard method from one end of the Rand to the other. It is indeed strange that different methods are worked with ease which produce quite appreciable differences in the results. It is not as if such methods were at different extremes, but rather it is hard to find two methods which do not yield concordant results; but we know that these results differ, and it is very difficult to compare practice on different mines in that respect.

The vote of thanks was agreed to.

THE THEORY OF TUBE MILLING.

By H. A. White (Member of Council).

(Printed in February Journal, 1915.)

REPLY TO DISCUSSION.

Mr. H. A. White *(Member of Council)*: I have to thank the gentlemen who took part in this discussion for their kind remarks, and to regret that so few members took advantage of this opportunity to bring our records on this subject up to date.

With reference to the formula devised by Dr. Caldecott, I would remark that on many mines a record is kept of the tonnage of 3·90 grade produced per mill per 24 hours, and that grading of tube mill and outlet is usually recorded daily, so that with the information already at his disposal it is very easy for the operator to calculate and record the daily tonnage fed to tube mills. No complications are caused by any consideration of classifiers or return sand, but this advantage is also secured by the system proposed by the late Mr. G. G. Smart (C.M.M.S., Vol. x., p. 154), in which the moisture tests are made to perform a similar service where clean water is used for diluting the tube feed. The definitions given by Dr. Caldecott represent a valuable thought on the subject which will be highly appreciated.

Mr. Dowling's contribution was of especial interest in the description of the diaphragm box which is working at the Simmer and Jack with the minimum loss of head. Having seen its smooth and regular operation, it is easy to realise its usefulness where a loss of available fall may cramp operations upon the extension of any existing plants.

The idea of safety cones after tube-mill classifiers is, as far as I am concerned, due to Mr. J. E. Thomas, and it was after he had discussed the question with me that the matter was adopted at the Princess Estate, where, I believe, the priority of application may be claimed. It will be obvious that it is a cone of long cones than are necessary for the grading aimed at are installed, for any given tonnages, which must be superseded by coarser grading, may be dealt with either by decreasing the number of cones so employed or allowing these used to fill up with sand and so decrease the working area to a point which may be shown by a simple sliding wire indicator. Except when the tonnage required is right up to the maximum capacity of the plant the use of the sand return cone after tube classifiers certainly minimises the advantage to be gained by a perfect pulp distributor such as is used at the Princess Estate and the Geduld Proprietary Mines, but it is probable that enough remains to make it worth while to pay the small maintenance costs and for the less of head involved.

NOTE ON A METHOD OF VENTILATING SINKING SHAFTS, WINZES AND DRIVES.

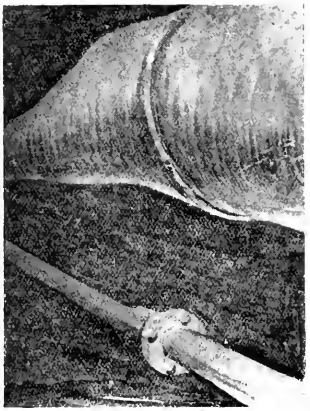
By S. Nettleton (Member).

Printed in Journal, February, 1915.)

REPLY TO DISCUSSION.

Mr. S. Nettleton *(Member)*: I believe that the only comments which have been made on this paper were made by Mr. Barry two months ago. He referred to the durability of the canvas tube for ventilation purposes; and this is probably the

weakest feature of the arrangement. I think that, however, this difficulty is not insurmountable. The arrangement which I suggested has been tried on the Nourse Mines. A conduit 600 ft. in length was installed in a drive where the ventilation conditions were very bad. There has been considerable improvement in these conditions, though the canvas has not been carried so near the face as appears to be desirable. The tube has not been in use sufficiently long to give an opinion as to its durability. The shock from blasting does not apparently damage the conduit. I found no difficulty in moving sections of tube near boxholes where blasting was necessary. With a couple of boys I took out sections from



Joint in Canvas Air Pipe.

the conduit along the drive and replaced these sections in twenty minutes. Practically the only damage that has been done to the tube has come from unexpected sources. Some of the native trammers discovered that the canvas material might be used for wearing apparel, and they have cut sections out and taken them away. I also found oil had got on some of the canvas and that the rats had sampled it. These difficulties might be got over by putting a broad arrow or some other distinguishing mark on the canvas and treating it with some material that is not palatable to the rats. The installation of canvas tube on the Nourse Mines is on a drive which follows the reef and has scarcely 30 ft. in one straight line. It would have been impossible to put wooden launders in this drive without great expenditure, owing to the bends and irregular section of the drive. The canvas conduit appears to be quite satisfactory for the purpose for which it was intended. On a straight drive it should be supported by hooks at a distance of about 20 ft. In going round corners the tube would require supports closer together. It has been pointed out to me that the canvas arrangement would not be airtight, but the air pressure in the tube is so low that there is very little leakage, and with a low pressure there is a good delivery of air at head of the drive.

THE USE OF SCOOP DISCHARGES IN TUBE-MILLS.

By W. R. Dowling (Past-President).

(Printed in Journal, March, 1915.)

DISCUSSION.

The President: I personally feel very disappointed at the very poor compliment paid Mr. Dowling in the absence of discussion on his paper. It was a very important paper indeed, and surely those who have similar experiences on other mines should give their experiences also. Perhaps it would be advisable to keep the paper open for another meeting; but the Council will deal with that.

SAFETY MEASURES IN THE USE OF EXPLOSIVES.

By Wm. Cullen (Past-President).

(Printed in Journal, March, 1915.)

DISCUSSION.

Mr. E. Pam *(Vice-President):* Mr. Cullen in beginning his paper apologises for having repeated arguments which have been used at other times in lectures under the auspices of the Safety First Committee and also by himself before this Society. I think we can justify him at once from our point of view, as I think we realise that any repetition of the important points which he has brought forward can only assist us in further decreasing to a great extent the accidents which even now are occurring. There are one or two points in Mr. Cullen's paper which I would like to discuss in detail, the most important being that in which he men-

tions his opinion that only a small proportion of miners use any mechanical means of fastening the fuse to the primer. I enquired of two mine captains their opinion on the matter, and they replied that the practice was most unusual. I am afraid that is true. In connection with the Nowag clip Mr. Cullen is possibly doing us some injustice. I know that I and some other mine managers tried to get these as soon as we saw them at last year's Mining Exhibition, and it is only owing to the war that none have come forward and that we have not been able to try them. I feel certain that they will fulfil the purpose for which they are intended and that a very large majority of the mines will make use of them.

Detonators.—Mr. Cullen recommends No. 8 detonators in the place of No. 6. I may mention I discussed this matter with him about a year ago, and his reply was if we used Modderfontein powder we could use No. 6 detonators; if we used anybody else's we should use No. 8. He points out that some gelatine is less sensitive, and mines in our group and several other groups have now given up No. 6 detonators, although in a tremendously large proportion of cases the No. 6 would be absolutely efficient. The crux of the matter, from our point of view, is the personal factor, and naturally as a manager of a mine I am bound to take it from our point of view and the miner would naturally take another attitude. We have found that, in spite of serious efforts by ourselves and inspectors of mines, men will not appreciate that we try to keep the regulations, not with a view to saving cost, but with a view to saving life. Every year, in fact every month I might say, we hear of accidents, and, in the majority of cases, avoidable accidents. If we could only induce men to attend meetings of this description, or to read this paper, there might be some possibility of their really appreciating the importance of keeping the regulations.

Mr. Cullen points out the danger of gelatine exploding owing to the heat of the rock when a man is charging up his round after blasting his cut. The new regulations state: "No man may return to his face within half an hour of blasting," and I think that within that half hour the rock would be sufficiently cool to prevent any possibility of its setting alight to or exploding the powder.

Reverting to the question of "Safety First" and the proposed circulation of this paper, it might be advisable for the Society to approach Mr. Cullen and ask him for his permission to republish this paper in pamphlet form, or to get the Safety First Committee to do so. They have published a short memorandum by Mr. Cullen, in which he lays stress on exudation, charging of holes, and detonation; but it is a very short memorandum, and, I do not want to say anything rude, but not particularly readable. Whereas, I think, if this paper were circulated in full, it might reach and be read by many men who have never considered the matter seriously, and it would in that way possibly avert certain accidents which are now occurring. I would like to support the vote of thanks to Mr. Cullen. I am afraid it may be his last paper to us for some time, and I think we should feel particularly gratified that he took the time, at a period when he was just leaving for Home and was probably very busy, to write this paper for our Society.

Mr. J. D Marquard (*Member*): I learned from this agenda that this evening was the last for final discussion on Mr. Cullen's paper on "Safety measures in the use of explosives," and have had no time to prepare any notes.

Mr. Cullen says that even to-day some of the essential points are not appreciated, that is only too true. I agree with him that accidents would be of less frequent occurrence if necessary precautions were taken. He hits the nail on the head when he states that a proportion of these accidents is preventable; I will drive that nail further in, and say the proportion can be a high proportion. I again agree with him when he says that the quality of explosives and accessories supplied to the market is good. I am doubtful if there is much further room for improvement. He mentions that by far the most fruitful cause of accidents is drilling into unexploded charges, and then five reasons are given why unexploded charges remain, viz.:—

(a) Neglecting to securely fasten detonator and fuse to the primer cartridge.
(b) Cut out shots.
(c) Insensitive explosive.
(d) Bad charging up.
(e) Damp detonators.

Recently when I lectured on accidents due to explosives to miners, I tabulated some 35 causes for misfires:—

Causes of Misfires.—(1) Improper distribution of detonating force ;(2) weak detonators; (3) damp detonators; (4) sawdust in detonators; (5) grease in detonators; (6) working ends of detonators pointing wrong ways; (7) axes of detonators and charges out of alignment; (8) detonators setting charges alight; (9) improper fixing of detonators to fuses; (10) improper fixing of detonator and fuse to primer cartridge; (11) non-greasing fuse and detonator for wet holes; (12) choking fuse; (13) fuse damaged by tamping rod; (14) irregular fuse; (15) deteriorated fuse; (16) wet fuse; (17) bunching fuses; (18) omitting to light fuse; (19) bad quality of explosives used; (20) deteriorated cartridges as primers; (21) deteriorated cartridge in between other cartridges; (22) air cushions between cartridges; (23) tamping in between cartridges; (24) improper tamping or confinement of charges; (25) exudation; (26) running fuse (so-called); (27) improper and unscientific charging; (28) explosives forced into cracks after explosions; (29) explosives burning and smothering charge; (30) cut outs; (31) insensitive explosives; (32) over-charging); (33) improper cleaning of sludge out of hole before charging; (34) contraventions of the Mining Regulations; (35) disobedience, indifference, carelessness, etc., etc., etc.

Improper distribution of detonating force.—Many think that in a fully charged hole of, say, five cartridges and a primer, that the detonator sets off the primer first, it in turn sets off cartridge No. 1, No. 1 sets off cartridge No. 2, and so on to the bottom of the charge. That may happen, but is not usually the case. What really happens is that all the cartridges behave as if they formed one long cartridge, and the force of detonation is conducted right through from top to bottom like a streak of lightning. The same detonating force that sets off the primer sets off the last cartridge practically at the same instant, so that all act in conjunction to shatter the rock. If the whole charge did not so act simultaneously, then explosives would not be so useful. Thus it will be seen if the detonating force is not distributed to the bottom cartridge how the bottom cartridge can remain intact and cause a misfire.

Weak detonators.—If a No. 6 instead of a No. 8 is used the detonating force may not be strong enough to reach the bottom of a charge, one or two cartridges may be left intact, against causing a misfire.

Damp detonators.—An experiment first made in America will explain this. A box of newly made No. 6 detonators had all the sawdust removed, and then the detonators were hung over a basin containing wet sawdust, the open ends downwards, of course. The evaporated moisture from the drying sawdust moistened the fulminate. *After 24 hours* six of these detonators fully exploded six charges consisting of five cartridges of blasting gelatine and a primer. *After 48 hours* another six detonators were taken, four did the work properly, two left the last cartridge in each charge unexploded. *After 72 hours* three left last cartridges unexploded, three left last two cartridges unexploded. *After 96 hours* two partially exploded charges, four set charges alight. *After 120 hours* with all nothing happened.

These experiments show how misfires can occur with damp detonators. Damp detonators also cause soft shots, improper explosions, gassing, and other complications.

Sawdust in detonators.—If the sawdust is left in the detonator, the fuse sets the sawdust alight. There is not enough oxygen for the burning of the sawdust, it smothers out before the heat gets to the fulminate, result misfire.

Grease in detonators.—In connection with wet holes, some miners dip the ends of fuses into grease and then fix them to the detonators, instead of first crimping the detonators on to the clean fuses and then greasing the junctions with hard grease. I made about 10 experiments thus, and found in four cases detonators did not explode.

Working ends of detonators pointing wrong ways.—If the fulminate end is placed pointing in the direction of the whole charge it will explode all the cartridges. This was an experiment conducted by two well-known American engineers. To some extent I verified it. Now it is said that if the detonator is placed in the middle of 12 cartridges in line, the fulminate end will explode six and the other end will explode none. I found that generally one or two cartridges exploded on the open end side of the detonator as well. In shaft sinking it is common to have the primer cartridge at the bottom of a hole, the open end covered by five cartridges. Will all the top cartridges explode or leave some intact?

Some miners put their primers in ordinary stoping holes upside down with the detonators foremost, and the fuse coming along the side of the primer. After explo-

sion, therefore, some cartridges may remain unexploded in the bottom of the hole, for the working end of the detonator in the primer pointed towards the collar of the hole. Result, misfire.

Axes of detonators and charges out of alignment.—Consider three cases: a) Working end of detonator pointing towards a cartridge; (b) working end of detonator lying parallel with a cartridge; (c) working end of detonator pointing away from a cartridge.

In (a) position the detonator can be placed twice as far away as (b) position and have the same effect on explosive. In (c) position the detonator will not set off the cartridge it more than 1 in. away.

Again, explode a detonator in a block of lead, and the shape of the cavity formed is like an inverted pear, the bottom end of the pear-shaped hole being twice as far away from the detonator end, as its sides form the cylindrical sides of the detonator.

These experiments conclusively show that the main force of an exploding detonator passes out at the back or fulminate end of a detonator. To further illustrate this, two rows of 12 cartridges were taken, in one row the detonator was placed in the primer cartridge so that its axis and the axes down the centre of the charge were in one straight line. On explosion all cartridges exploded. In the other row the detonator was placed at right angles to the position of the detonator in the former line of cartridges. Result, eight out of 12 cartridges exploded. Therefore, if no misfire is desired, place axis of detonator in line with axis of charge.

Improper tying of detonator and fuse to primer cartridge.—To illustrate how this causes misfires I will give you the result of nine experiments I carried out personally, each experiment was conducted three times. About 1 oz. of blasting gelatine was put into each of a number of test tubes:

(a) Three were properly charged with fuse and detonators, and lit; result, all explosives exploded.

(b) Three were charged with detonators 1 in. away from explosives; result, all explosives exploded.

(c) Similar to (b) but tamping was placed in between the detonators and explosives; result, all explosives exploded.

(d) 2 in. between detonators and explosives; result, all exploded.

(e) 2 in. between detonators and explosives with clay tamping in between detonators and explosives; result, two fully exploded, one improperly exploded.

(f) 4 in. between detonators and explosives; result, two exploded and one charge set alight.

(g) 4 in. clay in between; result, one charge exploded, and in two only detonators exploded.

(h) 6 in. between detonators and explosives; result, all charges set alight and all detonators exploded.

(i) 6 in. clay in between; result, one charge exploded, and in the other two only detonators exploded.

A miner once told me he could always feel when charging up if the detonator separated from the primer. I have no doubt one could, that is if the fuse is held in the left hand and charging stick in the right. How about the man who only uses the tamping rod and tells his helper to hold the fuse aside?; if that helper's hand does not advance at the same rate as the charging stick, then the detonator separates unknown to the man.

I said to the miner, "Have you ever felt it separate?" "Oh, yes, many times," he said. I said: "What did you do then?" He said: "I generally put my tamping in then, and trust to luck that the tamping will force the detonator against the charge." It is this "trusting to luck" that is a serious cause of many misfires, and often results in other serious accidents at charging up time.

Choking fuse.—I have often crimped detonators on to fuses very hard, and in some cases found that the detonators did not explode after lighting the fuse; in six cases it was found that the fuse side-spit just at the junction of the fuse and detonator. Another cause for misfires.

Irregular fuse.—Incontinuity of firing line in fuse.

Deteriorated fuse.—One thing I would like to mention, and that is that many miners store their fuse in boxes where they keep their machine oil. Oil gets to the powder core and destroys the continuity of the fire line.

Bunching fuses.—Some miners (especially in shaft-sinking or in winzes where there is water), coil their fuses round a floating lagging pole so as to keep spits dry. When the first hole goes off stones strike the lagging pole and some of the remaining fuses are drawn out or damaged; result, misfires.

Air Cushions between cartridges.—Mr. Cullen has fully explained this. I made

various experiments in the open air, and found that 2 in. air space in between cartridges is often sufficient to arrest distribution of detonation.

Tamping in between cartridges.—It may be of interest if I mention that with various experiments I found that 4 in. solid rock in between cartridges often passed on the detonating force, whereas 4 in. of tamping did not do so.

Improper tamping or confinement of charges.—Tamping prevents misfires, for

(a) It prevents fuses being jerked out by concussion or flying rocks.

(b) It turns an inflammation into a detonation.

(c) It makes weak detonators more effective, the force is confined.

(d) It causes more complete detonation and explosion, thus preventing noxious fumes.

(e) It helps old deteriorated explosives.

(f) It helps unbalanced shots.

All these may prevent a cartridge or two remaining behind in a hole.

Exudation.—Mr. Cullen has gone into exudation thoroughly. I would like to mention one thing that is often unconsciously done in magazines both above and underground. Boxes of explosives are often allowed to stand on end, with the result that the bottom cartridges in the bottom packet gradually have nitroglycerine squeezed out by the approximate 50 lb. pressure resting on them. If cartridges are kept in hot drives for some little time, then nitroglycerine starts to sweat out.

Running fuse (so-called).—There is no such thing. The theory of the action of burning fuse is against it. What may be mistaken for a so-called running fuse is:

(a) Primers at bottom of holes, fuse near top of charge sets explosives alight, temperature rises to 285° C. and charge explodes before fuse has burned down to detonator.

or (b) Side of hole smeared with blasting gelatine or film of nitroglycerine, fuse sets these alight, and then there is a quick fire line to the detonator.

(c) Short-circuiting fuses owing to excessive coiling.

(d) Time being wasted at lighting up time.

(e) Accidentally cutting fuse too short.

(f) Bits of burning gelatine falling into hole from the old-fashioned gelatine firing stick and setting fuse alight lower down.

Improper and unscientific charging.—

(a) Clean out bore holes properly.

(b) See that holes are correct gauge and bottom cartridges are against bottom of holes.

(c) Do not forcibly press or ram.

(d) Use good, fresh, non-exuding explosives.

(e) Do not bunch cartridges, *i.e.*, put in more than one at a time.

(f) All cartridges must be in direct contact.

(g) No loose detonators must be put in between cartridges.

(h) No tamping must be put in between cartridges.

(i) Cartridges must not be doubled up or broken across their short axes.

Disregard of these and other points already touched on result in misfires.

Explosives forced into cracks after explosion.—The force of the top cartridges exploding cracks the rock about that bottom cartridge which does not explode, and the same force forces explosives from that bottom cartridge to fill those cracks. Drill into one of these cracks and the result is evident.

Explosives burning and charges being smothered.—There is enough oxygen in an explosive for complete explosion. But explosives require oxygen to burn completely, that is, oxygen from the air. In a hole it is confined behind tamping, the burning explosive may smother out before it gets to a temperature of 285° C (the exploding temperature), thus unexploded sticks may remain, especially as it is well known that even detonators can set charges alight.

Overcharging often results in probable burning of charge and ultimate misfire. We use twice as much explosives here per ton of rock broken as in Australia, and our stoping conditions are somewhat easier; there is something to look into for an explanation.

I see Mr. Cullen's point about copper-sheathed tamping rods, but I think there are other considerations which outweigh his point. I will just repeat what I said to miners in my lecture. "This long wooden rod I like best as a charging stick, brass and copper rods are somewhat expensive and a little too heavy. There is no knowing what force a powerful miner may exert when charging with one of them. Then I do not like these ends of this wooden rod, they get frayed and little bits of grit get there and stick there. These little bits of

grit are dangerous. I suggest the supply of two copper caps for the ends of this charging rod. One cap 1 in. in diameter, the other 1¼ in. in diameter. Why? Because:

(a) You will then have nice, smooth, clean non-gritty ends.

(b) You cannot be tempted to pare down the end of your charging rod, so that it will be able to go into a hole out of gauge. Pointed rods are dangerous to charge with.

(c) You can hear the ring at the bottom of a hole with either side of the rod, and know what size cartridges are required.'

It has been determined that nitro-glycerine evaporates at 185° C., burns at 217° C., and explodes at 275° C. I got fuse from a mine, I tested it and found it did not sidespit. What I did notice was that the walls of the fuse were badly damaged and got very hot.

(a) Three times I laid a bit of burning fuse alongside a cartridge of blasting gelatine, and found in all three cases the papers of the cartridges were slightly scorched.

(b) The same experiments were tried with the fuse pressed hard against the cartridges and again only the papers were scorched.

(c) I then took the papers off the cartridges, leaving the explosives bare. It was found that out of six experiments when the burning fuses were not pressed against the bare explosives, the explosive got alight only once. Under pressure, however, the explosives got alight four times during six experiments.

Follow me to holes in the bottom of a shaft, where in many cases primers are put in first. Each cartridge as it is pressed home bursts and bare explosive is exposed to the side of the hole and may come in contact with the fuse. That bare explosive will start to burn, not from spitting fuse, but from the heat coming through the walls of the burning fuse.

At one time I fancied the idea of the primer at the bottom; these experiments have taught me my idea was wrong. Experience teaches.

Mr. R. A. Barry (*Member of Council*): The ground has been so well covered by Mr. Cullen and those who have followed him that I fear I can hardly hope to find any fresh points, and I trust repetition will be pardoned.

Whilst I suppose there is always room for improvement in the quality of materials—explosives, fuses, detonators, and such like—and whilst difficulty and divergence of opinion will always exist as to how far it is wise to reduce the prices paid for such stores—(the decision as to where the real economic line lies)—yet I think on the whole we are pretty well served here to-day with regard to quality of stores in connection with blasting operations. Occasionally we come across bad samples of gelatine or gelignite—and we may have some faults to find with a consignment of fuse; even detonators are not always beyond the range of criticism, yet I see but little real margin for improvement in these directions, as far as quality goes. A manager of a mine always hopes to live to see prices reduced without any fall in quality, but then if he were not an optimist he could never hope to manage a mine and retain his reason. Where the big margin for improvement does seem to me to lie is in the men who have to handle this stuff daily, and in their methods. We are probably worse supplied to-day on this field with a sufficient number of really competent, skilled and careful miners than ever we have been. Good men exist, but there are not enough to go round: one sees it in every direction. The marvel is not that we fail to do more to reduce accidents, but that we have succeeded as well as we have. Of course, we must go on trying with things as we find them, but to my mind our *big effort* should be bent on the attempt to improve not the material which we use in blasting operations, so much as the *human material* that conducts the operations.

One constantly finds men styling themselves miners who have been underground but a few months at most. They have never had any real teaching in mining, and have fallen into the very worst habits in all sorts of directions. The possession of a blasting certificate imbues them with the idea that they are above being taught. They do not know or dream of the dangers they daily incur, and force others to incur. If they are criticised they leave the mine. Jobs are plentiful all along the Reef for underground men.

Familiarity, it is said, breeds contempt—and so it does, and it is often contempt which leads a man to use his teeth to a detonator ,or to sit on the bridle of the skip; but what shall we say about ignorance? Ignorance surrounds you with danger, and though "fools may rush in", it so often happens that they rush in, or send other rushing in, to their destruction.

Of course, not 1 of the gangers ever tie their detonators to the primer—many cheerfully break sticks of gelatine and run the detonators into them, some will vigorously force the explosive when it shows any tendency to jamb in the hole, misuse their fuses, and take a hundred and one unnecessary risks. The biting of detonators on to the fuse is perhaps less common than it used to be, but even to-day it is by no means a rare performance.

Blasting accidents are numerous, as has been stated. The cure lies not so much in improving the supplies as in improving the men. What we want is a better supply of steady underground men, less shifting from mine to mine, a habit which is as bad as ever it was; and a longer period of training. Much of the trouble really lies at the door of miners' phthisis. Miners have ceased to come to this field from oversea in appreciable numbers and, as a rule, men born in this country only take up mining when they find themselves with their back to the wall and the wolf at the door. The fact that silicosis has laid so many underground men low is so widely known and dreaded that, despite successful efforts to improve the position, many men regard underground employment as the resort of the desperate. This is bound to have a very far-reaching effect upon everything underground, and especially on the incidence of accidents. When one thinks of the enormous force of unskilled and utterly untutored workmen employed underground, who are absolutely in the hands of those immediately in charge of them, it makes one tremble at the trend of affairs as far as skilled supervision goes. If we remember that to-day the natives are being trained by white men who are often far from careful, and that the tendency amongst the miners themselves is to put more and more responsibility and work on the native, we will see how important it is to make an effort to prevent bad and careless practices from becoming a habit.

The remarks that have been made with regard to the practice of recharging holes which have failed to break at the first blast, and often recharging them whilst they are still hot, show once more how difficult, if not impossible, supervision is at blasting time, and what extraordinary risks men will take.

Recent Mining Regulations have laid down a careful procedure when the practice is followed of returning to blast the round after the cut has been blasted, but all the evidence we have shows so clearly how difficult it is to ensure the faithful observance of necessary precautions, that one can only hope that every effort will be made to avoid the cut and round system entirely. It cannot help us in our fight against silicosis or against accidents to practice a system so open to such serious abuses.

With regard to alleged "running fuses" one frequently hears wild statements about accidents arising from "running fuses," and though the possibility of such an occurrence cannot be absolutely refuted, yet I have never met with really clear evidence that the fuse has been to blame in such a way, and in many cases where such an accusation has been made, it has subsequently been disproved when all the evidence has been carefully gone into.

Quite recently a blasting accident occurred in an incline shaft which I am sinking. It is a big shaft 35 ft. wide. Some 32 holes were being blasted. The first hole lit, went off before the firing party had left the face, and the ganger at once declared that the fuse had "run". Exact proof of the cause of such an accident is practically unobtainable, but fortunately, as no one was killed, I was able to collect enough evidence to show that there were other factors—carelessness in charging up and in firing, which were far more likely to be the real cause of the trouble; and in all similar cases within my experience I have found the same thing—a fuse may *delay* or go right out, it may even exceed its proper rate of burning, but one can scarcely see how it can do so to such an extent that it could set the charge off considerably before its proper time, at any rate where a good class of fuse is used, unless there were other causes at work, such as exudation from the explosive when charging up.

"Cheesa" or firing sticks in general use to-day are capable of much improvement, and many of us will await with interest the advent of the perfect article which Mr. Cullen refers to.

The red wrapping of explosive cartridges has been found useful at times in practice in rendering a loose cartridge lying amidst broken rock more easily distinguishable; whilst the Modderfontein Factory's scheme of providing cartridges of a smaller diameter for the bottom of holes is most distinctly a good move.

Stale fuse, stale explosive, or stale detonators undoubtedly tend to increase the chance of accidents; the obvious remedy of

course being to see that stores in hand are used up before freshly arrived stock is issued.

Crimping pliers.—I agree that crimping pliers are often badly designed, and submit three samples. No. 1 has an unchecked shearing action, which is apt to damage the fuse and cause misfires. No. 2 and No. 3 are both better articles, and attention to this matter is well worth the while of those responsible for the purchase and issue of stores.

Quality of fuse.—I think that the regular and reliable rate of burning of the fuse is most important, as much less difference in length of fuses is necessary when the rate of burning can be relied upon, and hence there is far less liability to misfires from "cut out" fuses.

Storage underground.—More care might well be expended in the storage of fuse underground, and in the necessary organisation to prevent the use of stale stores of all sorts.

Rough usage.—The prevention of rough usage to fuse is again a matter of education.

Natives firing holes.—The practice, which is fairly prevalent, of allowing natives to do the firing of holes, should be discouraged, at any rate in firing development ends where the white man can easily fire all his own shots. A native is very apt to light holes in the wrong sequence, and this causes misfires from cut outs. Where the ganger must have assistance from natives, care should be taken that the native is given the work where he can less easily make such mistakes.

Removing stoppers.—I have recently come across two cases where gangers cheerfully admitted to the practice (despite Regulations) of removing the wrapper from several cartridges of explosives entirely—they argued that they were thus able to squeeze more explosive into the bottom of the hole and thus get better breaking results!

Mal-practice in charging up.—Placing the primer in the middle or at the bottom of the hole and the use of tamping between cartridges is far from common, and should be discouraged. Those who argue in favour of such practices forget that one accident is much more far reaching and harmful than years of slightly increased efficiency can atone for, and I do not myself believe that the claim of improved efficiency can be really proved.

Of all the possible and probable causes of accidents from explosives Mr. Cullen and others have touched upon, I think the most prolific to-day are ignorance and lack of experience and carelessness.

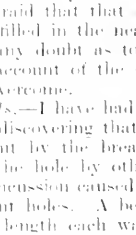
Mr. E C. J. Meyer *Member:* It was with great interest that I read Mr. Cullen's paper on "Safety Measures in the use of Explosives" in the March number of this *Journal*. The various points dealt with are of extreme importance.

I wish to endorse Mr. Cullen's statement that the quality of explosive supplied on the Rand is good, not only where effectiveness is concerned, but also as regards safety in the handling.

I quite agree with him that the main cause of unexploded charges is the neglect of precaution to fasten detonator and fuse to primer cartridge. This appears a simple matter, and I wish to support Mr. Cullen's efforts in getting all men connected with mining to realize the importance of this point and to spare no pains in the endeavour to have it remedied. It might be mentioned that inefficient attachment of detonator to fuse is also a prolific source of danger, and time spent in drawing the attention of miners to this small point may tend to decrease the accident rate.

As regards the suggestion "that the mining effect could be settled once and for all by making the bottom cartridge the primer and firing it by an electric detonator", I am afraid that that is too great a wish to be fulfilled in the near future, not that there is any doubt as to its effectiveness, but on account of the practical difficulties to be overcome.

Cut out shots.—I have had the fortunate experience of discovering that not only are primers cut out by the breaking away of the collar of the hole by other shots, but also by the concussion caused by the explosion of adjacent holes. A bench with five holes of 6 ft. length each was charged up in the usual way, but only No. 1 hole was ignited, and 30 minutes after the explosion the bench was again examined. To my great surprise I found that the primer of



Front View of Bench.

No. 3 hole was missing, and on searching the footwall of the stope I found the primer with the detonator and fuse intact, the barrel of the hole not having been damaged at all. This proves that a certain amount of compression in the rock had taken place, which was sufficient to force the primer and tamping out of the hole, much in the same way as paste is squeezed out of a tube.

Another point I should like to emphasise is that I would strongly advocate the more frequent use of No. 8 detonators to prevent the incomplete detonation of insensitive explosives. In my experience I have found that the extra cost of No. 8 detonators was in many instances more than compensated for by the increased efficiency got by the better detonation of the charge.

Bad charging.—A periodical examination by shift bosses of all charging sticks used is recommended, so as to prevent the use of sticks which have been worn too thin at the end, as air gaps in a charge are easily made by using a stick which is too pointed.

In conclusion, I might say that I think the valuable information contained in the paper warrants it being printed in pamphlet form and distributed among the mines on the Reef.

Mr. W. Waters *(Associate):* Remarks have been made this evening re the all round use of No. 8 detonators, the interference being that a No. 8 detonator will prove more efficacious than a No. 6 in reducing the possibility of unexploded cartridges being found in sockets of holes after blasting. The question is often asked whether a primer of lig-dyn. or "straight dynamite", with either a No. 6 or No. 8 detonator, when used in conjunction with blasting gelatine and gelignite, would not prove even more effective in the desired direction than a No. 8 detonator alone. As I understand that recent experiments have been made at the Geldenhuis Deep Mine with lig-dyn. primers, I should like to ask Mr. Pam, as General Manager of that mine, whether he can throw any light on this subject. The point is important, and I think it would be wise to have it recorded in this discussion.

Mr. E. Pam *(Vice-President):* I cannot say I can give you any definite information on this point, but I may mention that, some four or five months ago, Mr. Quinan asked several of the managers as well as other mining men, to come out to Modder B., where he demonstrated, or attempted to demonstrate, the advantages of using one of these dynamite primers. The first experiment which he did was an attempt to prove the increased detonating power of lig-dynamite through an air gap. He placed an ordinary primer cartridge of gelatine with the detonator and fuse on a board hung on a wire; at a distance from this he placed another cartridge of gelatine, and continued experimenting, bringing the one cartridge gradually nearer to the other until the explosion of the first was successful in exploding the second cartridge. I have no notes on this subject, but, from memory, I think this actually took place when the cartridges were about 3 in. apart. He then put in front of the first cartridge another cartridge of lig-dynamite as a primer, and experimented again. Again speaking from memory, I think the distance at which this cartridge exploded the second one was something in the neighbourhood of 15 in. I would like to make it quite clear that I am speaking from memory. This was a very successful experiment; but did not prove much, because the conditions underground would be quite different. Mr. Quinan then proceeded to demonstrate by digging a trench in the slimes dam,—the trench was just sufficiently wide to put in the ordinary cartridges of gelatine which he placed in line over a length, I think, of about 10 ft., using an ordinary gelatine cartridge as primer. The second experiment was a similar trench, excepting that he used a primer of lig-dynamite. After blasting the first time Mr. Quinan expected that the excavation at the primer end would go considerably greater than that at the far end of the hole. On the other hand, when he used the special primer his contention was that he would get a parallel trench. Unfortunately, in practice the trenches were similar. We were told, however, that several other experiments had actually proved this particular point. We have since tried some experiments underground, but so far there is no positive information as to the advantage of this primer. The difficulty of proving anything is obvious, because in 9,999 cases out of 10,000 the ordinary primer answers its purpose perfectly well. I am afraid it could only be by laboratory experiments that one could prove or disprove the advantages claimed, and if Mr. Cullen will make some experiments and can assist us, I am sure we will all be very interested.

The meeting then terminated.

PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M.

(Printed in Journal, December, 1914-January, 1915.)

Discussion before the Institution of Mining and Metallurgy, London.
Bulletin Nos. 127 and 128 (abridged).

Mr. J. Douglas Hay: The President on 17th December said, 'The falling off in grade when the development of a mine proceeded from the one zone into the other was too much a matter of common experience to be a subject for useful discussion.'' Can this point be ignored when the converse is known to have occurred, where a mine has passed through a rich zone and another is discovered lower down, and yet a still richer one at a greater depth, after having passed through comparatively barren ground in the interval? The theory of deposition by ascending thermal waters is so vague, that for all practical purposes it can be ignored, as it is impossible even to speculate as to the conditions of temperature and pressure during deposition.

In all quartz mines the values are irregular, and vary in the different levels. It would be most unsafe to base one's estimates of values of any one mine on what has occurred at any other mine, even on the same line of reef. That all mines have certain pay shoots, and that these pay shoots do not continue down indefinitely, is universally recognized and admitted. The question is, what are the chances of striking payable ore, at horizons lower than that at which the mine happens to be working? The only means of ascertaining this is by further sinking. No amount of theorizing will ever tell us more. When there are sufficient examples to follow, more can be said on the subject.

Mr. Henry C. Jenkins: 'Persistence of Ore in Depth' appears to be rather a question of geology than of mining profits, and hence I confine my remarks strictly to the question as to whether there is yet to be seen some established variation in the assay of the metal-yielding constituents of the rocks that are mined for them, and whether that variation can be directly associated with the depth from the surface from which they have been obtained, and with no other factor.

A discussion so limited would at once discuss all questions of the profitableness of the operations—and must eliminate from consideration any general conclusion based on the closure of mines with increasing age or depth, and only admit such cases as evidence one way or the other, where it can be shown that adequate prospection at different levels in one particular ore had established some rule for that mine, locality or class of occurrence.

The one fact dependent upon depth that is brought out by a careful study of the assay values of mineral taken from different depths in any one mine is, that there is the often considerable disturbance of value which we term 'surface concentration' usually showing itself as an enrichment of some minerals near to the surface, and a manifest leaching with subsequent precipitation of other minerals below the leached zone. This is so well known now and has been so closely studied that we all agree about it. But beyond this fact, and eliminating its influence and taking such exact data that we have yet collected, we surely cannot assert yet that within the reach of the appliances and methods of today the metallic values decrease with the depth as such, when we have such a notable and clear case as the St. John del Rey to the contrary—or where we have so much evidence that each side claims as its own, without clear-cut evidence to the contrary.

As soon as we leave our practical data and step into deduction we are at present on dangerous ground, even if we are prepared to admit that some minerals show a tendency to pass downwards from the surface, and that others give us every reason why they should be considered to be derived from great depths; so that although it is true that many mines have found their graves 'in the deep,' and so many have been successful only in the zones of surface enrichment (or of secondary precipitation, which is after all but a rather deeper phase of similar phenomena), yet for all practical purposes we can only be sure of what we can actually see as regards depth.

Mr C. F. Heathcote: The deposition of primary ore (ore unchanged by surface conditions) depends on varying chemical action under varying physical conditions—the actions and reactions of acids and alkalies under variations of temperature, pressure and time or rate of flow; in every such case, whatever the depth at which deposition takes place, there is a focus where the best conditions prevail, a point or series of points, a line or series of lines. It is the duty of the

mining engineer to fix in each individual case with as close an approximation as possible the situation of such focus; as the mine workings approach it the ore will improve in value, as they retreat from it impoverishment will take place both laterally and vertically. The idea of 'depth' in mining is conceived by starting from the surface and is eventually completed by arrival at the centre of the earth. As this conception in whole or part has nothing whatever to do with the position of the focal centre of any ore deposit it is equally misleading to state 'that as a rule mines get poorer in depth' as to say 'mines improve in depth'. When considering the probable values of ore in a mine, 'depth' should only be applied in relation to depth of focus of deposition. This applies equally to those mines whose values depend on secondary ores (ores changed by surface conditions). Such mines are not under discussion, as they are notably comparatively shallow; but Mr. Rickard quotes some of these as examples of general impoverishment in depth. If the original focus of deposition was at 6,000 to 10,000 ft., as coal was deposited at surface it may be taken as certain that some deposits extend to over 12,000 ft. in vertical depth and values will be as good as any yet found. This is beyond the present economic range of mining—it is an academic discussion that is not of importance. Whether such deposits outcrop at surface is another question; in some exceptional cases where mineralization is widely extended they undoubtedly do so.

The number of mines that have been closed down cannot be used as an argument against persistence in depth, as a large proportion of these mines owed their value to secondary enrichment. From the remainder must be deducted those that failed from bad financial management by having insufficient funds to complete the objective, those wantonly wrecked by technical managers seeking for dividends only during their short-term engagement by gutting the mine and shelving developments; lastly the small low-grade mines must be deducted where increased depth very materially affects the cost per ton, e.g., the maintenance of shafts and airways depends not upon tonnage but on depth, time and situation.

Mr. J. W. Fairweather: In my opinion the word 'lateral' might be used in a more exact way than it is. Mr. Rickard gives instances where the results from 'lateral' exploration have been the means of reviving dying mines, usually indicating that fresh orebodies were discovered at right angles or obliquely to the known orebody, but, when referring to the North Star Mine, he evidently means an exploration longitudinally and not laterally. Why not confine the use of the word to exploration at either side of it?

Mr. J. Egerton Wood: The discussion on this most interesting paper is rendered more or less futile by the confusion in the use of the term 'ore'. The author uses the word with a meaning differing from that adopted by perhaps ninety per cent. of the members of the Institution. In fact using the author's definition, it follows that there can be no 'persistence in depth.' The deeper a mine becomes, if the management has always been as economical as it might have been, the greater the expense of working, and at a certain depth the material being won is no longer profitable, and ceases to be ore in the author's sense.

Review.

"Explosives, Their Manufacture, Properties, Tests and History," by A. Marshall, F.I.C., Chemical Inspector to the Indian Ordnance Department. (Published by J. & A. Churchill, London.)

This handsome and compendious work (624 pp., price 24s.) is well timed. The subject has grown rapidly since O. Guttman's time, and at present is of course of fundamental importance: it would, for example, do our journalists a heap of good to read the chapters on "choice of a high-explosive" and on shell-filling, and so perhaps save us from some of their sensational statements about Turpinite and TNT.

Most of the existing works on explosives are weak on the chemical side, a fault from which the present volume is refreshingly free. The illustrations are a feature copious in number, and unsurpassed in clearness on account of the use of heavy paper: it is a minor defect that the book is consequently too heavy for its size, though the author himself calls it "moderate".

The books covers the whole field (except minor details of the manufacture and use of explosives) and is very complete as regards analyses and other statistical matter from all parts of the world. It begins with

Contributions and Correspondence.

SAND PULP TABLE.

By W. A. Caldecott (Past President).

1 fluid ton = 32 cub. ft. = 2,000 lb. of water. Sp. Gr. of dry sand = 2·7.

Moisture in Pulp: Ratio by weight of water to solids (W)	Moisture in Pulp: Percentage by weight (P)	Specific gravity of pulp (S)	Tons weight of pulp per ton of dry sand	Volume in cubic feet of pulp per ton of dry sand	Difference (in fluid tons of water per ton of dry sand) between pulp containing "P" moisture and pulp containing 30 moisture	Difference (in fluid tons of water per ton of dry sand) between pulp containing "P" moisture and pulp containing 12 moisture	Grade of Launder: Percentage (G)	Grade of Launder: Fall in inches and decimals per foot	Grade of Launder: Degrees (D)	Ratio by weight of water to solids in pulp (W)	Symbols
$W = \frac{12}{G-1}$	$P = \frac{100W}{W+1}$	$\frac{W+1}{W+0.37}$	$\frac{W+1}{100+P}$	$1{\cdot}185(27W+10)$	$\frac{W-0{\cdot}429}{\frac{P}{100-P}+0{\cdot}429}$	$\frac{W-0{\cdot}136}{\frac{P}{100-P}-0{\cdot}136}$	$G = \frac{1}{\frac{1200}{P}-11}$	$\frac{12G}{100}$	$\tan D = \frac{G}{100}$		
$\frac{P}{100-P}$	$\frac{1200}{G-1}$	$\frac{27}{0{\cdot}17P+10}$		$\frac{1{\cdot}185(27P+1000)}{100-P}$							
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10) deg. min	(11)	(12)
0·4	28·6	1·817	1·1	21·65	−0·029	0·264	31·00	3·720	17 13	0·4	W = Ratio by weight of water to solids in pulp (Col. 1).
0·429	30·0	1·788	1·129	25·57	Nil	0·293	29·00	3·480	16 10	0·429	P = Percentage by weight of water in pulp (Col. 2).
0·5	33·3	1·723	1·5	27·85	+0·071	0·364	25·00	3·000	14 2	0·5	G = Percentage grade of launder (Col. 8).
0·6	37·5	1·649	1·6	31·05	0·171	0·464	21·00	2·520	11 52	0·6	D = Grade of launder expressed in degrees (Col. 10).
0·7	41·2	1·588	1·7	34·25	0·271	0·564	18·14	2·177	10 17	0·7	S = Sp. Gr. of pulp (Col. 3).
0·8	44·4	1·538	1·8	37·45	0·371	0·664	16·00	1·920	9 5	0·8	$W = \frac{1-0{\cdot}37S}{S-1}$
0·9	47·4	1·496	1·9	40·65	0·471	0·764	14·33	1·720	8 9	0·9	$P = \frac{27-10S}{0{\cdot}17S}$
1·0	50·0	1·460	2·0	43·85	0·571	0·864	13·00	1·560	7 21	1·0	$G = \frac{11{\cdot}63S-11}{1-0{\cdot}37S}$
1·2	54·5	1·401	2·2	50·25	0·771	1·064	11·00	1·320	6 17	1·2	(See this *Journal*, Vol. XIV., June 1914, p. 487).
1·4	58·3	1·356	2·4	56·65	0·971	1·264	9·57	1·149	5 28	1·4	
1·7	63·0	1·304	2·7	66·25	1·271	1·564	8·06	0·967	4 36	1·7	
2·0	66·7	1·266	3·0	75·85	1·571	1·864	7·00	0·840	4 0	2·0	
2·4	70·6	1·227	3·4	88·65	1·971	2·264	6·00	0·720	3 26	2·4	
3·0	75·0	1·187	4·0	107·85	2·571	2·864	5·00	0·600	2 52	3·0	
4·0	80·0	1·144	5·0	139·85	3·571	3·864	4·00	0·480	2 17	4·0	
5·0	83·3	1·117	6·0	171·85	4·571	4·864	3·40	0·408	1 57	5·0	
6·0	85·7	1·099	7·0	203·85	5·571	5·864	3·00	0·360	1 43	6·0	
7·0	87·5	1·085	8·0	235·85	6·571	6·864	2·71	0·326	1 33	7·0	

40 pp. of vividly written history, followed by a small but sufficient article on blackpowder and its ingredients: then come the acids and denitration, leading to a full (100 pp.) account of glycerine and cellulose and their nitration. Next come smokeless-powders and blasting-explosives of all countries: the remaining sections are on physical properties, detonation and accessories, storage, fireworks, ingredients and analysis, with some good new information on stability and stabilizers. Appendices (thermotables and certain German railway rules) follow.

The following mistakes and mis-spellings have caught the writer's eye: p. 266, *eramine* (?bromine); p. 129, empyrical (?); p. 504, acet-*o*-toluidide is $CH_3{\cdot}C_6H_4{\cdot}NH{\cdot}CO{\cdot}CH_3$, whereas the formula given by the author is that of tolylacetamide; stablity (356), pyroxylene (127), silicoons (280), fairy! (175), and rammose (176) are other mis-spellings: in the latter case, the substance is classed both as pentose and hexose, which is somewhat miraculous.

Another disfigurement of this excellent book is an occasional Germanism, e.g., *saccharinic* and *carbaminic* acids (pp. 171 and 504) and *furfuran* (p. 266), the correct English being saccharic, carbamic and furane. In the analytical part two curious errors occur: (1) "a pale solution of $AmNO_3$" is mentioned as a reagent for silver (?) on page 530, and on p. 533 the substance NaN_3 is stated to have "an alkaline reaction and a very acid taste" (?acrid).

Nitric acid from air gets only eight lines from the author, despite its importance to our enemy, an enemy who could never, for lack of genius, have invented the process, but has evidently plodded it along to a commercial success.

The writer would like to know why the iodic-anhydride method of CO analysis is ascribed to Weiskopf (p. 328), when it was really invented in France about 1898, and was used by the writer here in 1902.

Another thing that strikes one is that it is rather extraordinary that only the worst kinds of cellulose are still used for nitration, giving great trouble in purification and never-ending worries about stability afterwards both to the factories and to the Inspection Departments of all Governments —which is surely a false economy in the long run.

J. Moir.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Auto-Reduction in the Precipitation of Metallic Gold.—" In the various studies which have been made on the precipitation of gold from solution, attention has been directed for the most part to the action of various reducing agents on gold solutions. The general chemical inactivity of gold enables us to deposit the metal in elementary form by most of the metals, the metallic sulphides, ferrous salts, various organic compounds, and, in fact, in the ordinary sense of the word, gold compounds are readily reduced to metal by the mildest of reducing agents. On the other hand, very little attention has been directed to the precipitation of gold in the metallic condition by oxidising agents. A few isolated cases of the auto-reduction of gold from its compounds by certain oxidising agents have been recorded, but the general principle of the auto-reduction of gold compounds has not received systematic study from the standpoint of the deposition of gold.

It has been known for a long time that hydrogen peroxide, when brought in contact with oxide of gold or oxide of silver, will reduce either of these metallic oxides to metal: water and oxygen being formed simultaneously.

$$Au_2O_3 + 3H_2O_2 = 2Au + 3H_2O + 3O_2.$$
$$Ag_2O + H_2O_2 = 2Ag + H_2O + O_2.$$

In a similar manner, gold solutions in contact with hydrogen peroxide yield metallic gold, oxygen being evolved at the same time. This reaction of hydrogen peroxide with gold compound takes place in either acid or alkaline solution. Sodium peroxide and sodium perborate precipitate metallic gold immediately from gold solutions. Barium peroxide and calcium peroxide act toward gold solutions in exactly the same way, precipitating metallic gold at once. These compounds are perhaps closely related to hydrogen peroxide and, as a consequence, act similarly toward gold solutions. Osmium tetroxide, or the so-called osmic acid (OsO_4), when dissolved in water will not reduce gold solutions, but when the free acid is neutralised, or when the solution is made alkaline with sodium hydroxide, sodium carbonate, or calcium carbonate, metallic gold is precipitated. The higher oxides of nickel and cobalt, prepared by the action of an alkaline hypobromite on solutions of nickel and cobalt chloride, when brought in contact with a solution of gold which has been rendered alkaline, precipitate metallic gold.

Lead peroxide, as well as red lead, precipitates metallic gold from either neutral or alkaline gold solutions. Ceric oxide, prepared in the hydrated form by rendering ceric chloride alkaline, precipitates metallic gold at once from a gold solution which has been rendered alkaline. The compounds of manganese present interesting deportment along this same general line of auto reduction. Manganese dioxide, prepared by the action of bromine on a manganese acetate solution, precipitates gold from its solution either under slightly acid, neutral, or alkaline conditions. Potassium permanganate, on being allowed to stand for some time with auric chloride, causes metallic gold to be precipitated along with manganese dioxide. The minerals pyrolusite, wad, braunite, and manganite, when brought in contact with a gold solution, which is either acid, alkaline, or neutral, slowly cause gold to deposit.

Recently Brokaw has shown that manganese salts, when made alkaline, or even manganese carbonate

will precipitate gold from solution. The direction of the reaction between manganous salts and gold chloride is considered by Brokaw to be largely due to hydrolysis of gold chloride solutions, although the equations, as given, show that gold chloride and manganous salts react in neutral solution to form manganese dioxide and metallic gold. All of the reactions recorded by Brokaw have been studied and corroborated. That manganese can be of considerable importance in the deposition of gold is unquestionably true, and its function may be that which Brokaw has suggested; but, in addition, manganese may play the role of a reducing agent, acting solely through auto-reduction.

Auric compounds, as is well known, are easily reduced to metallic gold, and, hence, can be considered as oxidising agents and can, obviously, oxidise manganous hydroxide to manganese dioxide. It can therefore be considered that the mutual precipitation of gold and manganese dioxide by calcite, as cited by Brokaw, is simply the neutralisation of the excess of acid, whereupon the oxidation of the manganese by the gold solutions results in the precipitation of both manganese dioxide and metallic gold.

That this is the direction of the reaction is evidenced by the duplication of the phenomenon by replacing manganese salts with cerium compounds. Cerous hydroxide precipitates metallic gold in alkaline solution; the cerium being at the same time oxidised to cerium dioxide. Similarly, when a piece of calcite is introduced into a solution of cerium chloride and gold chloride, as the free acid is neutralised by the calcium carbonate a deposit of cerium dioxide containing metallic gold begins to form on the calcite.

The presence of gold in the manganese deposits, which has been studied by Emmons, may be due to the fact that manganese is a significant agent in the superficial transportation of gold; but such is by no means necessarily the case. Two distinctly different kinds of reactions may be going on. In one case a chloride solution, containing free acid, on coming in contact with an oxidised manganese deposit will produce free chlorine, or its equivalent. This chlorine solution can then dissolve gold and cause the production of a gold bearing manganese solution, which, so long as it contains considerable free acid, is quite permanent; but, when it comes in contact with any neutralising agency, for example, limestone, metallic gold and manganese dioxide would be at once precipitated. This accords with experiments which have been reproduced in the laboratory, and which do not, as indicated by Emmons, require such a reducing agent as ferrous sulphate or other reducing agents as indicated by Wells, to produce a precipitate. On the other hand, the gold present in the manganese deposits adjacent to other gold deposits may be due wholly to the fact that gold solutions in contact with manganese dioxide yield metallic gold, by virtue of the principle of auto-reduction shown by other peroxides with gold solutions. The auto reduction of gold solutions to metallic gold has apparently, thus far, not been considered as an important geological factor in the secondary deposition of gold, but it is doubtless possible that in many instances one can conceive of the oxygen of the air as being the real agent which causes the gold to be reduced in presence of a manganese or similar compound which acts as a catalytic agent." VICTOR LENHER, *Journal of the American Chemical Society. Mining and Scientific Press*, Sept. 12, 1914, p. 411. (G. H. S.)

THE DETERMINATION OF IRIDIUM IN PLATINUM-IRIDIUM ALLOYS.—"In the presence of small quantities of Rh the tendency of Ag cupellation beads to spit or vegetate is greatly increased. Distinct surface crystallization of the Ag bead is also produced. Rh in alloy with Pt may thus be detected. Neither Pt or Ir produce these indications. Ag cupellation beads containing 0·3—1·6 % Pt, when examined under a low-power microscope, show a peculiar, corrugated structure. A similar appearance is caused by the presence of Pd. In the presence of Ir, Ag beads assume a more spherical form than is usual with pure Ag, and the surface is covered with lines or bands, which cross one another. The presence of Pt masks the effect of Ir, and for its detection in Pt-Ir alloys it is necessary, after first cupelling 0·05 gm. of the alloy with 2 gm. Ag, to part the bead in H_2SO_4, to separate most of the Pt by means of *aqua regia*, and to recupel the residue with another 2 gm. Ag. Ir may then be detected under a low-power objective. The relative solubilities of Pt and Ir vary with the composition of the alloys, so that simple digestion in *aqua regia* will not accomplish the separation, but can only be applied after separating the Pt and Ir into distinct particles. Two methods are described: Cupel 0·25 gm. of alloy with 1·5 gm. pure Ag and 10 gm. assay Pb. Finish the cupellation at a high temperature to expel the last traces of Pb. Clean, hammer, anneal and roll the button. Part in assay flasks with 40 cc. concentrated H_2SO_4. Decant the Ag_2SO_4 solution and boil the residue again with 40 cc. concentrated H_2SO_4. Decant and wash the residue twice with water and transfer to an annealing cup. Dry the Pt-Ir residue, anneal at a high temperature and weigh. An increase in weight will be observed, owing to the presence of small amounts of Ag. Digest the Pt-Ir residue in 10 cc. dilute *aqua regia* (HCl 4 volume, HNO_3 1 volume, H_2O 10 volumes) until action ceases, then evaporate to half the original volume. Decant the solution and again digest the residue with 5 cc. dilute *aqua regia*. Combine the solutions, dilute to 25 cc., boil and filter. Wash the residue, which consists of Ir and some AgCl, with dilute NH_4OH, to dissolve the latter. Then wash with hot water, dry, ignite at a high temperature and weigh. Evaporate the *aqua regia* solution nearly to dryness to expel HNO_3, add a few drops HCl, and dilute to 50 cc. Precipitate the Pt with pure Zn (free from Pb, etc.), allow to stand until the Zn has dissolved and the Pt precipitated, filter off the Pt, wash with dilute HNO and hot water, dry, ignite at a high temperature, heat for 10 minutes in atmosphere of H, cool and weigh. The second method described was published by George Matthey (*Proc. Roy. Soc. London*, *28*, 463 (1879)). The following results were obtained by the first method: In 20% Ir alloy 20·20 or 20·40 Ir was found and 79·76% or 79·52% Pt. In 25 % Ir alloy 25·20 or 25·24 Ir and 74·68% or 74·56 Pt. With the second method were found in 20 % Ir alloy: 20·60 or 20·40 Ir and 79·30% or 79·60 Pt. In 25 % Ir alloy, 25·32% or 25·28% Ir and 74·52 or 74·60 Pt."—C. O. BANNISTER and E. A. DEVEREER, *Analyst*, *39*, 340.6.—*Chemical Abstracts*, Oct. 20, 1914, p. 3403. (J. C.)

A NEW METHOD FOR THE DETERMINATION OF ZINC IN ALLOYS. "The Zn is precipitated according to the reaction $ZnSO_4 \quad Hg(CNS)_2 2KCNS = ZnHg(CNS)_4 + K_2SO_4$. To prepare the precipitating reagent dissolve 39 gm. KCNS in 200 cc. H_2O and stir in 27 gm. HgCl, whole diluting slowly to 1 litre. Filter if not clear. *Method (1)*: For accurate

determination prepare solution as for pyrophosphate precipitation. Not more than 5 free acid should be present. Nitrous acid should be absent. To the prepared solution add slowly with brisk stirring 25 cc. to 30 cc. of reagent for each 100 cc. solution, set aside for 5—30 minutes, stirring vigorously at intervals three or four times. A satisfactory bulk is 250 cc. When cold, filter through a gooch, wash with H_2O containing 20 cc. of the reagent per litre, dry at 100·8 and weigh. The precipitate contains 12·66 % Zn by weight. If preferred, titrate with $AgNO_3$ and KCNS. *Method (II.):* For a less accurate determination in brass, etc., dissolve 0·5 gm. alloy in the least possible quantity HNO_3, add 10 cc. concentrated H_2SO_4, heat to fumes, cool, dilute, precipitate the Cu with Al, and precipitate the Zn as before. *Method (III.):* For less accurate work with Al alloys free from Cu, dissolve 0·5 gm. in 10 cc. concentrated HCl, filter if necessary, and precipitate as in *(I.)*. Non-interfering or very slightly interfering substances are: salts of Pb, Hg^{++}, Asv, Sbiii, Sniv, Fe, Al, Cr, Ca, Ba, Sr, Mg and small amounts of Ni; and KCl, KNO_3, KCNS, NaCl, $NaNO_3$, Na_2SO_4, AcONa, NH_4Cl, NH_4NO_3, $(NH_4)_2SO_4$: AcOH, HCl, HNO_3, H_2SO_4, if not over 10 %. Interfering substances are: Hg^{++}, Cu, Bi, Cd, Asiii, Co, Mn and large amounts of Ni which cause high values. K_2SO_4, NH_4OAc, NH_4CNS and more than 10 % acid cause low results. A series of results appended shows good agreement with the pyrophosphate method."—G. E. F. Lundell and Nat K. Bee, *Trans. Amer. Inst. Metals.—Chemical Abstracts*, Oct. 20, 1914, p. 3404. (J. G.)

A Chemically Resistant Glass Lacquer.—"Fifteen gm. of pure, firm unvulcanized rubber are treated with 100 cc. of carbon bisulphide, and after a few hours 30 gm. of white ceresin, cut into small pieces, and 100 cc. of carbon bisulphide, are added to the homogeneous mixture. The resulting lacquer should be less viscid than glycerin. The cleaned glass vessel is rinsed with absolute alcohol, drained, one-third filled with lacquer, shaken gently round, and immediately drained. A smooth, translucent coat is thus obtained, especially if the process is modified by applying two layers of a somewhat thinner lacquer with an intermediate interval of 4 to 5 hours. Carbon tetrachloride may be substituted for carbon bisulphide if the lacquer is made thicker and applied hot. The protective coating is very effective with careful usage; a bottle so treated was filled with distilled water of conductivity 2·45 reciprocal megohms and rotated for 70 hours in a thermostat at 20° C. whereby the value rose only to 2·98, whilst a similar unlacquered bottle gave the value 58·0 under identical conditions.—P. J. Fox.—*Journ. Soc. Chem. Ind.*, Dec. 31, 1914, p. 1215. (J. A. W.)

Lygosine: A New Indicator.—Two mols. of salicylic aldehyde and 1 mol. of acetone are condensed in alcoholic solution by means of strong caustic soda. The substance which separates is recrystallized from alcohol and treated with acid, when *o*-dihydroxy-dibenzalacetone or lygosine is liberated. A 1% alcoholic solution of this compound serves as an indicator. A few drops added to an acid liquid gives a faint opalescence, and with an alkaline liquid an orange-red colour. The indicator is as good as phenolphthalein, even with ammonia.—A. Ferenez.—*Journ. Soc. Chem. Ind.*, Nov. 16, 1914, p. 1075. (J. A. W.)

METALLURGY.

Sulpho-Cyanides in Cyaniding.—"The action of these compounds in cyanide work, judging from the little discussion they have received in the technical papers, has been but slightly understood by those engaged in ore treatment. The importance of this action was not realized by myself until about three years ago, when in attempting to work out a regeneration process, now nearly perfected, I was obliged through encountering many obstacles to realize that sulpho-cyanides were of more importance than text-books and other publications had led me to believe.

Sulpho-cyanides are to be found in all cyanide solutions which have been in contact with ores containing sulphur, or compounds of that element. Cyanide has but little effect on sulphur in the cold, but when sulphur is treated with warm cyanide solutions of appreciable strength, a reaction occurs between the two which results in the formation of a sulpho-cyanide according to the following equation:

$$KCN + S = KCNS$$

Sulpho-cyanides are rarely formed during cyaniding by means of the direct action of cyanide on free sulphur, but they are readily formed through the action of cyanide on thiosulphates resulting from the action of caustic alkalis on sulphur, as is shown by the following:

$$6NaOH + 4S = Na_2S_2O_3 + Na_2S + 3H_2O$$
$$Na_2S_2O_3 + KCN = Na_2SO_3 + KCNS$$

Sulpho-cyanides are also formed through the action of cyanide on electro-negative or soluble sulphur, which results from the decomposition of sulphur compounds like Na_2S.

Alkaline sulphides do not react directly with cyanide to form sulpho cyanide. They must first be broken down to soluble sulphur before their sulphur contents will react with cyanide to form sulpho-cyanides, and as solutions usually contain zinc, and possibly traces of lead, it is doubtful whether they would have a chance to break down into soluble sulphur, and then combine with the cyanide to form sulpho-cyanides, before they would combine with the metals to form insoluble sulphides. Therefore they do not cause the formation of sulpho-cyanides in cyanide solutions to any considerable extent.

Lead acetate does not prevent the formation of sulpho-cyanides to any greater degree than that due to its combining with alkaline sulphides, before the alkaline sulphides have had a chance to break down to soluble sulphur. A number of tests which I made on Goldfield Consolidated ore show that the amount of sulpho cyanide formed when lead acetate was added at the rate of 0·25 lb. per ton of ore, was only 2·100 lb. greater than that which was formed when 1 lb. of acetate per ton of ore was used in otherwise parallel tests.

Lead acetate does not precipitate sulpho-cyanide from solutions which have sufficient protective alkalinity, but it precipitates the sulpho-cyanide as lead sulpho-cyanide, $Pb(CNS)_2$, from solutions which have no protective alkalinity. The following test shows the action of lead acetate on sulpho-cyanides:

Test No. 1.	KCN lb.	Pro. alk., CaO, lb.	KCNS, lb.
A—Solution before adding acetate	2·38	0·44	0·984
B—Solution after adding 1½ lb. acetate per ton...	2·26	0·30	0·984
Test No. 2.			
C—Solution before adding acetate	1·90	0·05	0·699

Solution after adding			
1½ lb. acetate per ton	0·64	none	0·646

Sulpho-cyanides of alkalies, when treated with acids, form hydro-sulpho-cyanic acid, according to the following equation :

$$2KCNS + H_2SO_4 = 2HCNS + K_2SO_4$$

Sulpho-cyanic acid is non volatile in the cold, in which one respect it differs from hydro-cyanic acid. It has a greater affinity for metals like silver and copper than has hydro-cyanic acid. This is proved by the fact that it will replace the cyanogen contained in silver cyanide according to the following equation, which is based on the results of actual experiment :

$$AgCN + HCNS = AgCNS + HCN$$

Its affinity for silver is even greater than that of chlorine, as I have proved by tests that silver-sulpho-cyanide will precipitate in preference to silver chloride from a dilute mixture of sulpho-cyanic acid and hydrochloric acid, when silver nitrate is added to such a mixture. Its affinity for gold is great, as is proved by the fact that gold is not completely precipitated from acidified solutions containing HCNS by means of reducing agents, other than those which result in the formation of nascent hydrogen even when after all of the HCN has been removed from the solution, until all of the HCNS has been precipitated from the solution.

Oxygen in the molecular form has no appreciable oxidizing effect on sulpho cyanides, whether they exist in acid or alkaline solutions. Ozone in a dilute form, and nascent oxygen, will oxidize sulpho-cyanides to a corresponding cyanide, according to the following equation :

$$HCNS + O_2 + H_2O = HCN + HSO_4$$

Powerful oxidizers such as strong nitric acid, hydrogen peroxide, undiluted ozone, and others, have a tendency completely to convert the cyanogen contained in the sulpho cyanide to ammonia. Electrolysis converts sulpho cyanides to a yellow compound. Strong, or moderately strong acids, such as sulphuric even in a dilute state, will convert sulpho cyanides to ammonia compounds and carbon oxysulphide, as is shown by the following equation :

$$KCNS + 2H_2SO_4 + H_2O = KHSO_4 + NH_4HSO_4 + COS$$

Sulpho cyanides have but little or no solvent action on metals like silver and copper, according to tests which I have made ; but when an ore is treated with a sulpho cyanide solution in the presence of lime, by means of air agitation, some silver extraction will be observed. Silver sulpho cyanide is somewhat soluble in alkaline sulpho-cyanides, and it is quite possible that silver chloride in the ore would be dissolved by alkaline sulpho cyanides according to the reaction :

$$AgCl + 2KCNS = AgCNS.KCNS + KCl$$

In some cases I have noted that free sulpho cyanide had no harmful effects on the extraction power of a cyanide solution. I have not noticed any reverse to this. It is quite probable that some bad effects on the extraction would be noticed in cases where solution were loaded with free sulpho cyanide, but this would be the effect of the inactive salts.

Judging from the evidence shown by the foregoing reactions and facts, it appears that free sulpho-cyanide is not a reducing agent in the sense as reducing agents are classified in the art of cyaniding. I have found that copper from sulphide ores invariably goes into solution as copprous sulpho cyanide dissolved in cyanide, as is expressed in the following final equations :

$$2Cu_2S + 8KCN + 2H_2O = Cu_2(CNS)_2.6KCN + 2KOH + H_2$$

$$Cu_2S + 8KCN + S + 2H_2O = Cu_2(CNS)_2.6KCN + 2KOH + H_2$$

According to these, 1·52 lb. CNS in terms of KCNS would be formed, and 4·09 lb. KCN would be consumed per pound of copper dissolved. To prove the value of these equations, a test was made with 1 gm. of $Cu_2(CNS)_2$ and cyanide. In this the titrations with silver nitrate and potassium iodide indicator showed a consumption of 1·63 gm. of KCN. This agrees closely with the above equations, which would indicate a consumption of 1·606 gm. of KCN per gm. of $Cu_2(CNS)_2$. Further proof of the value of these equations is shown by results of tests made on sulphide ore from the Headlight mine in Trinity County, California. In this test titrations indicated a consumption of 65·16 lb. KCN per ton of ore with the solution of 13·494 lb. Cu, the formation of 21·18 lb. CNS in terms of KCNS, and 9·31 lb. $K_4Fe(CN)_6$, which is equivalent to 9·96 lb. KCN. The cyanide consumed per pound of copper, after allowing for the cyanide equivalent of the $K_4Fe(CN)_6$, was 4·4 lb. KCN while the sulpho-cyanide was formed at the rate of 1·57 lb. KCNS per pound of copper.

These equations, representing the form in which copper exists in solutions, differ greatly from those given by Will H. Coghill in the *Press* of August 17, 1912. Mr. Coghill mentions a compound having the formula $Cu(CNS)_2$. I may mention that I have failed to find any evidence of its existence in cyanide solutions.

Silver usually enters the solutions in the form of a sulpho-cyanide dissolved in cyanide, excepting in the case of oxidized ores. The silver is usually in the form of $AgCNS.KCN$. It is because both silver and copper exist in many solutions as sulpho-cyanides, that a number of operators have been misled about the regeneration of cyanide due to zinc precipitation. Some of these operators have been accustomed, in conducting tests, to allow for a regeneration of cyanide in zinc precipitation from the compound of silver existing in the solution which they presumed to be $KAg(CN)_2$. The reactions upon which they based their regeneration figures may be expressed as follows :

$$KAg(CN)_2 + Zn + 2KCN + H_2O = K_2Zn(CN)_4 + Ag + H + KOH$$

$$K_2Zn(CN)_4 + 4KOH + 4KCN + Zn(OK)_2 + 2H_2O$$

According to these, 1·207 lb. of KCN would be regenerated by zinc precipitation, from each pound of silver precipitated. The regeneration of cyanide as shown by the second equation only occurs when solutions have a high protective alkalinity. My experience has been that the regeneration usually occurs when solutions have over 1 lb. CaO protective alkalinity. That the regeneration as shown above is wrong in cases of sulphide ores is clearly proved by the fact that the silver exists in solution not as $KAg(CN)_2$, but as $AgCNS.KCN$. The regeneration in zinc precipitation from such a compound may be expressed by the equation :

$$AgCNS.KCN + 3KCN + Zn + H_2O = K_2Zn(CN)_4 + Ag + H + KCNS + KOH$$

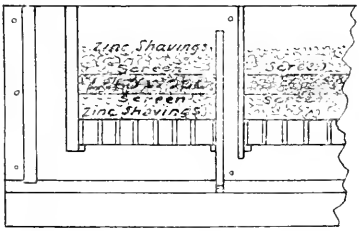
$$K_2Zn(CN)_4 + 4KOH + 4KCN + Zn(OK)_2 + 2H_2O$$

Here the regeneration of cyanide in high alkaline solution would only amount to 0·603 lb. KCN per pound of silver precipitated. While in the case of low alkaline solutions a higher consumption of cyanide would take place according to the above reaction, when precipitating the solutions resulting from the treatment of sulphide ores, than would occur when precipitating solutions resulting from the treatment of oxidized ores.

I have not as yet found any evidence of the existence of a compound of zinc and sulpho-cyanide; therefore, I have presumed that zinc does not exist in solutions as a sulpho-cyanide of zinc dissolved in cyanide.

That free alkaline sulpho-cyanides are regenerated during zinc precipitation is shown by the following typical test which I made at the Belmont mill at Tonopah. In this test the solution entering the precipitation plant for 24 hours contained 2.1692 lb. free KCNS and 2.66 oz. Ag, while the solution leaving the plant during the same period contained 2.47 lb. KCNS and 0.02 oz. Ag. Here 0.3098 lb. KCNS was regenerated. Of this amount, 0.164 lb. was due to the silver precipitated, while the remainder was due to the copper precipitated. Analyses of a head solution taken from the Belmont mill, at Millers, shows 4.24 lb. KCNS (free) and 1.62 lb. KCNS (terms) of combined CNS. In the case of a test on Jim Butler ore, with fresh solution 1.98 lb. combined KCNS (terms) were formed. The 0.24 lb. of copper dissolved would account for 0.3648 lb. KCNS, and the 25.2 oz. Ag or 1.727 lb. Ag dissolved, would account for 1.554 lb. KCNS; making a total of 1.918 lb. KCNS, against 1.98 lb. KCNS found by titration, only a trace of which was free KCNS."—H. R. LAYNG. *Mining and Scientific Press*, Sept. 26, 1914, p. 481. (G. H. S.)

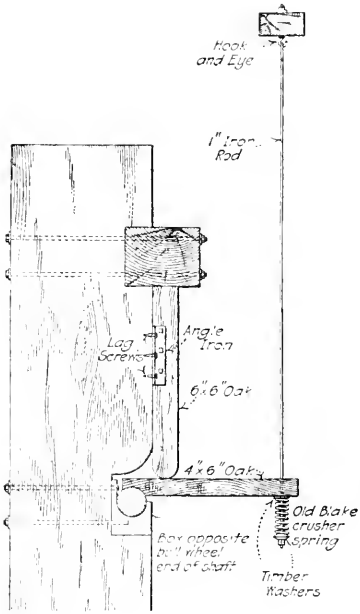
LEAD WOOL IN CYANIDING. "A number of years ago, while in New York City, I had my attention attracted to the lead wool used in calking gas mains, and at once thought of using it in conjunction with zinc as a precipitating agent in the cyanide process. Accordingly, I took some 10 lb. of it with me to Nicaragua and used the lead wool as an intermediate stratum in the head compartment of one of our zinc-boxes. The fibres, being just like zinc shaving but less elastic, lend themselves easily to



Section Showing Method of Placing Shavings.

the process. As the precipitate, for the month I used this lead-wool zinc shaving couple, was decidedly the higher grade (36.1% bullion against a 25% bullion precipitate), I thought it might interest some one else to give this material a more thorough test than I could give it. I realize there is nothing new in the use of lead or its salts, but I have never seen this lead-wool, which is analogous to zinc shavings, used, and I wish to offer this as a suggestion for some one who has the opportunity to make a conclusive test."—ARTHUR FEUST. *Mining Press*, Jan. 9, 1915, p. 79. (G. H. S.)

CAM-SHAFT DAMPER. "The accompanying sketch shows a simple contrivance for reducing cam-shaft [illegible] the Iola [illegible] in [illegible] years of continuous operation.



CAM-SHAFT DAMPER.

The installation of this damper has materially lengthened the life of the camshaft, has eliminated most of the wear of the box, and makes a noticeable difference in the smoothness of operation of the entire batter. The sketch needs little explanation. Two short pieces of [illegible], the iron rod, and any suitable spring is all that is required. The [illegible] piece must be fastened in some [illegible] to the front of the battery post, while the other piece is held in place by being fitted to the camshaft. The pressure is easily regulated by the nut at the bottom end of the rod, and by taking off this nut the entire device can be removed and gotten out of the way. A roof truss member, or some other substantial support, is required for the upper end of the rod."—A. B. PARSONS. *Engineering and Mining Journal*, Oct. 21, 1914, p. 743. (G. H. S.)

MINING.

A New Safety Detonating Fuse.—This new detonator is a French invention, known as *Cordeau detonant* or detonating fuse, and is sold under the name of Cordeau Bickford. It consists of a lead tube 5 to 6 mm. in diameter, filled with trinitro toluene.

In addition to greatly increasing the safety of blasting, the detonating fuse has the additional merit of increasing the efficiency of the explosive charge.

To summarize, we may state that this fuse has three very important qualities, viz.: 1. It is safe. 2. It is instantaneous. 3. It increases the efficiency of an explosive charge.

1. *Safety.* There is no danger in the handling or storage of the fuse. It cannot be exploded by friction, fire, or ordinary shock. It requires the use of a strong blasting cap properly attached to explode it. In blasting charged holes, the cap or exploder can be applied outside the hole, thus avoiding the danger of burning power caused by side spit from ordinary fuse; also any risk of accident while tamping and the risk from a portion of an unexploded charge accompanied by a cap remaining in the debris from a blast is entirely obviated.

2. *Speed.* The average rate of speed of this fuse is estimated to be close to 17,000 feet per second, so that when it is used the explosive charge is detonated instantly throughout its entire length, instead of at one point as is the case with the blasting cap or electric exploder.

3. *Efficiency.* It is known that the speed of an explosive decreases as the explosive wave travels away from the detonator. That the powder in a hole has the strongest explosive effect around the exploder is evident from an examination of the face of the bank after a shot. This can be demonstrated also by placing sticks of dynamite on the ground end for end, about 6 in. apart, with the cap in the first stick. The explosive force gradually lessens until it finally ceases to progress, leaving the farthest sticks unexploded.

By using this fuse the charge is detonated instantaneously throughout its entire length. This results in a saving of about 10% of explosives as determined by results obtained at Cornwall and elsewhere. It is not affected by heat, cold, or moisture, and lasts indefinitely without deterioration.

Fig. 1 illustrates the method of attaching the detonator to the fuse. Either ordinary fuse and cap or an electric exploder can be used to set off the detonating fuse.

Fig. 2 shows the method of making connections between main and branch lines. Branch line A is joined to trunk line B by twisting the split ends, one to the right, the other to the left of the intersection. These joints may be protected from danger of loosening or breaking by winding with electric tape. This also protects them from moisture. This precaution is not necessary except where many connections have to be made or wet work is encountered. If the direction of the explosive wave is carefully borne in mind, there can be as many branches and sub branches as are desired.

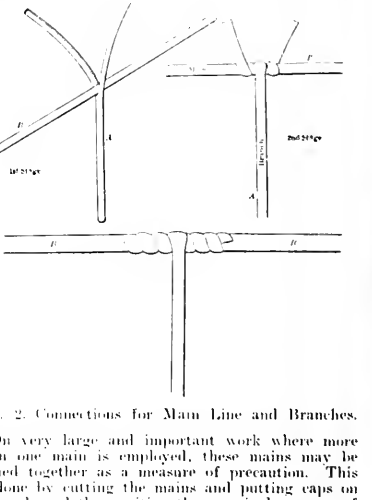


Fig. 2. Connections for Main Line and Branches.

On very large and important work where more than one main is employed, these mains may be joined together as a measure of precaution. This is done by cutting the mains and putting caps on the ends and then uniting them again by means of a brass sleeve, the ends of which are firmly crimped to the detonating fuse. Around these brass sleeves the ends of the connecting line are wound in

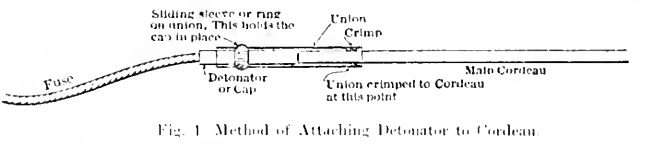


Fig. 1. Method of Attaching Detonator to Cordeau.

exactly the same manner as when attaching a branch to a main line.

In the test at the quarries of the Atlas Portland Cement Co., the rock blasted was a cement limestone shale bench 94 ft. high (perpendicular) with the bottom well cleaned up. There were eleven 5 in. diameter holes, 26 ft. back from the face of the bench and spaced 13 ft. apart, 100 ft. deep and extending 6 ft. below the bottom of the bench. The holes were cleared of water just before loading. The charge was made up of 50% gelatine and 40% low freezing dynamite, with the detonating fuse 6 mm. in diameter, running to the bottom of each hole.

All the holes were connected with a line of detonating fuse to the end of which a single No. 6 detonating cap was attached. This was fired by a 220 volt current from a live wire. Instantly there was a sharp crack as the fuse exploded, followed by a rumble of the crumbling bank. The rock was thrown well out, leaving a clear wall or face.—Harrison Souder, *Eng. Magazine*, Dec., 1914, p. 416. (H. A. W.)

MISCELLANEOUS.

ARTIFICIAL RESPIRATION.—The U.S. Bureau of Mines has been studying various methods of inducing artificial respiration suitable for use in mines when men are overcome by electric shock or gas. A committee consisting of Dr. W. B. Cannon, chairman, Professor of Physiology, Harvard University; Dr. George W. Crile, Professor of Surgery, Western Reserve University, Cleveland, Ohio; Dr. Joseph Erlanger, Professor of Physiology, Washington University, St. Louis; Dr. Yandell Henderson, Professor of Physiology, Yale University; and Dr. S. J. Meltzer, head of the Department of Physiology and Pharmacology, Rockefeller Institute for Medical Research, has reported the following recommendations. As these not only apply to men who are overcome by electric shock or gases in mines, but also to persons suffering from the effects of illuminating-gas poisoning or from electric shock anywhere, they are of importance to many thousands of workmen

In case of gas poisoning, remove the victim at once from the gaseous atmosphere. Carry him quickly to fresh air and immediately give manual artificial respiration. Do not stop to loosen clothing. Every moment of delay is serious.

In case of electric shock, break electric current instantly. Free the patient from the current with a single quick motion, using any dry non-conductor, such as clothing, rope, or board, to move patient or wire. Beware of using any metal or moist material. Meantime have every effort made to shut off current.

Attend instantly to the victim's breathing. If the victim is not breathing, he should be given manual artificial respiration at once. If the patient is breathing slowly and regularly, do not give artificial respiration, but let nature restore breathing unaided.

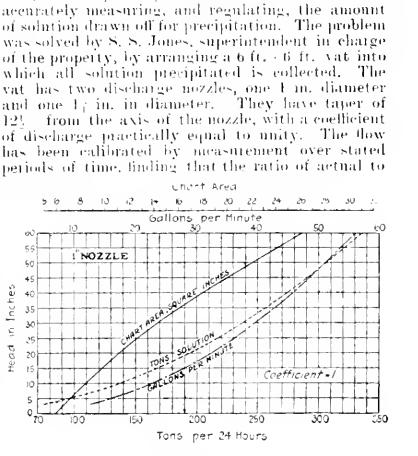
In gas cases, give oxygen. If the patient has been a victim of gas, give him pure oxygen, with manual artificial respiration. The oxygen may be given through a breathing bag from a cylinder having a reducing valve, with connecting tubes and face mask, and with an inspiratory and expiratory valve, of which the latter communicates directly with the atmosphere. No mechanical artificial resuscitating device should be used unless one operated by hand that has no suction effect on the lungs.

Use the Schaefer or prone pressure method of artificial respiration. Begin at once. A moment's delay is serious. Continue the artificial respiration. If necessary, continue two hours or longer without interruption until natural breathing is restored. If natural breathing stops after being restored, use artificial respiration again.

Do not give the patient any liquid until he is fully conscious. Give him fresh air, but keep his body warm. Send for the nearest doctor as soon as accident is discovered.—*Mining and Scientific Press*, Nov., 14, 1914, p. 765. (G. H. S.)

FLEXIBLE WATER-PROOF MATERIAL.—"A new process for the manufacture of flexible and water-proof material, composed of a number of superimposed paper sheets, has recently been patented by Mr. Giovanni Magnasco, of Genoa, Italy. This invention is said to be of special interest to the chemical industry as it provides a very useful packing material for chemicals. Paper sheets are separately coated with a glueing solution comprising a mixture of animal glue, glycerine, albumin, camphor, linseed oil and sulphur, and these are afterwards 'tanned' by means of formaldehyde solution, tannic acid, or the like. The paper used for this process is first coated or impregnated with the glueing mixture, afterwards one side of the sheets is coated with the tanning mixture (composed of formaldehyde and water or chromic acid, etc.), and the necessary number of sheets superposed in order to obtain the desired thickness. Lastly, the whole is compressed. According to the use to which it might be subjected, the material is again immersed in a tanning medium and then allowed to dry. Lastly, before it is put into use, it is coated with paraffin or other fats. Additional strength is added by placing between the sheets so treated tissues, woven threads, or metal wires, running parallel to or intersecting each other." —*Chemical World*, Oct., 1914, p. 289. (J. A. W.)

SIMPLE SOLUTION METER.—"Operation of the counter-current decantation process at the Tom Reed mill, Oatman, Ariz., necessitated some method of accurately measuring, and regulating, the amount of solution drawn off for precipitation. The problem was solved by S. S. Jones, superintendent in charge of the property, by arranging a 6 ft. × 6 ft. vat into which all solution precipitated is collected. The vat has two discharge nozzles, one 1 in. diameter and one $1\frac{1}{2}$ in. in diameter. They have taper of 12° from the axis of the nozzle, with a coefficient of discharge practically equal to unity. The flow has been calibrated by measurement over stated periods of time, finding that the ratio of actual to



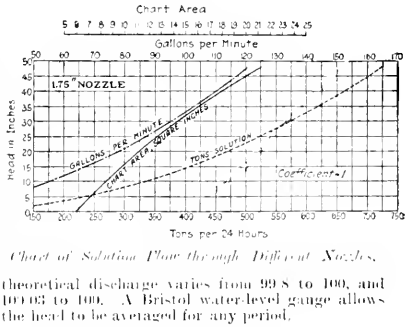


Chart of Solution Flow through Different Nozzles.

theoretical discharge varies from 99.8 to 100, and 100.03 to 100. A Bristol water-level gauge allows the head to be averaged for any period.

Solution measurement is based on the equation in which the discharge in cubic feet per second equals

$$AC\sqrt{2gh}$$

where C is the coefficient of discharge, A the area of nozzle in cubic feet, g the acceleration due to gravity, and h the head of solution in the vat in feet. Charts were plotted from this equation for each nozzle, and are shown in the accompanying illustration. The arrangement is entirely satisfactory. There is but one moving part, the dial of the gauge. The whole system was worked out and designed at the Tom Reef plant, with the single exception of the water-level gauge."—*Engineering and Mining Journal*, Oct. 10, 1914, p. 659. (G. H. S.)

Abstracts of Patent Applications.

(C.) 21.15. J. D. Sleeth. Improvements in tools for screening, hoisting, hauling and the like. 2.3.15.

This application refers to rope rollers for inclined shafts, and especially to alterations in the grease-box of a roller thereof.

(C.) 29.11.14. Arthur Edward Dexter, Arthur Edward Reynolds, and Improvements in or in connection with the ends of test mills. 5.3.15.

The specification refers to an improved liner for the ends, and more especially to an arrangement of the liner bars.

(C.) 6.12.14. William Mauss. Improvements in or connected with separators. 7.3.15.

[illegible] that the [illegible] are mounted at the end of the [illegible] with [illegible] separator No. 564,[illegible] patented by the same inventor. The [illegible] elements of the [illegible] are secured by [illegible] should the [illegible] in the drum be [illegible] that the [illegible] shall be carried right to the [illegible] side of the drum before [illegible] and discharged.

(C.) 29.11.14. George W. Porter, Ltd., and Cyril H. Richards. Improvements in or connected with [illegible] for [illegible] and the like. 27.3.15.

This application refers to [illegible] for tube mills, and consists in arranging [illegible] so that metal balls and [illegible] or more [illegible] the [illegible] that is, the [illegible] the [illegible] the [illegible] was [illegible] the [illegible] the [illegible].

[illegible] that a very [illegible] [illegible] from mine [illegible] [illegible] is [illegible].

(C.) 29.11.14. Edgar Allen Ulsh. Improvements in and in connection with the treatment of waste [illegible] for the production of [illegible]. 27.3.15.

This [illegible] the [illegible] relating to the treatment of waste [illegible] of the type in which [illegible] to recover the product [illegible] the [illegible].

(C.) 30.11.14. William Peterson. Improvements in or in connection with [illegible] for mechanical [illegible] and the like. 31.3.15.

The [illegible] [illegible] or linkage [illegible] [illegible].

[illegible] and [illegible] to the stem of the [illegible] and [illegible] [illegible] the [illegible] of the [illegible] and [illegible] [illegible] to prevent its being [illegible] out of place by the [illegible].

The loose jaw is operated by contact of [illegible] surfaces on the back of the jaw and on the [illegible] lever.

(C.) 5.1.15. Edgar Nathaniel Strover. Improvements in bearings. 6.3.15.

This application refers to a form of bearing particularly applicable to wheels of electric trolley-heads.

Its principal feature consists of a loose bush or series of bushes between the pin or spindle and the bore of the wheel or pulley, perforated to permit the passage of the lubricant from the hollow pin or spindle so as to lubricate the inside and outside surfaces of the bush or bushes.

The invention is also applicable to rope and guide pulleys generally.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

Aspland, C. H., *c/o* Nigel: Simmer Deep, Ltd., P.O. Box 178, Germiston.

Beath, W. B., *c/o* Australia: c/o H. T. Brett, Esq., Falcon Mine, Umvuma, Rhodesia.

Cavers, J. J., *c/o* Johannesburg: Nourse Mines, Ltd., P.O. Box 32, Denver.

Davies, L. J., *c/o* Mozambique: New Primrose G.M. Co., P.O. Box 193, Germiston.

Drake, F. J., Rhodesia: 12, Portland Court, Gt. Portland Street, London.

Fell, A. J., Mexico: Socorro Gold and Silver Mines, Ltd., Avenida de Angelis, Tegucigalpa, Republic of Honduras, Central America.

Honnet, W. L., Johannesburg: 147, Fraser Avenue, Orange Grove, U.S.A.

Horsfield, J. F., *c/o* Planet Mine, Arcturus, Rhodesia.

Main, J., *c/o* Gold in Veins Mine, Rhodesia.

Meiklejohn, J. T., *c/o* Bjelkestroom: Sheba G.M. Co., Nelspruit, Transvaal.

Mellor, H., *c/o* Hillside Farm, Highland Creek, Rathgar.

New, W. S., *c/o* Johannesburg: P.O. Box 26, Randfontein.

Rido, J. H., *c/o* Johannesburg: c/o Messrs. Price, Waterhouse, Snell & Rider, 8, Queen Anne's Gate, Westminster, S.W.

Ross, A. E., *c/o* Johannesburg: Geldenhuis Deep, Ltd., P.O. Box 34, Cleveland.

Seaton, P., *c/o* Swaziland: Modder Deep Levels, Ltd., P.O. Box 320, Benoni.

Strange, F. H., *c/o* Idaho: c/o Western Sales Co., 607 S.W. Realty Bldgs., Spokane, Wash., U.S.A.

Thomas, W., *c/o* India: "Lamorna," Heanton Avenue, Redruth, Cornwall, England.

Wyk, H. J. v., Kaap-la-Dop: Roodepoort United G.M. Co., P.O. Box 115, Roodepoort.

Wyss, E. M., *c/o* Bulawayo: Porges Section, Rhodesian Goldfields, Ltd., Randfontein.

Williams, G. W., *c/o* Australia: Captain, 222nd. Field Company Royal Engineers, British Expeditionary Force, Army Post Office, London.

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THE JOURNAL
OF THE
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OF SOUTH AFRICA.

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Vol. XV. JUNE 1915. (Last part of Volume) No. 11.

Proceedings
AT
Annual General Meeting, June 26th, 1915.

The Nineteenth Annual General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, 26th June, 1915, Prof. G. H. Stanley (President) in the chair. There were also present:—

27 Members: Prof. J. A. Wilkinson, Messrs. R. A. Barry, J. Gray, H. Meyer, C. Toombs, F. W. Watson, John Watson, H. A. White, Dr. W. A. Caldecott, W. R. Dowling (Members of Council), H. R. Adam, R. W. Bennett, H. C. Boydell, J. A. Campbell, J. M. Dixon, W. Lee, J. D. Marquard, P. T. Morrisby, J. M. Neill, S. Nettleton, S. Newton, W. S. V. Price, A. F. Rose, D. F. Scott, A. M. Sim, A. Thomas, and W. E. Thorpe.

11 Associates and Students: Messrs. H. C. F. Bell, O. A. Garber, A. King, G. Lundholm, A. Milliken, C. H. Olsson, F. J. Pooler, D. W. Pugh, I. R. Trollip, H. Ward, and W. Waters.

6 Visitors, and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

The President: The Secretary will now read the Report of the Council for the last year, and submit the Statement of Accounts.

ANNUAL REPORT OF COUNCIL.

It was anticipated and hoped last year, that the year which marked the coming of age of the Society would have been noteworthy as one of the most successful and progressive it had had during its working life of nineteen years. At the last Annual Meeting the distinct decrease reported in membership, revenue, and attendance at meetings, was rightly ascribed to the labour disturbances, and with the settlement of those troubles a return of prosperity was confidently expected.

The great European war has deeply affected every portion of our Empire, and has naturally made its influence felt here on the Witwatersrand. South Africa, in taking its place as a unit of the Empire, undertook as its share of assistance the subjugation of the German Colony of South West Africa, and a number of our members volunteered for active service when the call was made by the Government.

It was essential for the Empire's needs, however, that our great Industry should continue its work in providing gold necessary for the successful prosecution of the war, and as far as the Society was concerned it was resolved to give to that work the utmost assistance in its power.

Owing to the events of the last two years your Council regrets to report that although the work of the Society has been carried on, it has not received due support from the membership to enable it to carry on "business as usual" as successfully as it had hoped. It has been enabled to place before you excellent papers for discussion at the monthly meetings, but despite that, the attendance has fallen off, the membership has decreased, and the revenue has been considerably affected. In this time of stress there have been many calls on the generosity of everyone, and it is only natural, although unfortunate, that our Society should suffer in some degree.

A reference to the accounts which are submitted this evening will show you that the income of the Society has fallen off by no less a sum than £627. It was evident early in the year that economy would have

to be practised, and steps were taken to reduce the Society's expenditure as much as possible, consistent with the provision which has always been kept in view of giving members a return for their subscriptions, which compares more than favourably with other local Societies. In this direction a saving over the previous year's expenditure was made of over £786, every item of expenditure being considerably reduced. Despite this, the accounts show a deficit on the year's work of just over £201, an amount somewhat less than that of the previous year.

The subscriptions outstanding for the current year amount to £175 8s. 9d., of which £53 11s. has since been paid, and while the Council is fully aware that demands still continue to be made in connection with the war, it hopes that those members on the mines will recognise that a Society of this kind and calibre is one of the tools of their profession, and its work one of the necessities of their activities. The Council looks with confidence, therefore, to those members who are in arrear with their subscriptions for a recognition of their obligation to the Society by an early and prompt payment.

Membership.—Only 8 Members, 13 Associates, and 2 Students have been elected during the year, a number less than in any previous year. The resignations, 53 (32 Members and 21 Associates), have been greater than usual, owing to many relinquishing the mining profession for health and reasons connected with the War. There is a lamentable decrease in the number of Student members, which may be largely ascribed to the mobilisation of the Defence Force. The following losses by death are noted with regret: Messrs. G. A. Darling, A. M. Thomas, and Lieut. W. G. Fronde, D.C.M., of the Imperial Light Horse, who was killed in action during the rebellion, while Prof. S. B. Christy, one of our Honorary Members for many years, passed away in November last.

By the decease of Mr. George A. Darling the Society lost one of its foundation members, who was also at an earlier period an active member of your Council. The Society has recently been notified by the late Mr. Darling's executor that his relatives have decided to present to the Society all the note-books and experimental data which he collected during his long and meritorious connection with the Industry generally, and the cyanide process in particular.

Following the rule adopted in the past that all those whose subscriptions are in arrear for two years or over be struck off the roll, 13 Members and 55 Associates have thus ceased to be connected with the Society. For the benefit of such as may desire reinstatement, it may be mentioned that the Council has often considered, sympathetically, cases of hardship which have been submitted to it in confidence.

With regard to members who are on active service, special arrangements have been made.

The total membership is now therefore 805, showing a total decrease since last year of 145, the usual table being appended (p. 316).

The *Journal* has continued its useful work of supplying members with reports of the Society's proceedings, as well as extracts from the best technical literature available. Unfortunately, reasons of finance have demanded curtailment of this latter feature of the *Journal*.

The following is the list of papers and notes submitted to the Society during the year:—

July, 1914.—"The Blue Iron Cyanogen Compounds," by Mr. H. E. Williams.

August, 1914.—"Tube Mill Ends," by Mr. J. F. Pyles. (Note.)

"The Presidential Address, by Prof. G. H. Stanley.

September, 1914.—"Some Notes on Treatment of Zinc Gold Slime," by Mr. H. A. White. (Note.)

"A Method of Determining Gold in By-Products containing Platinum and Iridium, etc.," by Mr. H. R. Jolly. (Note.)

"Gold and Prices," by Prof. R. A. Lehfeldt.

October, 1914.—"Notes on the Use of Potassium Permanganate as Cyanicide in Sand Filling Solutions," by Mr. R. A. Cooper.

November, 1914.—"Notes on Treatment of Zinc Gold Slime," by Mr. A. King and Mr. A. Thomas. (Two Notes.)

"Notes on the Dust Allaying Plant at the Ferreira Deep," by Mr. S. Newton.

December, 1914, and January, 1915.—"Persistence of Ore in Depth," by Mr. T. A. Rickard.

February, 1915.—"Scale of Sulphate of Lime from Water Launder in Main Incline Shaft," by Mr. John Watson. (Note.)

"Notes on Treatment of Zinc Gold Slime," by Mr. J. F. Pyles. (Note.)

REVENUE ACCOUNT for year ending 1st June, 1915.

Dr.		1913-14. £ s. d.		1914-15. £ s. d.	£ s. d.	Cr.	1913-14. £ s. d.		1914-15. £ s. d.	£ s. d.
	To Charges (including Postages, Bank					By Subscriptions —				
(148 6 4	,, Charges, Insurance, etc.) ...		104 4 3			(1,037 8 0	Members, current year ...		709 16 0	
198 15 2	,, Printing and Stationery ...		106 0 7			86 1 0	do. arrears collected ...		12 12 0	
39 14 0	,, Rent, Lighting and Telephone...		29 0 5			438 12 9	Associates, current year ...		286 13 0	
627 0 0	,, Salaries *and Bonus*		596 0 0			65 17 9	do. arrears collected ...		7 17 6	
10 0 0	,, Depreciation on Furniture ...		10 0 0			6 16 6)	Students		1 1 0	
	,, Amounts Written Off as Irrecoverable		26 5 3							1,017 19 6
136 13 9)	,, *Balance Cost of Transactions — Written off*					1,634 16 0				
1,160 9 3				871 10 6		30 9 0	,, Entrance Fees			9 9 0
(50 0 0	,, Grants: *Seymour Memorial Library*					53 17 3	,, Interest on Fixed Deposits ...			19 10 5
20 5 0)	Prizes: School of Mines & *Society*			35 10 0			,, Sales of Transactions		33 18 9	
70 5 0							*Less* Cost of Covers and Binding		18 19 6	
16 17 6	,, *Loss on Annual Dinner*									14 19 3
	,, Society's Journal —					331 2 9	,, Balance, deficit on year's working			201 12 10
(1,271 10 0	Cost of Reporting, Printing, etc.	718 18 6								
−471 16 9)	*less* Received for Sub's. and Advts. after deducting commissions on									
802 13 3	Advts. and Collections ...	362 8 0								
				356 10 6						
£2,050 5 0				£1,263 11 0		£2,050 5 0				£1,263 11 0

BALANCE SHEET, as at 1st June, 1915.

Dr.		1913-14. £ s. d.		1914-15. £ s. d.	£ s. d.	Cr.	1913-14. £ s. d.		1914-15. £ s. d.	£ s. d.
	LIABILITIES.						ASSETS.			
(129 12 5	To Sundry Creditors		163 13 0				By Cash at Standard Bank —			
29 8 0)	,, Subscriptions paid in advance ...		26 5 0			(88 3 10	Johannesburg ...		36 12 9	
53 18 0	Mining Exhibition Fund ...		37 9 0			36 13 8	London		14 1 6	
				227 7 0		85 1 0)	,, *On Hand, Johannesburg*			
504 18 5						294 18 6				50 14 3
	,, Revenue and Expenditure Account —					(405 13 4	,, Cash on Fixed Deposit with accrued Interest		405 13 4	
(930 5 4	Balance at June 1, 1914 ...		579 2 7			102 9 8)	,, Cash in Post Office Savings Bank with accrued interest ...		53 0 8	
−351 2 9)	*Less* deficit as per Revenue and Expenditure Account, June 1, 1915		201 12 10			508 3 0				458 14 0
				377 9 9		80 0 0	,, Office Furniture ...			70 0 0
579 2 7						(294 2 3	,, Sundry Debtors ...			25 8 6
						2 2 9)	*less* Provision for Doubtful Debts			
						292 19 6				
£1,083 7 0				£604 16 9		£1,083 7 0				£604 16 9

We have examined the above Balance Sheet and accompanying Revenue and Expenditure Account with the books and vouchers of the Society, and have obtained all the information and explanations we have required. In our opinion such Balance Sheet is properly drawn up so as to exhibit a true and correct view of the affairs of the Society, according to the best of our information and the explanations given to us, and as shewn by the Books of the Society.

Sgd. ALEX. AIKEN } *Hon. Auditors*
and CARTER, }

JOHANNESBURG, 25th JUNE, 1915.

(Signed) JAMES LITTLEJOHN, *Hon. Treasurer*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

	June, 1895	June, 1896	June, 1897	June, 1898	June, 1899	June, 1902	June, 1903	June, 1904	June, 1905	June, 1906	June, 1907	June, 1908	June, 1909	June, 1910	June, 1911	June, 1912	June, 1913	June, 1914	June, 1915
Hon. Members	5	5	5	8	14	9	13	18	18	16	17	18	21	22	20	23	25	25	26
Life Members		1	1	1	1	1	1	2	3	4	7	7	8	9	9	9	10	10	10
Members	19	81	101	197	281	294	111	639	680	680	683	618	630	731	711	733	618	550	481
Associates				180	238	243	281	338	321	314	342	357	342	402	380	442	396	351	287
Students								1	8	4	7	18	20	60	57	22	21	15	2
	54	87	107	386	534	511	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224	1,177	1,229	1,070	951	806

"The Theory of Tube Milling," by Mr. H. A. White.

"Notes on a Method of Ventilating Sinking Shafts, Winzes and Drives," by Mr. S. Nettleton.

March, 1915.—"Lead Acetate and British Glass and Porcelain Ware." (Discussion.)

"The Use of Scoop Discharges in Tube Mills," by Mr. W. R. Dowling.

"Safety Measures in the Use of Explosives," by Mr. W. Cullen.

April, 1915.—"Robson-Salter Gate for Cone Discharges," by Mr. H. A. White. (Note.)

"Notes on the Practical Testing of Working Cyanide Solutions," by Mr. E. H. Croghan.

May, 1915.—"Sott Sandstone from Sand-Filling," by Dr. W. A. Caldecott. (Note.)

June, 1915.—"Notes on the Treatment of Antimonial Gold Ores from the Murchison (Transvaal) Range," by Mr. H. R. Adam.

Mining Exhibition.—The question of holding the Fourth Exhibition this year was carefully considered by your Council, but under the exceptional circumstances it was decided to postpone it for the present.

Awards and Grants.—Prizes were again awarded by the Society to successful evening class students at the School of Mines and Technology for success at the examinations in Chemistry and Metallurgy held last December. The Council has again decided to offer prizes for competition at the forthcoming examinations.

The question of making further grants to the Seymour Memorial Library, an institution which deserves every support, has for the present been postponed.

Visits.—A visit, of which advantage was taken by a large number of members, was made to the Modder "B" Mine in March. Other interesting visits were contemplated, but owing to both the war and rebellion these have had to be postponed.

A most cordial invitation has been extended to members to attend the International Engineering Congress at San Francisco in September next, and should any of our local members be visiting America about that date, and be desirous of attending the Congress, the Secretary would be glad to furnish the latest particulars. It is anticipated that the Society will be officially represented at the Congress by one

of its Past-Presidents and Corresponding Members of Council.

On the outbreak of the War the Council conveyed to the Minister of Mines the Society's offer to assist the Government in any possible way, an offer which was gratefully acknowledged, but up to the present no particular call has been made on the Society. In the meantime, however, the Society has freely lent the services of the Secretary and the use of its office to the South African Red Cross Society and the Defence Force Comforts Association. The two bodies inaugurated the work in the northern part of the Union of supplying comforts to the hospitals and troops on service, both in South Africa and overseas, and some idea of the magnitude of the work which has been accomplished in this direction may be gained from the fact that 1,485 parcels and 316 cases of comforts weighing approximately 38 tons have been despatched.

Unfortunately, however, large as these quantities may seem, spread over nearly ten months and between 40,000 and 50,000 troops and accompanying hospitals, they have been far from sufficient to meet the demands which have been and are still being made, and any further help in that direction will be greatly welcomed. The Council trusts that its action in lending its assistance in this manner will meet with your approbation.

The South African Red Cross Society, which you will remember was practically sponsored by this Society, is also actively engaged in further First Aid and Ambulance work on the mines and elsewhere. Your Council has arranged that the Red Cross Society undertakes the work of conducting the competitions for the Ambulance Shields presented by the Society and the Transvaal Coal Owners Association, and has handed the Shields to the Red Cross Society for that purpose under certain conditions. Owing to lack of entries, no competition was held for the Colliery Shields, but the competitions for the Underground and Surface Shields were duly held, with the result that teams from the Crown Mines, where particular attention is paid to ambulance work with much profit both to the mine and its workers, were successful in winning both Shields.

The Council has endeavoured to obtain a complete list of its members on active service, but has experienced much difficulty in connection therewith. Our Roll of Honour, so far as it is possible to obtain it at present, is as under: —

Allen, D. L., Rand Rifles, Union Forces.
Anderson, W., Analytical Branch, Union Forces.
Aymard, M., 2nd Lieut., Royal Garrison Artillery.
Barratt, R. L., Machine Gun Section, Union Forces.
Baumann, M., Q.M.Sergt., Natal's Scouts.
Bell, D. B., Rhodesian Regiment.
Blewett, E., Europe.
Bolitho, E. J., S.A. Medical Corps, Union Forces.
Burt, W., Max. Gun. Sect., Union Forces.
Cockburn, P. A., Chief Petty Officer, R.N.V.R. (Anti-Aircraft Searchlight Corps).
Collins, W. L., Imperial Light Horse, Union Forces.
Dick-Cleland, A. F., 2nd Lieut., 9th Batt. Border Regt.
Dixon, Clement, Rhodesian Reserve Force.
Donkin, W., Sergt., 2nd Rhodesian Regt.
Flynn, A. R., Natal Carbineers, Union Forces.
Fox, G. C., Major, Transvaal Scottish, Union Forces.
Gardiner, B. L., Union Forces.
Gerds, F. N., 2nd Lieut., Royal Engineers.
Gilbert, T. W., Europe.
Gill, H. W., Sergt., S.A. Sanitation Corps, Union Forces.
Graham, W. Hastie, Rhodesian Platoon, 3rd Batt. King's Royal Rifles, 80th Brigade, 27th Division, B.E. Force.
Gross, L. A., Rand Rifles, Union Forces.
Halford, J. E., King Edward's Horse.
Harris, J. V., Natal Carbineers, Union Forces.
Harrison, M. J., Imperial Light Horse, Union Forces.
John, W. E., Lieut., S.A. Aviation Corps., Union Forces.
Johnson, B. A., Pte., 5th Batt. The Buffs, India.
Leyson, L. T., Imperial Light Horse, Union Forces.
Mance, J. C., Capt., Train Conducting Officer, B.E. Force.
Mares, W. H., Imperial Light Horse, Union Forces.
Midgley, W. W., Union Forces.
Newbery, J. W., "C" Squadron, 2nd King Edward's Horse.
Nicolaus, R. C., Europe.
Nissen, P. N., Lieut., 103rd Field Co., Royal Engineers.

Paterson, G. S., 2nd-Lieut., Royal Garrison Artillery.
Patterson, P. F., Lieut., 7th Batt. York and Lancaster Regt., Flanders.
Porter, R. A., 2nd Transvaal Scottish, Union Forces.
Pryor, Thos., 2nd-Lieut., Queen Victoria's 2nd Sappers and Miners (Madras).
Rabone, P., Signaller, 2nd Rhodesian Regt.
Richardson, W. P., Botha's Light Horse (prisoner in G.S.W.A.).
Robinson, R. W., Capt., 5th Batt. Royal Inniskilling Fusiliers.
Robertson, G. A. Union Forces.
Robertson, J. R. H., Kolar Gold Fields Volunteers.
Ruggles-Brise, H. R., Lieut., Union Forces.
Simpson, G., 2nd-Lieut., 18th Batt. Durham Light Infantry.
Solomon, L., Sapper, S.A. Engineer Corps, Union Forces.
Stephens, V. H., "A" Co., Rand Rifles, Union Forces.
Smith, G. Hildick, Lieut., 1st Transvaal Scottish, Union Forces.
Stidolph, N. H., Lance-Corpl., 2nd Rhodesian Regt.
Stokes, R. S. G., Lieut., 17th Field Co., Royal Engineers, B.E.F.
Swinney, L. V. E., 28th County of London Regt. (Artists' Rifles).
Thomas, H., Witwatersrand Rifles, Union Forces.
Thomas, W., Lieut., Kolar Gold Fields Volunteers.
Trevor, T. G., Intellig. Dept., Union Forces.
Trump, F. J., Capt., 1st Monmouth. Regt.
Waterson, H., Witwatersrand Rifles, Union Forces
Weeks, R. F. J., Capt., Kolar Gold Fields Rifle Volunteers.
Williams, G. W., Capt., 222nd Field Co., Royal Engineers, B.E.F.
White, J. L., Witwatersrand Rifles, Union Forces
Winters, R., Heavy Artillery Brigade, Union Forces

KILLED IN ACTION.

Froude, W. G., Lieut., Imperial Light Horse. (During Rebellion.)

It will be noticed that the Council is represented by Lieut. G. Hildick Smith (1st Transvaal Scottish), at present serving in G.S.W.A., and of its Corresponding Members of Council, Capt. G. W. Williams, and Lieut. R. S. G. Stokes, both in Field Companies of the Royal Engineers, are serving on the Continent.

A trial is being given to a new procedure with regard to the papers presented at the monthly meetings, by printing them in the *Journal* issued prior to the meetings. It was felt that this procedure would enable members to study the papers beforehand, and hence be prepared to discuss them on their presentation to the Society. Up to the present, however, it cannot be said that the system has improved either the quantity of discussion or the attendance at the meetings. The trial will be continued for the present, but its continuance or otherwise will be considered by the new Council at an early meeting.

It has also been agreed upon, with your approval, to hold only eight meetings a year in future, the holiday months of July and August, December and January, being taken as a recess. This corresponds in a modified way to European practice, where a four months recess is usually taken in the summer months. The *Journal* will, however, be issued as formerly.

A suggestion was made recently by one of our members that a return should be published, showing instead of the present results of stamp duties of the Transvaal mines, the efficiency of plants calculated on the system introduced by Dr. Caldecott of -90 duty per N.C.U. (nominal-crushing unit). The matter was referred to the Chamber of Mines for its consideration, and, in reply, that body has stated that even if correct figures showing the crushing efficiency of the various mines could be obtained, the information would not be of sufficient benefit to the Industry to warrant the additional work involved on the mines. It was, however, decided that from January 1st next the stamp duty column in the Chamber's monthly return will be deleted, and in its place will be given a column showing the total nominal crushing capacity of each mine.

Attention has been given in the proceedings of our meetings to the necessity for encouraging local industry as far as possible in the provision of materials for the mines, the usual supply of which has been either considerably curtailed or stopped owing to the war. The subject has received great attention in England, where in the past vast supplies of chemicals, glassware, etc., have been obtained from enemy countries, which it has now been found commercially possible to manufacture in Great Britain. It is hoped that every support will be given to these attempts.

The offices of the Society, after being for 6½ years in this building, have been removed to the Old Municipal Building, owing to the Staff of the School of Mines requiring further accommodation.

During the year 11 Council Meetings have been held, as well as numerous meetings of the Editorial, Finance, and Special Committees. The attendances, which averaged 11 per meeting, have been as follows:—

G. H. Stanley ...	11	C. Toombs (5) ...	7
J. A. Thomas (1)	6	F. W. Watson .	8
E. Pam	7	John Watson (6)	8
J. A. Wilkinson .	10	A. Whitby (7)...	3
J. Littlejohn ...	5	H. A. White ...	10
R. A. Barry	4	J. S. Cellier (8)	6
F. L. Bosqui (2)	3	R. G. Bevington	1
K. L. Graham (3)	1	W. Cullen	1
J. Gray	8	W. R. Dowling	1
H. Meyer	9	A. McA. Johnston	2
G. H. Smith (4)	2	J. Moir	4

(1) Granted 6 months leave.
(2) Resigned in March.
(3) Appointed in March, *vice* Mr. Wartenweiler, resigned.
(4) On Active Service in German S.W. Africa.
(5) Granted 4 months' leave.
(6) Granted 2 months' leave.
(7) Absent from 5 meetings on account of illness.
(8) Appointed alternate to Mr. M. T. Murray, and to Council *vice* Mr. Bosqui, resigned.

Messrs. M. T. Murray and F. Wartenweiler, elected last year, were granted leave of absence owing to absence on holiday and business, but resigned late in the year when they found it impossible to return. Both members were greatly missed. The Council desires to place on record its appreciation of the services of one of its Past-Presidents, Mr. W. Cullen, who left for England to place his expert knowledge of explosives at the service of the Empire. Since joining the Society in 1902 he has been indefatigable in working for its interests, as well as those of the country generally, and his services will be greatly missed. We regret to state that Mr. Andrew F. Crosse, our oldest Past-President, resigned his membership of the Council in November last, but at the same time we desire to assure him how highly the Society values the work he has done in its interests.

Our membership in Mexico has been greatly affected by the turbulent state of the country, and our Corresponding Member of Council for that State, Mr. Walter Neal, after being wounded during the troubles there, left for the United States. Our thanks are due to him, as also to the other Corresponding Members of Council, for the services they have rendered to the Society.

Two of our Honorary Members have recently been honoured by the King, Sir Thomas Kirke Rose and Sir R. A. S. Redmayne, and the Council forwarded its congratulations to them.

Mr. F. D. P. Chaplin, M.L.A., resigned his membership of the Society on taking up the position of Administrator of Rhodesia, and the Council are pleased to state that His Honour will retain his interest in the Society, having accepted the position of an Hon. Vice-President.

Your thanks are due to the members of the various Committees of the Council for much valued work in the interests of the Society.

The Society is again indebted to the local press for reports of its proceedings, and also to Messrs. Aiken and Carter, Hon. Auditors, for the painstaking care with which the audit of our books is carried out.

The President: I beg to move the formal adoption of this Annual Report and Balance Sheet. I have not very much to say about it, excepting that it might have been a lot worse under the circumstances.

Mr. W. R. Dowling *(Past-President):* I have pleasure in seconding it.

Prof. J. A. Wilkinson *(Vice-President):* I have much pleasure in rising to support this resolution. I should just like to say a few words about the Balance Sheet and the financial position of the Society as it stands at present. In these times of stress and adversity we are now no longer confronted with the cost of the Society's *Transactions*, which was finally written off last year, and, practically speaking, if that is deducted, it will be found that the Balance Sheet is, roughly speaking, just about the same as it was last year. Considering the times through which we have passed, I think we can congratulate ourselves that such is the case, and I think we can also congratulate ourselves that the Society has been able to carry on business as usual. We have been able to meet regularly and publish our *Journal* as usual, and we have been able to carry on from the financial point of view, which, after all, deserves every consideration. This year I hope those of our members who can will do their best to help us in obtaining advertisements for our *Journal*, because it will assist in reducing its cost to the Society—or, in other words, if we are able to get more money, we are able to give our members slightly better value. The

nual subscription to the Society is small, the same or less than many societies at Home, where the printing and other expenses are so much smaller. Our other great expense is in the administration of the Society. We have, at present, a small balance; but only a small balance; it is not nearly what I should like to see it. We do not want to draw upon that balance if we can possibly help it, because we hope the Society will have many more years of usefulness before it, and we should like to conserve that balance. Therefore, what the new Council will have to do is to try, if possible, to make revenue and expenditure meet as far as possible. With your present Treasurer who will be re-elected this evening, members can be quite certain that everything will be done to see that that is the case. I have much pleasure in supporting the resolution that the Annual Report and Balance Sheet be passed.

The adoption of the Report and Accounts was carried unanimously.

The President: Before we pass on to the next business, I feel I must refer very feelingly to the cordial co-operation and support which the members of the Council have given me during my year of office. It has not been for any one an easy year to get through, and the Council has done everything possible to keep things going with the least possible dislocation; and it has succeeded in doing so in great measure, and I hope its efforts next year will be even more successful.

SCRUTINEERS' REPORT AND DECLARATION OF ELECTION OF OFFICERS AND COUNCIL.

The Secretary read the Scrutineers' Report, as follows:—

Johannesburg,
26th June, 1915.

The President,
Annual General Meeting of Members.

Dear Sir,

We have to report that we have this day examined 100 ballot papers, handed to us by the Secretary.

Our scrutiny enables us to declare the result of the ballot as follows:—

President.

Mr. Jas. E. Thomas (unopposed)

Vice Presidents:

Mr. H. A. White.
Mr. R. A. Barry.
Prof. J. A. Wilkinson.

Hon. Treasurer.

Mr. James Littlejohn (unopposed).

Members of Council.

Mr. J. Gray.	Mr. H. S. Martin.
Mr. K. L. Graham.	Mr. C. Toombs.
Mr. E. Pam.	Mr. H. S. Meyer
Mr. A. Whitby.	Mr. E. H. Crogan.
Mr. John Watson.	Prof. J. S. Cellier.
Mr. F. W. Watson.	Mr. M. Torrente.

(Signed) H. R. ADAM.
PERCY T. MORRISBY.
S. NEWTON.
A. THOMAS.
F. J. PODLER.
Scrutineers

The Secretary: I have to apologise on behalf of the new President, Mr. Jas. E. Thomas, for his absence this evening, and to express, at his request, his thanks for the honour you have conferred upon him. Mr. Thomas has been away at the Kowie for the last six months, and will be back again next week.

Mr. R. G. Bevington has also desired me to express his regret at being unable to be present at this meeting.

NEW MEMBERS.

Messrs. F. J. Podler and S. Newton were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the following candidate for membership had been elected:—

MURRAY, ROBERT HORSBURGH MACDOUGALL, Durban Roodepoort Deep, Ltd., P.O. Box 110, Roodepoort Reduction Foreman.

The Secretary announced that the following Associates and Students had been admitted by the Council:—

As Associates

FORREST, BRYCE CHARLES, Bantjes Consolidated G.M. Co., Ltd., P.O. Box 2, Florida. Mine Surveyor.

OLDMAN, DAVID, 36, Upper Ross Street, New Doornfontein, Johannesburg. Cyanider.

As Students

PUGH, DOUGLAS WALTER, 16, Bezuidenhout Street, Troyeville, Johannesburg. Metallurgical Laboratory Assistant, S.A. School of Mines and Technology.

ROSS, GEORGE, 55, Ascot Road, Bertrams, Johannesburg. Metallurgical Laboratory Assistant, S.A. School of Mines and Technology.

GENERAL BUSINESS.

VOTE OF THANKS TO SCRUTINEERS.

Mr. F. W. Watson *(Member of Council):* I beg to move a hearty vote of thanks to the scrutineers for their labours in connection with the ballot for the election of officers.

The vote of thanks was carried unanimously.

Mr. F. J. Pooler briefly acknowledged the compliment on behalf of the scrutineers.

ELECTION OF AUDITORS.

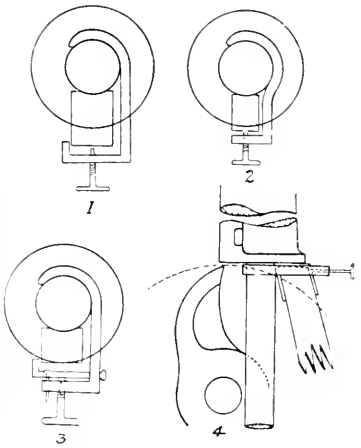
Mr. John Watson *(Member of Council):* I beg to move the re-election of Messrs. Aiken & Carter as honorary auditors of the Society for the ensuing year, and to propose a hearty vote of thanks to them for their work during the past year.

This was carried by acclamation.

SAFETY LOCKING DEVICE FOR STAMPS IN STAMP MILLS.

Mr. J. M. Neill *(Member):* I have here a small device to which I would direct attention. It is a "Safety-first" measure for the purpose of locking the stamps in a stamp mill. We have heard a good deal about this sort of thing; but not one of them gives continuous safety—that is, safety all the time the man is working. This one does. That is why I should like to call the attention of the members of this Society to it. It does not matter what is being done under the stamps (some of the devices have to be removed whilst the men are still working), but this device has not to be removed at any time at all until the actual stamp being worked on has to be brought into operation. The credit for the design of this apparatus belongs to Mr. Fuchs, of the Knights Deep. It is not patented; but it performs the whole of its duty all of the time and, for that reason, I am bringing it forward. I think if it is adopted by any of companies they ought to recognise the inventor of the idea in some pecuniary way.

Mr. H. A. White *(Vice-President):* I should just like to remark, with regard to this device, that I find one thing noticeable about it, and that is that the shiftsman actually uses it; but with regard to other safety devices for similar purposes, he finds it is too much trouble to use them.



Clamp. $1\frac{1}{2}$in. × $\frac{3}{4}$in. flat bar.
Cost from 5s. to 6s. each.
Five will be sufficient for 100 stamp mill.

1. Clamp shown where finger projects beyond tappet. Finger cap not slotted. Lug on clamp.

2. Clamp shown where finger does not project beyond tappet. Finger cap slotted. No lug on clamp.

3. Clamp shown where finger cap is solid cast and is not drilled or slotted. Clamp with small independent member with lug working freely on pin or pins, or pin and slot in clamp.

4. General arrangement showing clamp in position at side of stem away from cam.

The President: I am sure our best thanks are due to Mr. Neill for bringing this device before us, and I hope it will be adopted as a means of lessening accidents on the Rand

NOTES ON THE TREATMENT OF ANTIMONIAL GOLD ORES FROM THE MURCHISON RANGE.

By H. R. Adam, B.Sc. (Member).

The following notes are based on various experiments made on samples of ore from the so-called "Antimony Belt" in the Murchison Range District, N.E. Transvaal[1] and on observations made on the spot.

Among the numerous "reefs" of the district this is the most remarkable in extent and nature; it has been traced over forty miles east and west. The comparatively small amount of development work that has so far been done is quite promising, but the problem of obtaining a profitable extraction of the gold from these very refractory ores has not yet been solved.

From the metallurgical standpoint the ores may be classed as follows:—

I. Partially weathered or oxidized surface ore "rich" in antimony.

II. Partially weathered or oxidized surface ore poor in antimony.

III. Sulphide ore "rich" in antimony.

IV. Sulphide ore poor in antimony.

The reason for this classification is that when the antimony content is not too high, say not much above 1 %, the gold can be extracted with fair success by standard methods (amalgamation, concentration, cyaniding) without special preliminary treatment. In this connection it is to be noted that "rich in antimony" here means ore with an antimony content of 2 % and above.

Generally speaking, the nature of the bodies is such that by selective mining and sorting such a classification could be made. The ore richer in antimony is principally on the footwall of a quartz reef, which latter supplies the ore for classes (II.) and (IV.). This, of course, is a general statement: periodically lenses of stibnite occur in the quartz, sometimes of great extent and purity, but such ore would no doubt be specially mined for the antimony.

Treatment of Class I. ore.—Attempts to obtain the gold from this material by direct amalgamation and cyaniding gave very poor results. About 50 % may be extracted with a very high cyanide consumption. Although the ore is referred to as oxidized, it always contains a large proportion of sulphide and oxysulphide of antimony. Even on the surface practically pure stibnite is often found with only a film of oxide. This is probably due to the very dry climate and to the fact that antimony sulphide does not weather easily as compared with iron pyrite or arsenical pyrite. A sample of crushed ore very rich in antimony sulphide was exposed for five months and periodically moistened; at the end of that time it had altered very little in appearance.

The cyanide solutions from the treatment of this semi-oxidized material, and, in fact, from any ore containing much sulphide of antimony, always contain more or less antimony. This is due to the action of the alkali in commercial cyanide on antimony sulphide. Addition of lime or soda to the solutions, therefore, only increases the consumption of cyanide and the amount of antimony dissolved.

A sample of this class of ore assaying 9·4 dwt. of gold per ton and 10 % of antimony (a large proportion of which was sulphide) was roasted on a shovel over an open fire at a temperature considerably below red heat. An extraction of 86 % of the gold was obtained by cyaniding the roasted ore with cyanide consumption at the rate of 1·2 lb. per ton of ore. This result was surprisingly good, and attempts to repeat it were much less satisfactory. The reason probably was that in this case the roasting was carried out very slowly with excess of air and at such a temperature that absolutely no fritting or fusing of the antimony sulphide occurred. In other trials on similar material the roasting was done in a muffle at a higher temperature and with too little air.

This semi-oxidized material can be concentrated with fair efficiency; the more oxidized it is the better the concentration, since oxide of antimony wets readily and does not slime in crushing so badly as the sulphide. As the oxide is easily smelted for antimony metal, there would probably be a market for it, though whether its gold content would be paid for seems doubtful.

Treatment of Class II. Ore.—This generally has the highest gold content. There is, of course, only a limited supply of it, but very rich pockets occur. Even with this material direct amalgamation and cyaniding gave poor results, the consumption of cyanide being very high. Small scale tests showed that amalgamation followed by concentration and cyaniding of the tailings

would give a gold extraction between 70 and 80 % of the total contents. Pan amalgamation would give better results than plate, but results in the production of too much slime for efficient concentration. Since the cyaniding of the tailing is only feasible after good concentration, it would have to be decided whether fine grinding and amalgamation with poor concentration of the tailings would yield more profitable results than coarser crushing with a minimum of slime and good concentration. It is not only the antimony which necessitates concentration before cyaniding: copper may be present up to $\frac{1}{2}$ %, and traces of nickel and cobalt may also be present.

Treatment of Class III. Ore.—This is possibly the most interesting of the three classes, although probably the least profitable. It was generally found that the samples very high in antimony (40 to 50 % antimony) had, with a few exceptions, a low gold content. On the other hand, where stringers of quartz are found intermingling and the antimony percentage is lower, the gold values are better. The evidence available goes to show that there is an immense quantity of ore of this nature, that is, roughly speaking, an ore containing antimony as sulphide from 2 % to 20 % and 30 % and with gold from a few pennyweights to an ounce per ton; consequently a profitable method of treating it would be of great importance. It is to be noted, however, that ore answering in gold content at least to the remark applied to many "refractory" ores by Mr. A. F. Crosse, must be expected.[2] It may be remarked here that the word "ore" is applied to material in which there is a very doubtful profit. Unfortunately there does not seem to be a more suitable word.

Numerous patents have been taken out and many suggestions made with regard to the treatment of similar material. Particulars of most of these will be found in the reference quoted.[3]

Of the dry or roasting processes, those of Herrenschmidt and Chatillon are said to be the most successful. These depend on the production of the volatile oxide Sb_2O_3 by roasting the ore with coke or charcoal in a carefully regulated current of air, at a temperature not above 400° C. In small scale tests carried out in a muffle, from 80 % to 90 % of the antimony in samples containing about 20 % antimony was easily removed by volatilization. The residues were always found to be partly fused, and contained antimony as sulphide, oxide, and probably also as oxysulphide and antimonates. Cyaniding tests on these residues were unsuccessful, less than 50 % of the gold being dissolved with very high cyanide consumption. The results obtained indicated that smelting with lead would be required for such residues.

Of the numerous wet processes proposed for the treatment of similar ores, many depend on the decomposition of the antimony sulphide by means of solutions of the alkalies and alkaline sulphides. Since so many patents have been taken out for treatment on these lines, some success must have been attained. In small scale tests tried on Murchison Range ore, using solutions of sodium hydrate and sodium sulphide, the decomposition of the stibnite was so incomplete that any attempt to extract the gold from the residues by cyaniding would have been hopeless. In the MacArthur process, using caustic soda, it is said that roasting, chlorination, amalgamation and cyaniding may be employed to extract the gold from the residues.[4] In most cases these methods could only be applied to concentrates, and satisfactory concentration of antimony sulphide is rather a difficult problem owing to its great tendency to form slime in crushing.

Strong hydrochloric acid attacks antimony sulphide readily and is apparently used in the manufacture of paints from stibnite.[3] The quantity of acid required makes its use impracticable except perhaps for the treatment of rich concentrate. The reaction is expressed by the equation:

$$Sb_2S_3 + 6HCl = 2SbCl_3 + 3H_2S$$

It is possible that a practicable method might be evolved using gaseous hydrochloric acid and removing the antimony chloride as soon as formed by distillation (antimony trichloride volatilizes about 140° C.). Some attempts made on these lines were promising; the ore was mixed with magnesium chloride and steam was passed over the heated mixture, the object being to form hydrochloric acid gas by the action of steam on heated magnesium chloride, the hydrochloric acid in turn decomposing the stibnite. The antimony chloride and hydrogen sulphide were passed into water, where reprecipitation of orange-coloured antimony sulphide took place according to the above equation. The trituration of the antimony sulphide and magnesium chloride mixture usually caused the

stoppage of the reaction long before complete removal of the antimony as chloride.

Very interesting results were obtained on this class of ore by applying the acid flotation process of Delprat and Potter.[5] The greasy nature of the stibnite and the presence of calcium carbonate in the ore renders it suitable for this process, at least as far as the concentration of the antimony is concerned. The results so far obtained are only sufficient to show that this is a hopeful line for experimental work. The table of results given shows that the greater proportion of the gold remains with the residues, more or less free from antimony sulphide. It may be explained that the floating scum or froth of sulphide is called the "concentrate", and the remainder the "tailing", and that the aim has been to produce a "concentrate" rich in stibnite and poor in gold with a "tailing" containing as much of the gold as possible with the least quantity of stibnite. Generally there is more or less slimy intermediate material produced which has not time to settle with the "tailing", this is referred to as "middling". Except in one instance the middling was subsequently added to the "tailing", although better gold extraction would probably be obtained from the tailing by keeping the "middling" separate, since the latter contains more antimony sulphide.

There are considerable discrepancies in the results given, due to mechanical loss of material and the fact that the quantities of the products were not always sufficient for accurate gold assaying, which is by no means easy with antimonial ores. The actual separation of antimony sulphide from the sandy residue is undoubtedly much better than is expressed by the results in the table, but the difficulty is to prevent, in the form of apparatus used, the sinking of some "concentrate" during the removal of the products. It will be noticed that the percentages of antimony in tests 6, 7 and 8 are not so high, and that the gold values are higher. The explanation of this is that these tests were made on a mixture of ore of classes III. and IV. As will be mentioned later, Class IV. ore contains some copper pyrite in addition to the antimony sulphide, and the tests were made to indicate the effect of the treatment on unsorted sulphide ore. The results in these cases indicate that the gold is more closely associated with the pyrite than with the stibnite. The work is not sufficiently advanced as yet to give results of the treatment of the "tailing" for gold extraction: the fact is that with improved methods for removing the "concentrate" quickly and cleanly much better results may be hoped for, consequently it would be more profitable to work on the flotation process itself before dealing with the tailings for gold extraction. Unfortunately it is rather difficult to obtain information on the latest developments of flotation processes.

With regard to the methods adopted in these experiments, trials were first made in an ordinary beaker, the floating concentrate being skimmed off by hand. As recommended by Hoover[5], a 3% solution of sulphuric acid at a temperature of 70° C. to 80° C. was used. The ore was added to the hot acid in the proportion of about 1 to 6 by weight. Tests (2) and (3) were done in this way. It was found to be difficult to obtain a clean separation by this means owing to the sinking of some "concentrate" during skimming, consequently a tall glass cylinder, as shown in the illustration, was

Fig. 1. Showing floating "Concentrate" and settled "Tailing". Pulp admitted from upper side tube, and Steam at lower side tube.

Fig. 2. Showing overflow tube for "Concentrate".

Number of Test.	Weight of Ore Treated in Grams.	Dwt. of Gold per Ton.	Percentage of Antimony.	Weight of "Concentrate" in Grams.	Dwt. of Gold per Ton of "Concentrate."	Percentage of Antimony in "Concentrate."	Weight of "Tailing" in Grams.	Dwt. of Gold per Ton of "Tailing."	Percentage of Antimony in "Tailing."	Percentage Weight of "Concentrate."	Percentage of Total Gold in "Concentrate."	Percentage of Total Antimony in "Concentrate."	Percentage Weight of "Tailing."	Percentage of Total Gold in "Tailing."	Percentage of Total Antimony in "Tailing."	Details of Samples Tested.
1	175	8·0	16·1	39	2·4	62·9	130	10·2	1·7	22	7	87	74	91	8	Sample crushed to −60 mesh, slime not removed.
2	700	6·7	28·1	305	6·1	57·5	390	7·5	5·0	43	39	89	56	62	10	Sample crushed to −30 mesh, slime washed off.
3	230	3·0	19·5	57	2·4	12·8	162	3·3	11·8	25	20	54	70	77	13	Slime sample.
4	700	8·3	3·7	55	9·0	37·5	635	7·9	0·7	8	8	80	91	86	17	Sample crushed to −60 mesh, slime not removed.
5	500	7·5	7·2	51	7·6	42·0	400	8·0	2·1	11	11	63	80	85	23	Sample crushed to −60 mesh, slime not removed.
6	400	12·0	2·5	32	30	25·5	326	10·9	Trace	8	20	82	82	74	Trace	Sample contained stibnite and copper pyrite, crushed to −30 + 50 mesh
7	300	20·4	3·5	30	38	30·6	258	19·1	0·3	10	18	87	86	81	7	Sample contained stibnite and copper pyrite, crushed to −50 + 90 mesh
8	300	20·6	8·3	56	39	11·2	236	16·2	0·7	18	35	92	78	62	6	Sample contained stibnite and copper pyrite, crushed to −90 mesh

NOTE.—In all cases with the exception of No. 6, test most of the "middling" was subsequently added to the "tailing." In No. 6 test the following results were obtained for the middling.

Weight of middling, 30 gm. = percentage weight of "middling" ... ... 8.
Dwt. gold per ton, 23 = percentage of total gold in "middling" ... 14.
Percentage of antimony, 5·6 = percentage of total antimony in "middling" ... 17.

as dealt in subsequent work. With this device the tailing could be drawn off from the bottom. The acid solution in the cylinder was heated by steam, which also served for agitation. The ore was added pulped with a little acid, and after about one minute's agitation the steam was turned off and the tailing allowed to settle before drawing it off from the bottom. It was found that the proportion of ore to acid could be increased up to about 1 to 1 without apparent difference to the separation. In the tests numbered 14 and 15 the process was made to a certain extent continuous by adding sufficient pulp to cause the floating scum to overflow. The result of this was to give a "tailing" quite as free from stibnite, but a less clean "concentrate". As used in practice in the concentration of lead, zinc and copper ores, etc., the process is, of course, continuous, and although in the original processes of Delprat and Potter no oil was used, it has apparently since been found advisable to use a little. In these small scale experiments no oil was used. Although this separating funnel device is an improvement on the beaker, it is not quite satisfactory; it will be easily seen on experimenting that what is wanted is a method of immediately removing the "concentrate" so that there can be no possibility of its sinking after its formation. An experimental apparatus of this nature is described by Hoover.[5]

Class III. Ore.—At the Weigel Mine, at a depth of 200 ft., there is a very fine exposure of the quartz reef containing veins of the sulphides of antimony, copper and iron. The total amount of sulphide is rarely above 3%. Similar "pyrite reefs" have been exposed at other mines on the "belt". By concentrating and fine grinding and cyaniding 80% of the gold content was extracted from samples of this material.

Acknowledgments and thanks are due to Prof. G. H. Stanley for advice and information, and to the Weigel Gold Mining Company for permission to use data obtained when in their employment, also to Mr. G. Ross for much assistance in assaying.

References.

1. *The Geology of the Murchison Range*: H.M.S.A. Govt. Geological Surveys.

2. "The trouble is that very often the so refractory" ores are no more than ores more easily treated." A. F. Crosse, *Journal* of the Chemical Metallurgical and Mining Society of South Africa, January, 1913, p. 315.

3. *Antimony*, C. Y. Wang, pp. 154-160.

4. S. Sidwyn Brown, *Australian Mining Standard*, May, 1906, p. 522.

5. Hoover, *Concentrating Ores by Flotation*, p. 80.

Mr. Jas. Gray (*Member of Council*): I would like to propose a very hearty vote of thanks to Mr. Adam for his paper on Antimony.

Mr. F. J. Pooler (*Associate*): In the treatment of Class III. ore the author says that about 20% of the antimony was easily removed by volatilisation. I should like to know if he was able to collect the sublimate and was it all Sb_2O_3, or was some Sb_2S_3 sublimed unchanged? It seems to me that the economic extraction of the antimony may, in a measure, turn on this, as the ore seems "difficult" from the gold point of view.

Then with regard to the ingenious HCl extraction method, if the reversal of the action back to Sb_2S_3 could be prevented, would it not be an advantage? It occurred to me that heating with common salt by superheated steam might be worth trying, though as less (if any) H_2S would be evolved, I think it might probably be better to leave out the steam and heat the ore to about 150° C. with common salt alone, and try to volatilize the $SbCl_3$ directly. Can the author give me any information on the point, please?

Turning to the table, it is a pity that the author cannot inform us what percentage of the gold was capable of extraction by ordinary methods from the "acid-flotation" method that he has demonstrated to-night. I see that an 80% extraction is spoken of in No. 1 method, but in the table we are given the weights of gold in "concentrate" and "tailing", assayed as usual, presumably. This, however, does not mean that the extraction is feasible by ordinary means, as a very small quantity of Sb_2S_3 apparently complicates extraction.

We shall look with interest for some further information on this point. In the meanwhile we can congratulate ourselves on a very interesting paper, showing the possibilities, if not the probabilities, of "outside" propositions.

Mr. H. Meyer (*Member of Council*): This paper is of great interest, in view of the fact that the reefs of the Murchison range have been lying fallow for many years, partly on account of inaccessibility

and partly on account of the refractory conditions.

As far as I am aware there is only one mine in the country where gold ores contaminated with antimony have been treated on a large scale. At the Cam and Motor Mine, in Rhodesia, an ore consisting largely of arsenical pyrite with a comparatively small percentage of stibnite is treated for its high gold content. After extensive development, resulting in the disclosure of some 750,000 tons of ore valued at 42s. 6d. a ton, elaborate experiments were conducted to determine the most suitable method of treatment. These being satisfactory, an experimental treatment plant handling 600 tons a month, was put to work in 1912. Laboratory experiments indicated an extraction of 85 %. The treatment in the experimental plant consisted of dry crushing in ball mills, roasting in mechanically operated furnaces of the Edwards type, fine grinding in contact with cyanide, and final recovery from solution after vacuum filtration. No attempt was made to recover either arsenic or antimony. After further experiments and trials a large plant of 6 mills and 6 roasting furnaces, capable of dealing with 15,000 tons a month, was erected, and commenced running in February, 1914. Difficulties arose making it impossible to realize the original estimates of extraction, and until quite recently the extraction only averaged 75 %.

I bring these points forward to illustrate the difficulties attending the successful treatment of ore from a fully developed mine with high gold values, in contrast to the example cited by Mr. Adam, where the antimony takes first place and the gold second.

I trust that Mr. Adam's research work will do much towards directing attention to the future possibilities of mining fields which are to-day scarcely popular, perhaps not entirely owing to the refractoriness of their ore bodies.

SAFETY MEASURES IN THE USE OF EXPLOSIVES.

By Wm. Cullen (Past-President).

(Printed in Journal, March, 1915.)

DISCUSSION.

Mr. W. S. V. Price *(Member.)*: I must apologize for again speaking on this subject, but some remarks made by Mr. Marquard, I think, necessitate some reference. In view of the number of speakers who differ from my views, it may seem like presumption on my part to press my point, but I would like to point out in connection with anything I say, or any experiments I have made, that they are based on actual working practice underground, and I submit and maintain that not any of the experiments carried out on the surface quoted by others, have no analogy, or practically none, underground.

I notice continual reference has been made to the good quality of the explosives used on the Rand. It almost seemed to me as protesting overmuch. We miners know that some are good, and some are not so good. The manager for a particular make of explosive was at great pains to explain to me that there was absolutely no difference in the explosives used, their mode of manufacture being identical, therefore there could be no difference. If that is so, why are miners so partial to certain makes, and why do they get more misfires when using other brands.

Mr. Marquard agrees with Mr. Cullen as to the good quality of explosives used, but on page 295 of the *Journal*, he is reported as giving one of the causes of misfires as "bad quality of explosives used." This is not the only instance where he contradicts his own statements. On page 297, in accounting for a so-called running fuse, he says: (a) "Primers at bottom of holes, fuse near top of charge sets explosives alight, temperature rises to 285° C., and charge explodes before fuse has burned down to detonator." On the same page, in another column, I read as a cause of misfires, "Explosives burning and charges being smothered." "In a hole it is confined behind tamping, the burning explosive may smother out before it gets to a temperature of 285° C., thus unexploded sticks may remain, especially as it is well known that even detonators can set charges alight." So that it would seem as if a piece of fuse would be able to generate more heat than even a faulty detonator!

In support of my theory that in certain cases, such as development, or even stoping in broken ground where there are many parting planes, it is advisable, if not absolutely necessary, to put the primer elsewhere than at the top of the charge, I can quote an instance that has occurred since I attended the meeting two months ago.

At the last meeting I was unfortunately ill, and as I was absent from my work, another man was sent to work in my place. He called and saw me, and told me that after blasting a certain round, he had four cut-off fuses out of 10 in the round. When I asked him where he had his primers, he told me they were the last to go in! I should like to know what amount of heat, and what length of time would be required to fire a stick of gelatine. I have brought a couple of pieces of fuse with me to demonstrate the amount of heat given out by a burning fuse. The paper is a light ordinary packing paper, not nearly as stout as the covering on a stick of gelatine, but I will wrap a small piece tightly round the fuse, which will correspond to the pressure applied in a hole, and hold it in my hand while I light it, and the fuse spits at the other end. (Experiment shown.) It will be seen on examination that the paper shows no heat marks on it that would fire a stick of gelatine: it is not scorched; the only mark is where the tar oozed through the fuse. That was a new piece of fuse. This other piece is old, and has been in my house for some months. After demonstrating by burning, the speaker continued: It will be seen that the paper is scorched a little, though very little; but enough to show that an old or deteriorated fuse may possibly be a source of danger. But I say that even the demonstration I have given you here is not analogous to conditions underground, for the fuse and paper were both bone dry, while our drill holes are never pumped as clean as that, as most miners prefer a little water or sludge in the hole to fill up any interstices between the sticks of gelatine and prevent air spaces.

On page 297, under the heading "Improper and unscientific charging" (b) tamping in between cartridges is quoted as a cause of misfires. At present the law prohibits this, but it was not always so, and while with some gelatine it may be dangerous, with other gelatine it is quite safe. A few years ago I had charge of a small property about 30 miles west of Krugersdorp. The man who had been there before me had been using Nobel's gelatine in ground in which black powder, or 40 gelatine-dynamite, would have been more serviceable. It was like driving tin tacks with a 14 lb. hammer. However, I had to use what was there, so for a 36 in. hole I put half a stick of gelatine at the bottom, then a good stick of tamping, then a half stick of gelatine, another stick of tamping, and finally half a stick of gelatine for a primer, tamping firmly again. I have in my possession a stick of tamping that has been in a hole, and the charge broke the ground clear to the back of the hole I was developing, and the tamping is intact, and the printing on it can be seen yet.

On another occasion (which I have mentioned here before), I tried the same thing when excavating for new construction work, but had another make of gelatine, and it failed to explode even the first piece past the primer, and in that case it would be dangerous to use it.

I sincerely hope that nothing I may say here may be construed as trying to excuse careless or negligent mining practices. There is positively no excuse for neglect or carelessness, such as is quoted of a miner, who, when the detonator has been pulled out of the primer, tamps up and trusts to luck; or the other who chews the end of a detonator to save pliers. He is apt to get a very bad headache, as also the other who greased his fuse first and then put his detonator on. I have another name for them other than "miners". We have enough trouble underground without a lot of pilfling restrictions being added, which really do more harm than good. They are apt to make men reckless when in a hurry, and we have to hurry nowadays to make a living. On page 296, Mr. Marquard is quoted, "Miners keep their oil in the same box as their fuse, and the fuse deteriorates." Quite so, it it touches. If the miner leaves the oil out, the tin empties itself very quickly, the natives find it handy for their hair and bodies, and the mines cannot afford two boxes, one for fuse and one for tools, etc. Is this "Safety First" or "Economy First"?

Mr. Barry states that men put more and more work and responsibility on the natives. Why is this done? I wonder if the number of places a man has to oversee, or natives to supervise, has anything to do with it?

I must admit that a great and possibly increasing number of so-called miners should never have had a blasting certificate given them, especially those who cannot speak or understand English or Dutch. If a shift boss wished to instruct them, what medium would he use? I hear that a kindergarten for miners is being advised.

So I repeat that I certainly have no desire to encourage men to run needless or unnecessary risks, but simply to show that many so-called experiments merely prove nothing.

If I did not take a live interest in my work as a miner, something more than merely making my pay out of it, I should not be a member of this Society, and I quite agree with what Prof. Wilkinson has said to-night, that it is a disgrace to the mining men on the Rand for the Society to have such poor attendances at its meetings, and slackness in paying subscriptions. If men had real heart in their work, whether miners or mine captains, metallurgists or chemists, this hall would be full and the Society would not need to fear for lack of funds.

I have to thank you for your patience, but thought that sometimes it is as well to hear a miner's point of view.

The President: Our next meeting is in September. There will be no meetings for the next two months, on account of the Winter recess.

The meeting then terminated.

Contributions and Correspondence.

GENERAL FORMULÆ FOR RESULTS OF TWO-WASH DECANTATION SLIME TREATMENT.[*]

(First Wash only precipitated.)

By H. A. White (Vice-President).

Let V = Original value of slime in dwt. per ton.

v = Undissolved gold in residue in dwt. per ton.

r = Ratio by weight of solution to solids in pulp before settlement.

M = Ratio by weight of moisture to solids in settled slime.

u = Value of precipitated solution in dwt. per ton.

Then:—

(1) Value of first wash in dwt. per ton (A)
$$= \frac{r(V - v) + u(r - M^2}{r(r - M) - M^2}.$$

(2) Value of second wash (B)
$$= \frac{MA - u(r - M)}{r}$$

(3) Value of residue = MB + v.

(4) Percentage Extraction
$$= \frac{100(V - v - MB)}{V}$$

FORMULA FOR MAKING UP CYANIDE SOLUTIONS.

By H. A. White (Vice-President).

Let T = Tons of solution to be made up.

W = KCN already in solution.

S = KCN required.

P = KCN in solid cyanide used.

X = Pounds of solid cyanide required.

Then $X = \frac{2000\ T\ (S - W)}{P}$

For sodium cyanide, 120
$$X = \frac{100\ T\ (S - W)}{6}$$

For sodium cyanide, 130
$$X = \frac{200\ T\ (S - W)}{13}$$

PERSISTENCE OF ORE IN DEPTH.

In my contribution to the discussion of the above paper I wrote that Messrs. Twelvetrees and Ward were "blind to the vast panorama of stupendous denudation spread before their eyes".[*]

This was scarcely correct, as a perusal of the full text of this monograph shows.

It should read: "Messrs. Ward and Twelvetrees, while admitting that great denudation had taken place, do not appear to have realized the enormous effect it must have had on the question of persistence of ore in depth by the removal of so many thousand feet of the upper portions of the lodes."

E. M. Weston.

Randfontein, July 5, 1915.

[*] Cp. Rand Metallurgical Practice, Vol. I., p. 294.

[*] See this *Journal*, Vol. XV., February, 1915, p. 154.

Notices and Abstracts of Articles and Papers.

CHEMISTRY

THE PREPARATION OF POTASH FROM FELSPAR AND OTHER SOURCES.—In view of the stoppage of supplies of potassium salts from Germany, attention is directed to a process developed by Ward and Wynants (Eng. Pat. 3185, Dec. 30, 1857) for the extraction of potassium salts from felspar, which at the time was found to give thoroughly satisfactory results. Felspar ground to the fineness of Portland cement was mixed with powdered fluorspar or other fluoride, the fluorine being calculated equivalent to the alkali in the rock. A mixture of chalk and lime was then mixed with the materials so that the proportion of Ca was double that of the S.O and one and a half times that of the Al. On the large scale it was found preferable to employ about 10 % excess of lime. The mixture was made into cakes and heated to a yellowish red for a few hours. The potash frit obtained was lixiviated with boiling water which extracted 90 % of the potash originally present in the rock in the form of hydroxide and carbonate. The residue might be worked up into hydraulic cement by removing the fluorine. Another possible source of potash is the spent wash from the fermentation of molasses from the beet sugar industry. The wash, neutralized by chalk, is decanted, evaporated and calcined. The crude ash, if calcination has been carefully performed, contains about 50 % of potassium salts. Lastly, the revival of the kelp industry appears by no means impracticable as a source of potash, with the additional advantage of yielding iodine.—WARD AND WYNANTS. *Chemical News*, 1914, 110, 175. *Journal Society Chem. Ind.*, 31st Oct., 1914, p. 1007. (J. W.)

STANDARD SPECIFICATIONS FOR CEMENT.—*General conditions.* All cement shall be inspected and at least 12 days allowed for inspection and testing. A bag of cement shall contain 94 lb. of cement net; and each barrel of Portland cement shall contain 4 and of natural cement 3 bags of above net weight. Cement failing to meet the 7 day requirements may be held pending the results of the 28 day tests before rejection. All tests shall be made in accordance with the methods of the American Society of Civil Engineers. Acceptance or rejection shall be based on the following requirements:—

Natural cement is defined as the finely pulverized product resulting from the calcination of an argillaceous limestone at a temperature only sufficient to expel the carbonic acid gas. *Fineness.*—Not more than 10 % by weight to be left on a No. 100 and not more than 30 % on a No. 200 sieve. *Time of setting.* At least 10 mins. for initial set and from 30 mins. to 3 hours for hard set. *Tensile strength.* The minimum requirements for briquettes 1 sq. in. in cross section are given in the table; no retrogression in strength must be shown within the periods specified. *Constancy of volume.* Pats of neat cement about 3 in. diameter, ½ in. thick at the centre, and tapering to a thin edge, shall be kept in moist air for 24 hours, and subsequently shall remain firm and hard and show no signs of distortion, checking, cracking, or disintegration when kept in air at normal temperature or in water at about 70° F. (21° C.) for at least 28 days.

Portland cement is defined as the finely pulverized product resulting from the calcination to incipient fusion of an intimate mixture of properly proportioned argillaceous and calcareous materials, and to which no addition greater than 3% has been made subsequent to calcination. *Specific gravity.* Not less than 3·10. If a sample fails to pass, a second test may be made on a sample ignited at a low red heat; the loss in weight on ignition shall not exceed 4 %. *Fineness.*—Not more than 8% on a No. 100 and not more than 25 % on a No. 200 sieve. *Time of setting.* At least 30 mins. for initial set and from 1 to 10 hours for hard set. *Tensile strength.*—See table; same conditions as for natural cement. *Constancy of volume.* Same as for natural cement, with the addition that a third pat is exposed in an atmosphere of steam, above boiling water, in a loosely closed vessel for 5 hours. *Sulphuric acid and magnesia.* Not more than 1·75 % SO and not more than 4% MgO.—

No.	Tensile Strength of:			
	Natural Cement.		Portland Cement.	
	Neat.	Mixed with 3 parts of Standard Ottawa Sand.	Neat.	Mixed with 3 parts of Standard Ottawa Sand.
	lb.	lb.	lb.	lb.
24 hours (1 in moist air	75	—	175	—
7 days (1 in moist air, 6 in water)	150	50	500	200
28 days (1 in moist air, 27 in water)	250	125	600	275

(AMER. SOC. TESTING MATERIALS, Year book, 1914, 290-317. *Journ. Soc. Chem. Ind.*, Jan. 30, 1915, p. 81. (J. A. W.)

FERROCYANIDES.—"The soluble ferricyanides are produced from the ferrocyanide by oxidation, and the potassium salt is usually produced by the method first proposed by Leopold Gmelin, by passing chlorine through a solution of potassium ferrocyanide until the liquid yields no blue reaction when tested with a ferric salt.

This method, though fairly satisfactory for the preparation of the potassium salt, is inapplicable for the preparation of the sodium, ammonium, or alkaline earth metal salts, chiefly on account of the greater solubility of the ferricyanides of these bases, and the great difficulty, and in many cases the impossibility, of separating them in a pure condition from the associated chloride.

This difficulty may be easily overcome by the now suggested process, whereby any soluble ferrocyanide of the alkali, or alkaline earth metals, may be readily converted into a solution of the corresponding ferricyanide, entirely free from any other salts, and in a neutral condition.

To 100 gm. of sodium ferrocyanide dissolved in 1,000 cc. of water add a solution of manganese chloride, or sulphate, until the ferrocyanide is completely precipitated; the mixture, which contains white manganese ferrocyanide of the formula $Mn_2Na_2[Fe(CN)_6]$, in suspension, is raised to the boil and nitric acid added until oxidation is complete; by this means the white manganous ferrocyanide is converted into the brown manganous ferricyanide; the mixture is then thrown on a vacuum filter and thoroughly washed; this is not

a lengthy operation as the precipitate filters fairly well. The washed paste is then added to a solution of any ferrocyanide and agitated, when the following reaction takes place, *e.g.*, with calcium ferrocyanide—

$$2Mn_3[Fe'''(CN_6)]_2+3Ca''_2Fe(CN)_6 = 2Ca_3[Fe'''(CN)_6]_2+3Mn_2Fe''(CN)_6.$$

The addition of the brown manganous ferricyanide must be continued until it ceases to turn white on agitation and some of the brown compound remains in excess; the liquor is then filtered off and the pure ferricyanide solution evaporated under reduced pressure; the washed precipitate of white manganous ferrocyanide may be mixed with more water and reoxidised with nitric acid and thus be used for the conversion of a fresh quantity of ferrocyanide.

This reaction takes place in the cold for all soluble salts except the ammonium salt; in this case the conversion only takes place on heating; this salt is, however, best prepared by precipitating a solution of the barium salt with sulphate of ammonia.

The analysis of a ferricyanide in solution is usually accomplished by first reducing the ferricyanide to ferrocyanide and then estimating the latter body so formed by (*a*) *N* 10 permanganate solution; (*b*) titration with a standard solution of copper or zinc salt; (*c*) or by titration of the cyanide obtained by distilling the ferrocyanide with dilute sulphuric acid and a little cuprous chloride and absorbing the evolved hydrocyanic acid in a caustic alkali solution (*Journ. Soc. Chem. Ind.*, p. 315, Vol. 31).

If the ferricyanide is free from other cyanogen compounds it may be estimated by the latter method without previous reduction if a larger amount of cuprous chloride is added.

If the ferricyanide exists in the presence of ferrocyanide but in the absence of other oxidisable salts, the ferricyanide may be estimated by first titrating the ferrocyanide with *N* 10 permanganate solution, and then reducing the total ferricyanide by warming it with ferrous hydroxide in presence of an excess of alkali; the acidified filtrate is again titrated with permanganate, the difference between these two ferrocyanide readings will correspond with the proportion of ferricyanide originally present.

Instead of the ferrous hydroxide mentioned above for the reduction of the ferricyanide, manganous, or cobaltous hydroxide, may be substituted with greater advantage, and, if only the ferricyanide is to be determined, it may be reduced to ferrocyanide in the cold by the addition of cobaltous hydroxide and excess of alkali with agitation, black cobalto cobaltic hydroxide being formed thus :—

$$2K_3Fe(CN)_6+2KOH+3Co_2O = 2K_4Fe(CN)_6+Co_3O_4+H_2O.$$

The black hydroxide is filtered off, thoroughly washed and distilled with an excess of hydrochloric acid, and the evolved chlorine absorbed in potassium iodide solution, which is finally titrated with *N*/10 sodium thiosulphate solution. These reactions may be expressed :—

(1.) $Co_3O_4+8HCl=3CoCl_2+4H_2O+Cl_2$.

(2.) $Cl_2+2KI=2KCl+I_2$.

Each cc. of sodium thiosulphate solution required is equivalent to 0·03292 gm. potassium ferricyanides.

If the ferricyanide exists in the presence of ferrocyanide and thiocyanate or other oxidizable salt, the two iron cyanogen compounds may be separated from the other salts and from each other by taking advantage of the different properties of their zinc salts.

The solution containing not more than 1 gm. of the mixed iron cyanogen compounds is acidified with dilute hydrochloric or sulphuric acid and the two compounds separated together by the addition of an excess of a soluble zinc salt, and the precipitate filtered and washed; the precipitate is then digested with an excess of sodium carbonate, when the whole of the ferricyanide of zinc is decomposed and sodium ferricyanide passes into solution, but the ferrocyanide of zinc will be unattacked; the mixture is then filtered and washed; the filtrate which contains the whole of the ferricyanide is estimated as cyanide by distilling with dilute sulphuric acid and cuprous chloride into caustic soda solution; the precipitate on the paper of basic ferrocyanide and carbonate of zinc is then also put into the distilling flask together with the paper and dilute sulphuric acid, and cuprous chloride added and distilled in the manner described; 1 cc. of *N* 10 silver nitrate used to titrate the cyanide obtained from the filtrate will equal 0·01097 gm. $K_3Fe(CN)_6$ and from the insoluble matter = 0·0122 gm. $K_4Fe(CN)_6$.

Ammonia solution may be substituted for the sodium carbonate in the separation of the zinc salts, zinc ferricyanide being readily dissolved by ammonia while the ferrocyanide is insoluble, but this method offers no advantage and the precipitate is difficult to filter.

Soluble ferrocyanides cannot be separated from ferricyanides by means of ferric chloride as is frequently stated, for, although a pure ferricyanide solution is not precipitated by ferric salts, yet in the presence of ferrocyanides considerable proportions are precipitated as a green ferric ferrocyanide.

Ferrocyanides may be separated from ferricyanides by shaking the solution, made alkaline with carbonate of sodium or potassium, with an excess of freshly precipitated zinc carbonate; the whole of the ferrocyanide is precipitated while the ferricyanide remains in solution.—H. E. Williams—*Chemical World*, Nov., 1914, p. 298. (J. A. W.)

Specific Gravity Separation Applied to the Analysis of Mixing Explosives.—"Chemical analysis of a mixture of salts indicates the quantities of the various ions present, but throws no light on the manner in which these were combined in the original mixture. After removal of nitroglycerine and oily matter by means of ether the finely powdered and dried salts can be roughly separated by treatment in a separating funnel with mixtures of chloroform (sp. gr. 1·49) and bromoform (2·83) of specific gravity between those of the salts expected, most of which, as shown in a table, lie between 1·5 and 2·5."—C. G. Storm and A. L. Hyde.—*Journ. Soc. Chem. Ind.*, Dec. 31, 1914, p. 1227. (J. A. W.)

Quick Method to Precipitate Tin.—"The tin (0·2–0·3 gm.) is separated as sulphide, dissolved in ammonium sulphide (16 cc.), 20 cc. of sodium sulphite solution (40%) added, the liquid diluted to 120 cc., and electrolysed at 60° C. with a current of 5·5 amp. at exactly 4 volts; cylindrical gauze electrodes are employed, one within the other, and the electrolyte agitated by a stirrer making 600 revolutions per min., the deposition of the metal being affected in 25 mins. Before use, the platinum cathode is either coated with copper and tin in

succession or with cadmium alone. The accuracy of the process is sufficient for technical analysis, but is impaired by the impurities present in ordinary ammonia solution; for this reason the ammonium sulphide used is prepared by saturating with hydrogen sulphide a solution of ammonia (sp. gr. 0·91) obtained by dissolving the gas (compressed) in water. Sodium sulphite is employed to render the tin solution colourless." L. HUMPHRYVILLE. *Journ. Soc. Chem. Ind.*, Dec. 31, 1914, p. 1228. (J. A. W.)

SILICATE ANALYSIS. "The $Na_2CO_3 + K_2CO_3$ method used in breaking down silicates is not the most desirable because of the necessity of using a Pt crucible and the attendant dangers in case of the presence of easily reducible metals like Pb, Zn, Sb, Bi. The addition of small amounts of saltpeter is undesirable, in overcoming the dangers of reduction, because it makes the crucible brittle. Mention is also made of the difficulties occasioned by sulphur in such analysis. The author suggests, as a substitute for the above method, the use of NaOH as the fluxing medium, and says that a Ag crucible can be used very satisfactorily if that material be employed. One gm. of silicate is used to 4 to 5 gm. of the NaOH." E. BUTTNER, *Silikat Z.*, 2, 149-51. *Chemical Abstracts*, Dec. 10, 1914, p. 3765. (J. G.)

REDUCTION OF FERRIC SULPHATE IN ACID SOLUTION BY MEANS OF CADMIUM AMALGAM, ETC. (ANALYSIS OF MINE WATERS). "Free sulphuric acid in mine water cannot be determined directly owing to gradual hydrolysis of the ferric salt during the titration. Electrolytic reduction of the ferric salt invariably resulted in the liberation of measurable amounts of hydrogen; this source of error was avoided by reducing with cadmium amalgam in an atmosphere of hydrogen. A portion of the mine water (50 cc.) was first titrated with potassium permanganate (1 cc. = 0·005258 gm. Fe) for ferrous iron. A small separating funnel and a three-way tap were sealed to the central tube and side tube, respectively, of a Drexel gas washing bottle, and after introducing a convenient amount of the mine water and exhausting, the bottle was well shaken; hydrogen was admitted and the bottle again shaken till equilibrium resulted. The process was repeated twice, the amalgam run in, and the bottle shaken vigorously until the attainment of complete reduction, *i.e.*, till a drop of the solution gave no colour with potassium thiocyanate (about two minutes). One aliquot part of the solution (50 cc.) was then titrated with *N*/10 sodium carbonate for free acid, and another portion (50 cc.) with permanganate for total iron. Methyl orange indicator was employed in the former operation, the end-point being ascertained by comparison with a standard neutral tint. The absolute end-point of the standard was ascertained by conductivity measurements made during titration of a similar solution in presence of methyl orange; the resulting tint was permanent when the solution was enclosed in a sealed bulb of clean glass. An artificial "mine water" containing 0·429 gm. of ferric iron, 0·774 gm. of ferrous iron, and 3·85 gm. of sulphuric acid per litre, also traces of aluminium, calcium, magnesium (as sulphates) and sodium (as chloride) gave, respectively on analysis by the above method 0·427 gm., 0·769 gm., and 3·825 gm. Similar solutions of ferric iron in sulphuric or hydrochloric acids, after reduction (4 to 5 minutes) in the same way with cadmium amalgam gave extremely concordant results upon subsequent titration with permanganate and bichromate, respectively. In these cases the preliminary expulsion of oxygen from the solutions was unnecessary, but the air in the apparatus was displaced by carbon dioxide. Zinc amalgam effected a rapid reduction (0·5 minute), but the results were vitiated by the simultaneous displacement of appreciable amounts of iron from the solutions."—J. H. CAPPS and O. W. BOLES, *J. Phys. Chem.*, 1915, *19*, 65-75. *Journ. Soc. Chem. Ind.*, Feb. 27, 1915, p. 175. (J. A. W.)

ANALYSIS OF PLATINUM. "Schemes are given for the qualitative and quantitative examination of nominally pure and commercial platinum, based on a critical review of known methods and the results of special experimental work. The qualitative examination permits of the detection of 1 mgm. each of Pd, Pt, Rh, Ru, Ir, Au, Ag, Cu, and Fe in a chloride solution.

In the quantitative analysis of commercial platinum the traces of ruthenium present may be disregarded. The approximate purity of the metal is first ascertained by dissolving 0·1 gm. in *aqua regia*, evaporating with sodium chloride, dissolving the residue in 10 cc. of water, heating with about 0·02 gm. of sodium carbonate, treating the cooled solution with 1 or 2 drops of sodium hypochlorite or hypobromite solution, and again warming. If the solution retains its orange yellow colour and no precipitate is formed, the metal may be regarded as "nominally pure" platinum, but if the colour darkens and a dark blue precipitate gradually forms, the metal is classed as impure or "commercial" platinum.

Group Precipitation—

I. The solution is evaporated nearly to dryness, diluted with 2 cc. of water and filtered.

Precipitate: AgCl.

II. The filtrate from I is saturated with NH_4Cl, filtered after 1 hour, and the precipitate washed with NH_4Cl solution.

Precipitate: double chlorides of Pt, Ir, Ru.

III. The filtrate from II. is evaporated to dryness with 3 cc. of dilute HNO_3, the residue treated with dilute HNO_3 so as to obtain a solution nearly saturated with NH_4Cl, filtered, and the residue washed with a very small quantity of a solution of $HNO_3 + NH_4Cl$.

Precipitate: double salts of Pd, Ir, and Ru.

IV. The filtrate from III. is evaporated to dryness with dilute HCl, the residue dissolved in 5 cc. of water with addition of 1 drop of dilute HCl, and the solution rapidly saturated with H_2S at about 18° C. The black precipitate is filtered off after 10 mins., and washed with water.

Precipitate: sulphides of Au and Cu.

V. The filtrate (6—8 cc.) from IV. is warmed and treated with H_2S for ½ hour, the excess of H_2S boiled off, and the brown precipitate filtered off and washed with water.

Precipitate: rhodium sulphide.

VI. The filtrate from V. is boiled, oxidized with HNO_3, treated with ammonia, and the precipitate filtered off. The evaporation residue from the filtrate should consist only of ammonium salts and be completely volatilized on heating.

Secondary Separations—

(6.) The precipitate is treated on the filter with ammonia, and HNO_3 added to the filtrate. White precipitate indicates Ag. The dark residue on the filter is transferred to (5).

(5.) The precipitate from II., together with residues from (1) to (6), is heated in hydrogen, ultimately to incandescence, the reduced metal treated with diluted *aqua regia*, and the solution evaporated, Pt being detected by formation of the yellow double ammonium chloride. The insoluble residue is ignited with 0·3 gm. $NaNO_3$ in a gold or silver spoon. Ru dissolves in the melt, colouring it yellow, whilst Ir remains as a blue residue.

(4.) The precipitate from III. is evaporated with HCl, the residue treated with 3 cc. of water, and the solution saturated cold with H_2S. The sulphide precipitate is evaporated with *aqua regia*, and the solution of the evaporation residue tested for Pd with $Hg(CN)_2$. Any residues containing Ir are transferred to (5).

(3.) The sulphide precipitate is dissolved in dilute *aqua regia*, the solution evaporated, the residue dissolved in dilute HCl, and the solution extrated with ether. If the ether layer is coloured yellow and gives a precipitate with SO_2, Au is indicated.

The aqueous layer is treated with H_2S, the precipitate filtered off, washed, ignited, and treated with dilute HNO_3. Cu is detected by a blue coloration of the solution on adding excess of ammonia.

Any residue insoluble in HNO_3 is transferred to (5).

(2.) The sulphide precipitate is dissolved from the filter in hot dilute *aqua regia*, the solution evaporated, and the residue warmed with concentrated HCl. A rose-red solution which gives a yellow precipitate on warming with a 1% solution of ethylmercaptan indicates Rh.

The filtrate may contain traces of Ir.

(1.) The precipitate from VI. is ignited, dissolved in HCl, and Fe detected by the Prussian blue reaction.

Any insoluble residue is transferred to (5).

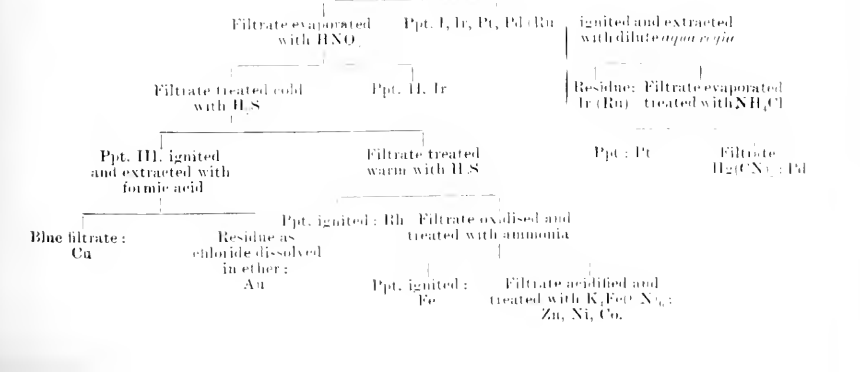
Commercial platinum.—10 gm. is dissolved in *aqua regia* in a flask of quartz glass and the solution evaporated with 6 gm. of sodium chloride. The residue is evaporated several times with water, then heated for 1 hour at 120° C., treated with about 100 cc. of water, and the solution filtered. The small quantity of residue may contain gold, silicates, iron salts, silver chloride, etc., and is analysed separately. The solution is diluted to 500 cc., made faintly alkaline with 0·8 gm. of sodium bicarbonate, heated to boiling, cooled, and again heated with a solution of 0·8 gm. of sodium bicarbonate in 20 cc. of saturated bromine water; the solution should remain faintly alkaline. When a test portion of the yellow solution no longer gives a dark coloration with hypobromite solution, the whole is filtered (after addition of alcohol, if necessary, to coagulate the precipitate), and the black precipitate washed with water, the filtrate being heated with alcohol and again filtered through a separate filter. The solution contains nearly the whole of the platinum as platinum-sodium chloride. The precipitate consisting mainly of the oxides of the foreign metals, together with some platinum, is analysed according to the following scheme:—

Nominally pure platinum.—50 gm. is converted as described above, into the platinum-sodium chloride, which is recrystallized several times from a 1% solution of sodium carbonate. The mother liquors, together with the solution obtained by treating the small quantity of insoluble matter left from the first operation, are evaporated first with *aqua regia*, then with hydrochloric acid, and the residue heated to 150° C., treated with water, and filtered. The solution may be analysed as described under commercial platinum, or it may be faintly acidified, treated with ammonium chloride, and filtered. The precipitate consists of platinum-ammonium chloride contaminated by a little iridium. The filtrate is treated warm with hydrogen sulphide for several hours, the sulphides ignited and the residue dissolved in *aqua regia* (the precipitate is dissolved directly in *aqua regia* if lead or other volatile metals are present). The filtered solution is analysed according to the scheme given above. All precipitates of platinum ammonium chloride must be treated for the separation of iridium, which is effected by reduction with hydrazine in strong hydrochloric acid solution, the platinum being precipitated and the iridium left dissolved.

The Ir content of commercial platinum can be rapidly estimated by dissolving 0·01 gm. in *aqua regia*, evaporating, dissolving the residue in concentrated hydrochloric acid to 100 cc., and comparing the colour with that of a standard solution of Pt and Ir in concentrated hydrochloric acid.

In the following table are given analyses of the

Chloride Solutions of the Oxides.



ompt ties continued in three platinum muffles: I and II behaved well in use, but III was strongly corroded, with formation of soot.

	I.	II	III.
Ir	2·2	2·23	1·61
Pd	—	0·43	0·15
Au	0·30	0·3	—
Rh	trace	trace	0·15
Cu	0·13	0·18	0·09
Fe	0·13	0·12	0·11
Ag	—	—	trace
Ni	trace	trace	trace
Undetermined	—	—	0·1
Total	3·0	2·96	2·25

— F. Mylius and A. Mazzucchelli, *Z. anorg. Chem.*, 1914, *89*, 138. *Journ. Soc. Chem. Ind.*, Jan. 30, 1915, p. 84. (J. A. W.)

Standard Specifications for Methods of Chemical Analysis for Plain Carbon Steel. — "*Carbon* is determined by direct combustion in oxygen at 950°—1100° C., the heat or container for the metal being lined with alundum (90 mesh or finer). The oxygen employed (of not less than 97% purity) is passed over a heated catalyst (*e.g.* CuO) and through a purifying train before use. The carbon dioxide produced is absorbed in potassium hydroxide or soda lime and weighed as usual, or in a solution of barium hydroxide contained in a Meyer tube, the precipitated barium carbonate filtered off out of contact with the atmosphere, washed, dissolved in standard acid, and the excess of acid titrated with standard alkali; in the latter case no precautions are necessary to prevent access of water vapour or sulphur trioxide to the absorption tube.

Phosphorus. — For routine work the steel is dissolved in nitric acid, potassium permanganate is added, then ammonium bisulphite to dissolve precipitated manganese oxide, the solution (at 80° C.) agitated with molybdate solution for 3—4 mins., the precipitate filtered off, dissolved in standard sodium hydroxide and titrated with standard nitric acid.

Sulphur. — In the oxidation method, the steel is dissolved in *aqua regia* and silica separated by two evaporations to dryness; the barium sulphate, precipitated by the addition of barium chloride to the cold hydrochloric acid solution, is allowed to stand for at least 24 hours, then filtered off, washed first with a hot solution containing 10 cc. HCl and 1 gm. BaCl per litre, until free from iron, and then with hot water. The washings are collected separately and evaporated to recover any dissolved barium sulphate. A blank determination is made on all the reagents used. In the evolution titration method, 5 gm. of the steel are dissolved in 80 cc. of hydrochloric acid (1 : 1) in a flask (480 cc.) fitted with a stopper and thistle funnel and having a straight side tube with which is connected a vertical delivery tube extending to the bottom of a tall beaker containing 150 cc. of water and 10 cc. of ammoniacal cadmium chloride solution (60 cc. 10 gm., water 100 cc., ammonia, sp. gr. 0·90, 100 cc.). The flask and contents are heated until solution of the steel is complete, extremely slow or rapid evolution of gas being avoided, after which the solution is boiled for ½ min.; the contents of the beaker are then acidified with 40 cc. of hydrochloric acid (1 : 1), starch solution is added and the liquid titrated with a solution of potassium iodate (KIO 1·116 gm., KI 12 gm., water 1000 cc.) previously standardized against a steel of known sulphur content.

Silicon. — The steel is dissolved in a mixture of nitric and sulphuric acids, or in the latter alone, the solution evaporated till sulphur trioxide is evolved and the operation concluded as usual, the silica obtained being evaporated with hydrofluoric acid. A blank determination is made on the reagents employed.

Copper. — Solution is effected in sulphuric acid (1 : 4), the liquid diluted, heated, saturated with hydrogen sulphide, filtered, and the precipitate washed with 1% sulphuric acid containing hydrogen sulphide until free from iron; the filter and contents are incinerated, the residue fused with sodium bisulphate, the melt dissolved in hot water and the solution filtered, the copper being then determined colorimetrically (with ammonia or potassium ferrocyanide) or by electrolysis.

Nickel. — The dimethylglyoxime method is employed, precipitation from the ferric solution being effected in the presence of tartaric acid. The precipitate is either dried at 110°—120° C. and weighed as such; or dissolved in nitric acid, the solution boiled with ammonium persulphate, cooled, made distinctly ammoniacal, silver nitrate and potassium iodide added, and the liquid titrated to a faint turbidity with a solution of potassium cyanide (2·29 gm. per litre) previously standardized against a steel of known nickel content.

Chromium. — 5 gm. of the steel are dissolved in 50 cc. of hydrochloric acid (1 : 1), the solution nearly neutralized with saturated solution of sodium carbonate and then completely with barium carbonate (10 gm. suspended in 100 cc. of water) of which about 1 gm. excess is used. The precipitate obtained by boiling the liquid in a covered flask for 10—15 mins., is filtered off, washed, and filter and contents incinerated, the residue fused for 10 mins. with a mixture of sodium carbonate (5 gm.) and potassium nitrate (0·25 gm.), the melt dissolved in water, 2 cc. of hydrogen peroxide (3% solution) added, the liquid boiled and filtered. Chromium is determined in the filtrate (if strong yellow) by boiling thoroughly, acidifying with sulphuric acid, adding excess of standard ferrous sulphate and titrating the excess with standard potassium permanganate; or (if pale yellow) by colour comparison with a standard solution of sodium chromate. Amer. Soc. Testing Materials, Yearbook, 1914, 169-176 and 181-192.— *Journ. Soc. Chem. Ind.*, Jan. 15, 1915, p. 32. (J. A. W.)

Methyl Red as an Indicator. — "Methyl red gives a sharper end point than methyl orange when the two indicators are tested in water with acid or alkali, and its sensitiveness, unlike that of methyl orange, is not greatly affected by the presence of neutral salts, such as sodium chloride or sulphate. When methyl red is used as the indicator in the titration of carbonates with acids, the solution must be boiled after each addition of the acid in order to expel free carbon dioxide. In general, methyl red most closely resembles litmus in its indications, but is to be preferred to the latter owing to its more decided change in colour with alkalis and acids. R. T. Thomson. *Analyst*, 1914, *39*, 518-519. *Journ. Soc. Chem. Ind.*, Jan. 15, 1915, p. 51. (J. A. W.)

STANDARD SPECIFICATIONS FOR AND ANALYSIS OF SPELTER.—"Virgin spelter (not re-worked metal) is considered in four grades, the maximum allowable impurities in which are:—

Grade.	Maximum Impurities Per Cent.				
	Pb	Fe	Cd	Al	Pb+Fe+Cd
A. High Grade	0·07	0·03	0·05	none	0·10
B. Intermediate	0·20	0·03	0·50	none	0·50
C. Brass Special	0·75	0·04	0·75	none	1·20
D. Prime Western	1·50	0·08	—	—	—

To obtain the sample for analysis, at least 10 slabs from each car-load shall be sawn completely across and the sawdust collected, or the slabs drilled through and the drillings cut into short lengths; particles of iron are removed from the sample by means of a magnet before analysis: *Lead.* Sufficient to be taken to contain at least 0·01 gm. of this impurity. *Iron.* At least 10 gm. of Prime Western, or 25 gm. of any other grade, to be completely dissolved in acid, the iron precipitated as ferric hydroxide, re-dissolved and determined volumetrically, as usual. *Cadmium.* 25 gm. to be treated with 330 cc. of hydrochloric acid (1:5), allowed to stand over night and filtered (filtrate discarded); the insoluble residue (containing about 5% of the zinc) dissolved in nitric acid, 10 cc. of sulphuric acid added, the liquid evaporated till sulphur trioxide is evolved, diluted and the lead sulphate removed by filtration; the filtrate is diluted to 500 cc., 5 gm. of ammonium chloride is added, a slow stream of hydrogen sulphide passed through for 1 hour, the liquid allowed to stand for 5 hours and filtered; the precipitate washed with hot water, re-dissolved in 60 cc. of boiling dilute sulphuric acid (1:5 by vol.), the solution filtered, diluted to 100 cc. and saturated with hydrogen sulphide as before, the precipitated cadmium sulphide to be weighed as such or dissolved in hydrochloric acid and titrated with potassium ferrocyanide."—AMER. SOC. TESTING MATERIALS, Year-book, 1914, 284-287. —*Journ. Soc. Chem. Ind.*, Jan. 15, 1915, p. 33. (J. A. W.)

VOLUMETRIC DETERMINATION OF MERCURY.—"A rapid and accurate method for the determination of mercury is based on the precipitation of mercuric oxide by excess of sodium hydroxide, re-solution of the precipitate in a measured excess of standard potassium cyanide solution, and titration of the excess of the latter with a standard silver nitrate solution. Potassium iodide added to the solution serves as indicator, and the end point is marked by the appearance of a yellowish turbidity due to silver iodide. The following reactions are involved:—

$HgO+4KCN+H_2O=K_2Hg(CN)_4+2KOH$;

$K_2Hg(CN)_4+AgNO_3=$

$Hg(CN)_2+KAg(CN)_2+KNO_3$.

In the case of samples containing mercurous oxide, the mercury can usually be completely dissolved by a preliminary treatment with saturated bromine water. The excess of bromine must be completely removed by boiling before adding the sodium hydroxide. Other metals which react with cyanide, such as copper, nickel, or zinc, can be readily eliminated by utilizing the insolubility of mercury sulphide in moderately concentrated nitric acid. The presence of iron obscures the end point in the titration with silver nitrate; it is removed by transforming into the ferric state by adding nitric acid to the solution and boiling before neutralizing with sodium hydroxide. On adding the cyanide, ferric hydroxide is precipitated and removed by filtration. To obtain accurate results a large excess of potassium cyanide must be avoided, but a moderate excess of alkali does not affect the results, and the loss of mercury salt on boiling the solution for 30 seconds is negligible. J. E. CLENNELL, *Eng. and Mining Journal*, Oct. 31, 1914, 787-789. —*Journ. Soc. Chem. Ind.*, Dec. 15, 1914, p. 1173. (J. A. W.)

URANIUM ORES IN MADAGASCAR.—"A mineral has been discovered at Fiadanana, Madagascar, externally resembling euxenite and containing 12% to 40% UO_3. Its radioactivity, tested by a aluminium foil electroscope, is about double that of the pure oxide, thus suggesting an important source of radium."—C. GROSSMANN, *Comptes rend.*, 1914, *159*, 777.—*Journ. Soc. Chem. Ind.*, Jan. 15, 1915, p. 34. (J. A. W.)

PRECIPITATION OF ZINC AND MANGANESE WITH AMMONIUM SULPHIDE.—"To obtain precipitates which will not run through the filter paper, the author recommends the following procedure: Add to the solution, the latter being neutral or neutralized with NH_4OH, and containing about 0·5 gm. of Zn or Mn in 100 to 200 cc., a large excess of NH_4OH (about 10 to 20 cc. of 25% NH_4OH). Warm to 60° to 80° and after adding $(NH_4)_2S$ until the liquid is yellow, heat until the precipitate coagulates. This will ordinarily require a few minutes. Decant and filter. A precipitate of MnS is obtained which is granular, olive green, and free from H₂O. ZnS so precipitated is grey white and flocculent. After washing with H₂O containing $(NH_4)_2S$, the precipitate is ignited and subsequently heated with a large excess of H_2SO_4. $ZnSO_4$ is heated to a dull red heat, but $MnSO_4$ does not require this precaution."—F. SELLIGMANN, *Z. anal. Chem.*, *53*, 594-5, 1914. *Chemical Abstracts*, Jan., 10, 1915, p. 40. (J. G.)

ATOMIC WEIGHT OF LEAD.—"In the *Journal of the American Chemical Society* for July, 1914, is a very interesting paper reporting an investigation of the atomic weight of lead derived from various sources in various parts of the world. The following were the final results:

North Carolina uraninite	206·40
Joachimsthal pitchblende (2 samples)	206·57
Colorado carnotite (2 samples)	206·55
Ceylonese thorianite (2 samples)	206·82
English pitchblende	206·86
Common lead	207·15

It is to be observed that these results agree with those of Curie in showing the atomic weight of the lead from the uranium minerals to be much lower than that of ordinary lead.

The ultraviolet spectrum of a radioactive sample of lead was compared with that of ordinary lead. No difference was found, except for a trace of copper too small to affect the result, and a negligible trace of silver. The inference appears to be that radioactive lead contains a mixture of some substance different from ordinary lead, and very difficult to separate by chemical means. This substance cannot be identified by the ultra violet spectrum, either because it has the same spectrum as lead or because it has no ultraviolet spectrum, or because it is masked by lead. "This amazing out-

ome," say the authors, "is contrary to Harvard experience with several other elements, notably copper, silver, iron, sodium, and chlorine, each of which seems to give a constant atomic weight, no matter what the geographical source may have been." T. W. RICHARDS and M. E. LEMBERT. *Engineering and Mining Journal*, Nov. 7, 1914, p. 823. (G. H. S.)

ACTION OF ACID MINE WATER ON INSULATION OF ELECTRIC MOTORS.—"Four different types of insulation were used in the tests: (1) Wire (0·162 in. diameter) covered with eight layers of varnished cotton cambric (0·016 in. thick), and one layer of cotton braid (0·056 in. thick), saturated with a compostion. (2) Wire (0·325 in. diameter); rubber layer (0·0925 in. thick), containing 30% Para; cotton tape (single layer); cotton braid with weather proof finish (2 layers 0·035 in. and 0·033 in. thick). (3) Wire (0·162 in. diameter); varnished cambric, 11 layers (0·116 in. thick); cotton braid with weather proof finish (2 layers, 0·029 in. and 0·04 in. thick). (4) Lead sheathed telephone cable, composed of 5 pairs of paper covered wires, surrounded by lead 3·32 in. thick. The cables were sprinkled with 0·45% sulphuric acid or a 1·2% solution of ferrous sulphate in this acid three times daily for the first 4 weeks, once daily for 10 weeks, twice daily for 76 weeks, and subsequently once daily, and, beginning eight months after the tests were started, the insulation resistance was measured at intervals over a period of two years. The rubber insulation showed no apparent deterioration, and the insulation resistance, although only about half the original, was still very good. The telephone cable also gave good results. The cambric covered wires, (1) and (3), had practically lost their insulation, mainly through the penetration of the liquid between the layers." H. H. CLARK and L. C. JESSY, *U.S. Bureau of Mines, Techn. Paper* 58, 1913, 24 pp. *Journ. Soc. Chem. Ind.*, Jan. 30, 1915, p. 90. (J. A. W.)

NEW VOLUMETRIC METHODS FOR THE DETERMINATION OF COBALT IN THE PRESENCE OF NICKEL.—"*First method.*—Add to the Co solution in flask (volume of solution should not exceed 100 cc., weight of Co, 0·2 gm.) 15 cc. of a 10% H_2O_2 solution, 30 cc. $NaHCO_3$ solution (50 gm. in 1 litre of H_2O) and 30 cc. of 10% NaOH solution; shake vigorously till the green compound first formed is decomposed, heat 20 to 25 minutes on a water-bath till excess H_2O_2 is destroyed, cool, dissolve the precipitate in KI and dilute if SO_4, and titrate the I with $Na_2S_2O_3$. (1 c.c. I = 1 Co.) *Second method*: To the neutral Co solution in a 250 cc. wide-neck flask (volume of solution not over 50 cc., weight of Co, 0·1 to 0·3 gm.) add 5 cc. of NH$_4$Cl solution (166 gm. in a litre of H_2O), 5 cc. of NH$_4$OH and 7 cc. to 10 cc. of 10% H_2O_2 solution; heat 5 to 10 minutes till evolution of O_2 is brisk, and then add 25 cc. of 25% KOH solution, boil till colour of NH_3 has disappeared (about 10 minutes), keeping volume fairly constant by additions of H_2O. Cool, add KI and acidify, shake till precipitate is dissolved (requires 10 to 15 minutes) and titrate with $Na_2S_2O_3$. If Ni is present, the amounts of reagents are doubled. In the presence of Ni, the final precipitate dissolves readily after adding KI and acid. Excellent results reported by both methods when working with pure salts, even in presence of much Ni (Cl. C. I. S. 2536)." A. MIOLATI, *Z. anal. Chem.*, 53, 537-41, 1914. *Chemical Abstracts*, Jan. 10, 1915, p. 140. (J. G.)

METALLURGY.

OXY-ACETYLENE WELDS.—"The point of this paper is that it emphasises the fact that a weld of this nature does not show up as satisfactorily under dynamic as it does under static tests, and therefore in practice may prove to be unsafe in certain critical situations. In the first test round bars of ⅝ in. diameter of 0·25% carbon mild steel were used and tested under a variety of conditions as to previous treatment. The table gives comparative results for maximum stress and elongation, it being found impracticable to measure the contracted area of section at point of fracture as the fracture in most cases was very ragged and uneven. The results are for 18 static tensile tests, the mean results for the original material are entered as 100, and the results for welds as percentages of 100.

	Maximum Stress.	Elongation.
Two unwelded bars ...	100	100
Five welds untreated ...	84·6	22
Four welds hammered ...	75·3	26·9
Two welds hammered and then reheated to 800° C. ...	80·8	32·1
Four welds reheated to 800° C.	82·5	45·3
Three welds quenched in water, then reheated to 800° C.	94·5	36·2

Considering untreated welds it may be said that the tensile strength is about four-fifths of that of the original metal, since the maximum percentage, as shown in the above table, is 84·6 and the minimum 79·6. Ductility, however, varies largely, with a maximum of 22 and a minimum of 9·3%. Hammering seems to reduce tensile strength somewhat and increase ductility. Reheating after hammering increases both tensile strength and ductility. Reheating without hammering also increases both properties, the increase in ductility being marked. Quenching in water and then reheating increases the strength to nearly original strength and increases ductility, though not to the limit effected by reheating only. It will be seen, however, that on the whole the ductility of a weld always remains well below that of the original metal. The next experiment was with bars 6½ in. long and ½ in. diameter, subjected to impact in a Stanton machine. Analysing results as before we have the following table:—

	Means.
Two unwelded bars	100
Six welds untreated	54·1
Three welds hammered	84·4
Three welds reheated to 800° C.	65·3
Three welds reheated to 900° C. ...	59·3
Three welds quenched in water, then reheated to 800° C.	49·6

We thus see that under impact an average untreated weld has half the strength of the original metal, that hammering is the best form of treatment for a weld, and that whereas quenching and reheating treatment, produced the best results under static tensile test, it produced the poorest results under impact. In connection with hammering, however, it must be borne in mind that the metal should be reheated to a full red-heat—800° C.—in order to remove strains of brittleness which might have been set up, because the heating during welding is extremely local, the original metal in the vicinity of the weld may be at a comparatively low temperature, and the action of hammering at this low temperature is liable to render the original metal brittle. All the fractures in this test took place in the weld itself because a notch had been cut in the weld to bring about this result; but in actual practice fracture

commonly takes place in the original metal near the weld and not in the weld itself; and this fact is often pointed to as proving the superior strength of the weld. As a matter of fact it is due to the metal near the weld being rendered brittle by the action of welding. The ends of the pieces to be joined are heated to the point of fusion, and a little beyond the metal is heated to a temperature sufficient to make it brittle. A section examined under the microscope will indicate a structure in the weld itself resembling cast-iron, a structure just beyond exhibiting a coarse mesh-work of overheated brittle metal.

Impact tests upon plates of different thickness have shown that the thicker the plate the less reliable the weld and the greater the reduction in strength. There is also less improvement in annealing thick material; and in consequence of the highly local heating action of acetylene there is considerable danger of contraction stresses being set up in thick metal. When metal is very thin an acetylene welded joint may be as strong as the original metal, in fact in one case of ⅛ in. metal tested the joint actually proved stronger, nor was there any deterioration of adjacent metal. The danger with thick plates is that a larger portion of the neighbouring metal becomes overheated, the cooling is slower, and the danger of contraction stresses being set up increases. It is said that the Directors of Lloyd's Register have had great pressure put upon them to accept repairs by this process, but they refuse to do so where, if the weld failed, disaster might follow, although no objection is taken to such welds in non-essential situations in a ship. The conclusions of the authors of the paper in their own words are: — 'Although there is a wide field for usefulness for oxy-acetylene and similar processes for joining and repairing steel plates, bars, etc., which are not required to resist very severe stresses, considerable caution must be exercised, and due regard paid to all the conditions before making use of them. A very serious responsibility attends the use of autogenous welding in cases where the parts are likely to be subjected to considerable mechanical strain. In any case, where failure tended to endanger life and limb, welds, if not entirely prohibited, should be accepted with extreme caution. Autogenous welds always represent a somewhat uncertain quantity, as there is no means, except by destructive tests, of ascertaining whether a weld is good or bad."—A. Campion and W. Gray.—*Indian Engineering*, Oct. 17, 1914, p. 253. (J. A. W.)

An Improved Form of Cam for Stamp Mills. —"The cams at present universally used in stamp mills lift the tappets with an involute form of curve, to which the surface of the tappet is always tangent; moreover, the line of contact between tappet and cam, if produced, would pass through the centre of the stem. This is no doubt a desirable feature, but the writer has long believed that it would be much more desirable if the cam were to pick up the stamp without shock, and gradually increase the upward velocity of the stamp throughout its upward movement. The involute form of cam attempts to impart instantly a considerable upward velocity to the stamp, starting it from a state of rest, and the result is a destructive blow, a great deal of noise, and much wear and vibration.

The writer has developed a form of cam which will lift the stamp with a motion similar to that of the piston of an engine between the end and middle of its stroke; in other words, harmonic motion, from the point of zero to maximum velocity. The curve will give this ideal motion only when the stamp is set for the exact drop for which the cam was designed, but the improved cam will not be as bad as the involute cam until the drop is reduced one-half. In other words, if a cam is designed for a 6 in. drop, it will be some improvement over the usual form of cam until the drop is only 3 in.

Any cam, whatever its form, must of course lift the stamp in the same number of degrees of revolution; therefore, with this new form, since the stamp starts more slowly on its upward course, it must end up by going faster, the average speed being the same as with the involute cam. The stamp, after it is no longer in contact with the cam, keeps on going up a certain distance, which is easily calculated. It is therefore possible to hang up a stamp without a cam stick, if the fingers are not more than ½ in. longer than necessary to hold the tappet above the cam.

With this design of cam the surface of the tappet is not tangent to the surface of the cam throughout the lift. The possible consequences of this were studied with a full-size model, and did not seem serious, as the engagement between the surfaces was an easy sliding motion, instead of a blow. If the drop is shortened, the blow becomes more and more pronounced, but the surfaces also become more and more nearly tangent.

Five cams of this new design have been running in one of the mills of the North Star Mines Co. now for over a month, fulfilling every expectation of the writer. Holding the hand on the tappet it is impossible to feel the cam strike the tappet, although the mill is running 107 drops per minute.

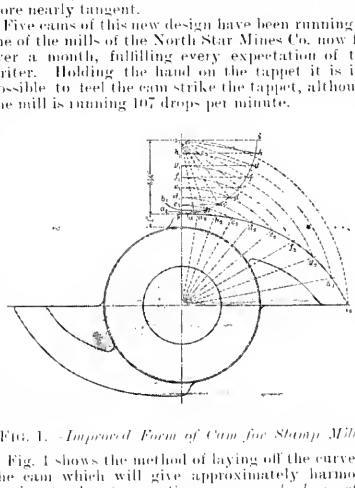


Fig. 1.—*Improved Form of Cam for Stamp Mills.*

Fig. 1 shows the method of laying off the curve of the cam which will give approximately harmonic motion to the stamp. Spaces $s a_1$, $a_1 b_1$, $b_1 c_1$, etc., represent the distances traversed by the stem in equal intervals of time."—A. B. Foote.—*Bulletin Amer. Institute of Mining Engineers*, Dec., 1914, p. 2765. (J. A. W.)

Dissolved Gold in Slime Residue. "There are two methods by which a proportion between the dissolved and undissolved gold contents of a residue may be obtained. The method most used perhaps is that in which the solution in the residue is filtered and assayed and the equivalent of dissolved gold calculated from the percentage of solution to dry slime in the residue. The value thus obtained when deducted from the assay

result of the dried residue giving the undissolved equivalent. The water washed residue is also used as an undissolved figure by some.

The other scheme of obtaining the separate values is performed thus:—The dried residue is assayed and the undissolved equivalent (obtained by assaying the water-washed residue) is deducted therefrom, giving the dissolved equivalent as the difference. In the first method it is obvious that in the event of the fire assay of the slime being low, a correspondingly low undissolved figure would result. If the second method be used, two errors enter (1) the low result from the fire assay and (2) the almost certain high result shown by the washed residue, and though one might counteract the other, the accuracy of the figures thus obtained must always remain in doubt, even more, than would be the case in the first method mentioned, where one correct result at least is obtained.

It is evident then that if either the washed residue sample be treated so as to obtain the complete displacement of dissolved gold, or the residue sample itself so treated, that a reliable total gold valuation could be obtained. As no easy or rapid method of so treating the former could be arrived at, a means of arriving at the total gold value had to be evolved. In order to carry out the latter course it was necessary to make comparative experiments which would indicate at what stages and in what manner the gold losses which had to be prevented occurred. Probably the most general method of preparing pulped slime residues for assay is as follows: The plant sample is well-stirred and the requisite amount for an assay sample withdrawn and placed in a shallow iron sample dish to dry over a fast fire. It would not take more than about 30 minutes to evaporate a 1,000 gr. (dry slime) sample of the "thinnest" residue to dryness in the above manner. The dried residue is then rubbed through a screen preparatory to weighing out for assay. Except, perhaps, in cases where the amount of dissolved gold is very small, it will generally be conceded that the assay results from samples prepared in the above manner do not show the whole of the gold contents. Likewise it must be admitted that the discrepancy is due to the influence of high or comparatively high temperatures which in some way bring about a loss of gold. There are then two definite periods during which the slime is subjected to higher temperatures:

(1) During the period of drying over a fast fire.
(2) During the fusion of the assay portion of the slime and the flux in the crucible.

Whether the heat developed in either stage is responsible for the whole of the loss, or whether the latter is distributed between the two, was a matter for determination, the results of which are included in this paper.

Gold Loss During Drying of Assay Sample.—The generally accepted theory as to the manner in which the loss occurs during this period may be set out as follows: By a process of capillary attraction during the latter stages of drying the concentrated gold-bearing solution is drawn through the tiny channels in the slime to the hotter sides and the bottom of the vessel being deposited there as an incrustation together with other salts. It may not always be possible, and certainly it is generally impracticable to remove all the crust so formed from the vessel, with the result that the main sample is impoverished and that portion adhering to the dish enriched. In a similar manner slime dumps drying in the hot summer atmosphere may get a thin surface crust in places so enriched by capillary attraction that the assay value obtained therefrom may rise to many hundred times the average value of the residue itself. The theory outlined above has been proved to account for most of the gold lost in assaying, and probably on that account further investigation has been carried out to a limited extent only. The writer has frequently been led to suspect some other agency in bringing about portion of these losses and has favoured something in the nature of volatilisation process. Although unable to definitely prove volatilization (or mechanical carrying over) of gold during the drying period it seems feasible enough that such a process could exist. It is freely acknowledged that, in the presence of alkali chlorides, gold may be volatilised at comparatively low temperatures. May it not be possible then that combined gold intimately associated with such chlorides would also tend to dissociate and then re-combine with the chlorine from the alkali salt? Whether the loss during drying be due to capillary attraction and its attendant effects or to volatilization, or is caused by a combination of both the above factors, the fact remains that there is always a definite loss, and also that such loss varies directly with the amount of dissolved gold present.

Since the losses so far described have been traced to the effect of high temperatures during drying, it might reasonably be expected that losses on samples dried at or about atmospheric temperatures should be appreciably reduced. Experiments in this direction have shown that the value of one portion of a sample dried at low temperatures is invariably higher than that of a similar portion dried under hotter conditions, the difference between such values being proportional to the dissolved gold present. A few typical examples of such experiments conducted under similar conditions will suffice to show the marked effect of higher temperatures in bringing about a very appreciable loss of gold.

TABLE "A"

Ref. No.	Value per Ton	Remarks.
	s. d.	
1	1 3	Drying 20 minutes over fast fire
2	1 6	4 hours at about 200° C.
3	2 0	12 hours over steam

N.B.—Dissolved gold values quoted throughout refer to actual "dissolved" content of 1 ton dry slime deduced from per cent. solution in pulped sample.

The above tests were run on three portions of the one sample, which contained dissolved gold to the value of 1s. 5d. per ton dry slime.

Again, a sample containing 3·8s. dissolved gold assayed only 2·4s. per ton on a portion rapidly dried in the usual manner, while another portion of this sample drying for six days in the open air assayed 3·2s. per ton, an increase of 1·1s. on the former, but still 0·6s. less than the actual dissolved content of the sample. Although the two typical cases quoted above show that slow drying will give a result more nearly correct than rapid drying, the latter of the two experiments referred

to clearly demonstrates that even if slow drying were practicable the valuation obtained thereby is by no means correct, or even approximately correct, as there is still a deficiency of 0·6s. on the dissolved gold estimate, without taking into account the undissolved figure.

The above figures speak for themselves and show how very unreliable the usual method of obtaining the residue may become.

Having thus ascertained the extensive nature of these losses, the question as to how and where they take place now strongly asserts itself. In the main there are three distinct directions in which gold losses may occur; any one, or any combination of which may result in the total loss entailed. They are as follows:—

(1) By capillary attraction and its attendant loss in scale;
(2) Expulsion of gold by heat employed in drying;
(3) Subsequent expulsion of gold from the assay charge during fusion.

Causes (2) and (3) are dealt with under the heading "losses during fusion."

Referring to No. 1:—As has been previously stated, this process is commonly reckoned to be the bugbear met with in the assay of residue containing dissolved gold. Without fear of contradiction, it may be said that in nearly every case loss by capillary attraction accounts for the greater part of the discrepancy that invariably occurs, but, at the same time, the whole loss can by no means be attributed to this cause, as will be shown later. The slime upon which experiments set out in these notes were conducted was of a highly oxidized and clayey nature, in an extremely fine state of division, and very difficult to settle from suspension in water.

In drying the sample as the last of the moisture is being expelled, little vughs or cells are formed throughout the cake, and in these cells at and near the bottom most particularly may occasionally be seen the salts drawn thither in concentrated solution by the heat at the bottom, gradually crystallized out and then dehydrated. The inclusion of this deposit in the scale accounts for the enrichment of the latter, and the impoverishment of the bulk of the sample. The amount of gold deposited in the above manner, and consequently the value of the scale enriched thereby, is a function of the rate of drying and the dissolved gold content of the sample. The experience of the writer has been that the richest scale is formed when—other things being equal—the sample is dried so that the atmosphere immediately above the drying cake is much cooler than the bottom of the drying vessel itself.

Table B hereunder shows the effect of capillary attraction on samples of different dissolved gold content:—

TABLE "B."

Ordinary Assay Valuation.	Loss per Ton due to Scale.	Soluble Gold.
Shillings 2·1	Shillings 1·5	Shillings. 3·8
1·1	0·4	1·8
1·3	0·25	0·8

The loss in scale was arrived at by multiplying the actual scale valuation by the percentage of scale formed. For further scale assays see Table F.

It is evident then that a great portion of the gold lost is due to the presence of the latter in solution prior to drying; therefore, it is natural to conclude that if such gold be precipitated prior to drying, the loss should be minimised, if not altogether prevented.

Minimising Losses by Precipitating Gold in Sample Prior to Drying.—Research work carried out in South Africa several years ago pointed to the fact that for the above purpose the precipitant of most merit was cuprous chloride, and, after use of this salt over a considerable period, the writer is of the opinion that it would be difficult to obtain a reducing agent which would give better or more consistent results. Practically, the only fault to be found with the salt is its readiness to crystallize out during cold weather, the storage bottle having to be kept in a warm place, or the solution heated before use. Comparisons made over a lengthy period between assays of slime dried in the ordinary way and samples dried with the addition of cuprous chloride prior to drying show an increase of over 30 per cent. in favour of the latter.

Assuming that the following equation represents the re-action between the small quantity of dissolved gold and the cuprous chloride (Cu_2Cl_2 in acid (HCl) solution*:—

$$Na_2(Au(CN)_2)+3Cu_2Cl_2+2HCl=$$
$$2Au+3Cu(CN)_2+3CuCl_2+2HCN+2NaCl.$$

Or, in other words, the dissolved gold molecule, after decomposition by the acid thus:—

$Na_2(Au(CN)_2)+2HCl=2NaCl+2HCN+2Au(CN)$

is attacked by the reducing agent, throwing the gold out of solution according to the equation:

$$2Au(CN)+3Cu_2Cl_2=2Au+3Cu(CN)_2+3CuCN.$$

Considering the small actual quantity of combined gold present in residue, it is evident that the use of a very strong solution of cuprous chloride, or the presence of a large quantity of a weaker solution is neither necessary nor desirable.

A solution containing about 5 per cent. Cu_2Cl_2 has been found to give the best results. It is prepared by reducing from 40 to 50 gm. of cupric chloride ($CuCl_2$) with about 10 gm. of copper foil in 200 cc. of 1:1 HCl (the pure acid being used for preference) boiling until all the copper is dissolved, make up to a litre with water containing sufficient HCl to prevent the Cu_2Cl_2 being thrown out of solution. To prevent oxidation 5 gm. of Cu are placed in the storage bottle. It has been found inadvisable to make up more than a litre at a time owing to liability of oxidation. Fifty cc. of the Cu_2Cl_2 solution at the most are added to an amount of pulp that will furnish about 4,000 gr. of dry slime, and after thorough stirring, the sample dish with its contents are placed to dry over the ordinary fast fire. Long contact with the heat at this stage has been found to have no effect on the value of the sample treated as described above.

It may be mentioned at this juncture, that another formula for making up the reducing solution from copper sulphate and common salt in the proportions of 150 $CuSO_4$ to 350 NaCl and acidulating with H_2SO_4 was used, and comparisons made side by side with the other formula. The results of these experiments led to the rejection of the $CuSO_4$NaCl formula after a fair trial, the reasons for so doing were:—

* These equations require a grain of salt! ED. COM.

1) That values obtained from the use thereof were slightly lower than those shown by the other, even when both solutions were freshly made up.
2) After a few days values, compared with the results shown from the use of the reduced cupric chloride formula, showed a remarkable decrease, and at the same time a large amount of hard scale was formed. The scale thus formed was too hard to rub through a screen and required grinding in order to reduce it to a sufficient degree of fineness for assay, and it was subsequently found that the remarkable drop in values was due to the loss of gold in this scale, which, on account of the inconvenience of grinding was not added to the main portion of the sample. Reference to Table F *re* Losses in Scale will show the great discrepancy between the actual assay of the sample and the value of same after the addition of the proportion lost in scale.

The following Table C indicates what may be regarded as typical examples of the effect of cuprous chloride on residues containing different proportions of dissolved gold:

TABLE "C."

Bal. No	Plain Dried	Dried with 50 cc. Cu_2Cl_2	Dissolved Gold Content of one Dry Ton	Washed Residue
	s. d.	s. d.	s. d.	s. d.
1	4 3	5 3	2 0	3 0
2	2 1	1 3	3 10	1 3
3	1 3	2 9	2 0	1 3
4	0 10	2 0	1 10	0 9

A glance at the above table will suffice to show the uselessness of the washed residue valuation in residue containing a large proportion of dissolved gold. There is little doubt but what the impracticability of washing the cake absolutely free from soluble gold is responsible for the discrepancy between the sum of dissolved and undissolved values and the assay of Cu_2Cl_2 and at the same time is strong evidence against the use of washed residue valuations for basing calculations upon.

In everyday practice, with the use of Cu_2Cl_2 it has been found that less than 10 per cent. of the assays made show even approximate equality between the value by Cu_2Cl_2 and the sum of dissolved and washed residue figures; the reason for this will doubtless be understood when it is stated that 96 per cent. of the slime upon which these experiments were conducted would pass a 200-mesh IMM screen and 0·8% remain on a 100 mesh, only a limited time for washing the sample cakes being available.

Loss During Fusion. For purposes of demonstration sample No. 2 in Table C, a typical example of the effect of higher temperatures on samples containing a relatively large proportion of dissolved gold will be taken. The sample, hot dried, without any precipitant assayed 2·1s. per ton. The scale from this sample amounted to 6%, and assayed 30s. per ton, equivalent to a loss of 1·2s. per ton of slime. The addition of this loss to the value of the rest of the sample only brings the total to 3·3s., whereas the assay of the portion to which Cu_2Cl_2 was added prior to drying showed 4·2s. per ton.

Where has the discrepancy of 0·9s. per ton occurred? Either—

(1) By expulsion during drying of sample;
(2) By expulsion during fusion of sample so dried;
(3) During fusion of scale assay;

or by any combination of the above causes.

There appears to be no means of determining the truth of the first assumption except by an indirect method, viz., by difference.

Passing to the 2nd cause:—In the event of the hot dried sample containing combined gold, it is possible that the effect of the high temperatures attained in fusion would be such as to cause a further loss during this latter stage. In order to ascertain whether a fusion loss actually took place, a portion of the hot dried sample was taken, shaken up with 0·05% cyanide, the usual 50 cc. of Cu_2Cl_2 added, and the whole placed in a dish to dry over a rapid fire. The value obtained from this sample was 2·5s. per ton, thus proving a loss during fusion of 0·4s. per ton.

(3) Seeing that the main portion of the sample, deprived as it was of a large proportion of dissolved gold, showed an appreciable loss during fusion, it was not surprising that the Cu_2Cl_2 assay of the scale should show 38s. per ton, thereby pointing to a loss of 8s. per ton of scale during the fusion of the latter, or equivalent to 0·3s. per ton slime.

The addition of the losses determined in the preceding tests to the hot dried value and scale assay is as follows:

	Shillings.
Hot dried Assay ...	2·1
Loss in Scale ...	1·2
Loss in Fusion of Sample ...	0·4
Loss in Fusion of Scale ...	0·3
Total	4·0

The difference between this corrected result and that value obtained from the portion to which Cu_2Cl_2 was added prior to drying now stands at 0·2s., which may be due to expulsion during drying, but may also be due to experimental errors, so it seems best, therefore, to designate it as unaccounted for. The results, however, show that the greater part of the 0·9s. formerly unaccounted for was lost during fusion. From many similar and even more conclusive results the writer has come to the conclusion that the fusion loss is anything but a bogey; also that this loss always occurs and varies directly with the amount of gold remaining in the combined state after drying. The following are a few typical examples:—After drying the sample contained the equivalent of 2·1s. of combined Au per ton slime and lost 0·4s. during fusion, *i.e.*, 10% of the gold content of the sample; another sample containing the equivalent of 3·8s. of combined Au per ton slime lost 1s. during fusion, *i.e.*, 26% of the gold content. Again a residue showing 1·75s. combined Au after drying, lost 0·4s. during fusion, equivalent to 20% of the gold content of the sample. With a view to drawing up a table which would show the stages at which losses occurred and their respective proportions, the following series of experiments were conducted. The sample contained 3·8s. dissolved Au per ton dry slime, and included in Table D given below are results of assays of sam-

ples treated in a similar manner to that in which the above was split up and experimented on :—

Portion No. 1 was left to dry in the open air for several days, being contained in an ordinary iron dish. Assay value = 3·2s.

Portion No. 2.—Dried side by side with No. 1 but contained in a glass vessel in order to prevent any possible loss by capillary attraction and inclusion of Au in pores of vessel itself. Assay value = 3·2s. per ton.

Portion No. 3.—Treated with 50 cc. Cu_2Cl_2 in iron dish and placed with Nos. 1 and 2 to dry in the air. Assay value = 4·2s. per ton.

Portion No. 4.—Same as No. 3, but contained in glass vessel and dried in air. Assay value = 4·2s. per ton.

Portion No. 5.—Hot dried in usual fireplace, without addition of any reducing agent. Value = 2·1s. per ton.

Portion No. 6.—Dried side by side with No. 5, but had 50 cc. Cu_2Cl_2 added prior to placing over fire. Value = 4·2s.

There was practically no scale formed by the first four tests, the cakes falling away clean from the receptacles. The scale from No. 5 amounted to 4% of the weight of the whole sample, and assayed (with addition of Cu_2Cl_2) 38s. per ton. There was only about 1% of scale arising from the drying of No. 6, and this was easily rubbed away and added to the main portion of the sample. The assays given above show that there was no loss by capillary attraction in air drying, the glass vessels being brushed quite clean. The loss of 1s. shown on both Nos. 1 and 2 must, therefore, have occurred during fusion, and this was actually demonstrated by shaking up the slime with weak cyanide solution, adding the usual amount of Cu_2Cl_2, and drying under ordinary hot conditions, the result obtained being 4·2s., agreeing with both air dried and hot dried samples dried with Cu_2Cl_2. The greater part of the loss shown by No. 5 was traced to the effects of capillary attraction (accounting for 1·5s.) and high temperatures during fusion, 0·4s. per ton being directly traced to the latter.

As eight checks, representing four portions showed 4·2s. per ton, this valuation is designated Total Gold in the Table, as a basis on which to calculate the proportion of losses.

The main features exhibited by Table D are :—

(1) The constancy of the percentage lost during fusion, which amounts to about 10% of the total gold content.

(2) The almost constant ratio that exists between the combined loss and the dissolved gold equivalent.

Thus in No. 1a this ratio is 2 1s. to 3·8s., *i.e.*, 55% ; in No. 2a it is 1s. to 1·8s., *i.e.*, about 55% ; and lastly in No. 3 the ratio is 6d. to 11d. = 54%.

The figures obtained from an dried experiments bear out the statement that the fusion loss varies with the amount of combined gold, remaining in that condition in the sample when ready for weighing out.

It is not claimed for the Cu_2Cl_2 assay as set out herein that it will show the absolute gold content of the sample by fire assay, but experience of the method has proved that it is possible to clean up much more nearly to assay than by the old plain drying method, which has been shown to be in error to the extent of about 50% of the dissolved gold. For residue containing more than half the gold in combined form, an assay of the type set out herein is essential, while for residue which contain less than that quantity the method should prove beneficial.

The Effect of Cupric Chloride ($CuCl_2$).—It was noticed that if the last 100 cc. or so of the reducing solution remaining in the storage bottle were used, an abnormally low gold value was obtained. Suspecting that the oxidation of the cuprous salt had either directly or indirectly brought this about, a comparison was made between several portions of one sample treated (1) with 5% cuprous chloride, (2) 5% cupric chloride in HCl, and (3) dried without either.

The following Table shows a few of the results obtained. No. 1 sample containing a relatively small proportion of dissolved gold and No. 2 a larger amount of dissolved than undissolved gold.

Further loss when $CuCl_2$ is present in relatively large amount may be due to either or both of the following causes : (1) The dissociation of simple cyanide ($AuCN$) formed by the action of HCl upon the double cyanide, portion of the gold being volatilized ; or (2) the volatilization of the intact molecule $AuCN$ in the same way as the analagous halogen compound $AuCl$ is volatilized at

TABLE "D."

Ref. No.	Total Au by Cu_2Cl_2	Air Dried Value.	Hot Dried Value.	Total Loss.	Losses Distributed Between			Dissolved Au equivalent.	Remarks
					Drying.	Fusion.	Un accounted		
	Shillings	Shillings	Shillings	Shillings	Shillings	Shillings	Shillings	Shillings	
1a	4·2	—	2·1	2·1 =50%	1·5 =35·7%	0·4 =9·5%	0·2 4·8	3·8	Hot dried samples
b	4·2	3·2	—	1	*nil*	1 =24	—	3·8 90%	Air dried samples
2a	2·1	—	1·1	1 =48%	9d. =36	3d. =12		1·8 86%	Hot dried samples
b	2·1	1·7	—	0·4	*nil*	0·4 =19	—	1·8	Air dried samples
3	1·5	—	1	0·5 =33%	4d. =22	2d. =11%	—	11d =61%	Hot dried samples

TABLE "E."

Dried with Cu_2Cl_2	Plain Hot Dried Portion.	Dried with addition of Cupric Salt.	Dissolved Gold.
s. d.	s. d.	s. d.	s. d.
5 3	4 3	3 9	2 0
2 3	1 6	1 3	1 9

moderate temperatures. The former supposed cause would, perhaps, be the more probable, as no doubt the tendency of the cyanide to break up into carbides and free carbon would have a reducing effect on the gold thus combined.

Losses in Scale.—Having dealt with the gold losses entailed during the periods of drying and fusion and the prevention of same to a large extent, it would perhaps not be out of place to conclude this paper with a tabulated list of various scale material, the values of same, and attendant effects upon the sample values from which such scale was derived. The effect of capillary attraction in enriching scale has been dwelt upon to some extent, but included in this last table are some examples of scale enriched by a combination of capillary attraction and other causes.

TABLE "F."

"Scale" from	Per Cent. Scale.	Assay Value Scale.	Assay Value Sample.	Sample Value after adding Scale Loss	Loss in Scale per Ton Slime.	Dissolved Gold Content.
		Shillings	Shillings	Shillings	Shillings	Shillings
Plain dried	3·5	1·5	1·25	1·3	0·05	0·7
$CuSO_4 + NaCl$	15	8	0·8	2	1·2	1·5
Plain dried	4	38	2·1	3·6	1·5	3·8

It should be mentioned that the slime used in all these tests, though extremely finely divided or rather due to that fact never formed more than 4% scale when rapidly dried alone. The addition of Cu_2Cl_2 had little or no effect on the amount of scale formed, whether the solution were fresh or oxidized; but the reducing solution made up from $CuSO_4$ and NaCl even when fresh gave more scale, and when oxidized the amount of scale formed rose to as much as 15% of the sample and assayed 8s. per ton, or ten times the value of the rest of the sample.

In conclusion, it may be said that the cuprous chloride assay as described in the foregoing pages is still in use and giving satisfactory results.—L. R. BENJAMIN: *Journal of Chemical, Mines of W. Australia*, Sept., 1914, p. 235; (J. A. W.)

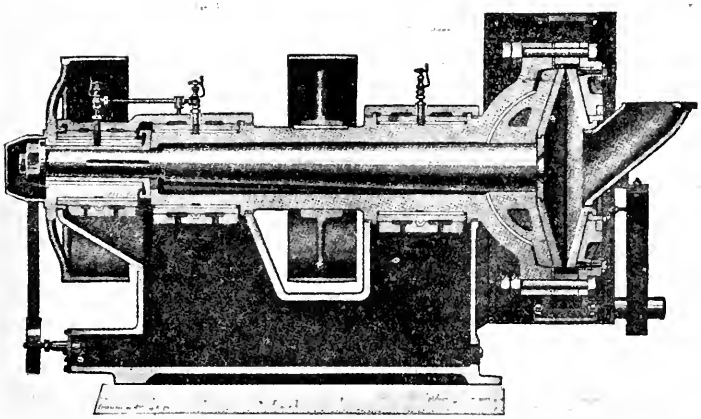
COLORIMETRIC METHOD OF DETERMINING TITANIUM IN IRON AND STEEL. *Amount above* 0·02%: Two gm. each of the sample and of steel free from titanium are dissolved in 80 cc. of sulphuric acid (1 : 3), 4 cc. of nitric acid (sp. gr. 1·2) added, and the solutions boiled until free from nitrous fumes, and (in the case of cast iron) filtered from silica. The liquids are diluted so that in equal volumes the colours match, and an addition of 2 cc. of 3% hydrogen peroxide is made to each. Standard titanium solution is then added to the solution of non-titanium steel to match the colour of the test sample. The standard solution is prepared by heating 100 gm. of ferro-titanium with about 50 cc. of strong hydrochloric acid and (after some moments) 5 cc. of nitric acid (sp. gr. 1·12). After evaporation of about half of the acid the liquid is diluted with 15 to 20 cc. of water and the ferro-titanium treated with more of the mixed acids until sufficient has been dissolved (about 12 times). The filtrates are evaporated until titanic acid separates, and then freed from iron by extraction with ether, united, heated with 150 cc. of sulphuric acid (1 : 3), and filtered. The filtrate is diluted to about 700 cc., boiled with excess of ammonia, and the precipitated hydrated titanic acid washed free from chlorides by decantation, dissolved in 50 cc. of sulphuric acid (1 : 1) and diluted to a litre. The strength is determined by precipitation with ammonia, and the solution diluted so as to contain 0·0002 gm. of titanium per cc. *Amounts below* 0·02%.—The solutions of the sample and of non-titanium steel are evaporated to 10 cc., transferred to separating funnels (the dishes being rinsed with about 15 cc. of dilute hydrochloric acid, 2 : 1), and shaken with 50 cc. of ether (free from alcohol) to extract the iron. The lower layers are drawn off, each diluted with 225 cc. of hot water, and boiled with ammonia in slight excess, the precipitates washed with hot water, and dissolved in 10 cc. of sulphuric acid (1 : 3), and filtered into 30 cc. colorimetric cylinders. The filtrates and washings which should be equal (15 to 18 cc.) in both cases, are then treated with 3 cc. of 3% hydrogen peroxide. In the presence of even 0·001% Ti a yellow tint quite distinct from the green tint of the non-titanium solution will be seen, and its amount may be determined as described above. In the case of cast iron the sample is dissolved in hydrochloric acid, the solution filtered, and the residue ignited (and after volatilisation of silica with hydrofluoric acid and a few drops of sulphuric acid) fused with 1 gm. of sodium carbonate. The mass is dissolved in hydrochloric acid, and added to the main solution, which is then extracted with ether, etc.—C. R. MCCABE: *Journ. Soc. Chem. Ind.*, Nov. 30, 1914, p. 1090; (J. A. W.)

PULP CONSTANTS. *Synopsis*: Heretofore the specific gravity of the solution has been generally taken as unity. The degree of error arising through this source is shown, together with corrections for it and the method of applying them. The error which may be introduced through variations in temperature is discussed. Extensive tables have been calculated from which corrected values may be read for the generally used con-

stants. Logarithms are also given for use where desired. Methods which compensate for differences in temperature are given for determining the specific gravities of dry slime, pulp and solution. Illustrative curves are shown embodying a portion of the data of the tables.—G. H. Clevenger, H. W. Young and T. N. Turner.—*Synopsis of a lengthy and important paper in "Engineering and Mining Journal,"* Dec. 19, 1914, p. 1079. (G. H. S.)

Symons Disc Crushers.—"Disc crushing is an old principle, the application of which to large scale milling operations has been accepted but slowly by metallurgical engineers. The past decade, however, has developed new ideas and refinements in the application of this principle, with the result that disc machines are finding a place in modern metallurgical plants and have given efficient service after passing the experimental stage.

The accompanying half-tone presents a cross-section of one type of disc crusher manufactured by the Symons Brothers Co. of Chicago. This crusher is being used in a number of crushing plants and is reported to be giving satisfactory service. The machine is rigidly constructed with manganese wearing parts. There is no grinding action, the action of the machine being a simple crushing movement. A general idea as to the service of the machine is supplied in a letter from a Canadian operator, who has had a disc machine in service for the past two years.



CROSS-SECTION OF SYMONS DISC CRUSHER.

'The crusher in this instance is a 48 in. disc machine crushing ore composed of greenstone, epidote, magnetite, and a little granite. The granite was much the easiest to crush of any of the four. The greenstone is hard and tough, and has a compressive strength varying from 30,000 to 50,000 lb. per sq. in. The material is abrasive because of the large percentage of silica in both the rock and the magnetite.

'The crusher has been used at different times throwing off the belts and choking down the with different sizes of feed. One size being a product from a No. 8 gyratory crusher giving slabs of about 4 in. thickness and a breadth and length much greater. The machine easily handles pieces as large as this, and it would seem will handle pieces up to 6 in. in thickness with equal results. The size of the feed to the machine is only limited by the size of the spout which admits it. The product from the machine will all pass through a 1½ in. revolving screen.

'The capacity of the machine is 50 gross tons per hour, with the machine set to give a product as stated. This machine is driven by a 75 H.P. motor and probably uses an average of 60 H.P. The inertia in starting is great, and it requires a motor of heavy starting torque, and a coil-wound rotor motor is used for this purpose.

'I have no way of knowing, in the absence of data, whether this particular type of machine will make a more dusty product than a gyratory, but observations lead me to believe that there is little difference, if any.

'In studying the machine, one will come to the conclusion that, in view of the fact that there is no differential rotation between the two discs, but simply a nip and, of course, a quick discharge because of centrifugal force, that the machine sizes its product with the largest opening of the discs and not the closest. There is a great amount of tramp steel from the mine going into this crusher, and it brings it to an almost instantaneous stop. This, of course, is severe on any machine, but I am unable to remove the tramp steel in the ore with a magnetic separator, since the ore itself is magnetic. This steel getting into the crusher has caused little damage, as during the entire time of operation, nearly two years, only one part, the outer spider, has broken. The machine is rigid and stands a great deal of abuse without apparent injury.'"—*Mining and Scientific Press*, Oct. 31, 1914, p. 708. (G. H. S.)

OPERATION OF OLIVER FILTERS.—The last few years have shown a tendency to the displacement of the submerged leaf by the continuous type of slime filter in the cyanide mills of the United States and Mexico. Of the several forms of continuous filter being used, the Oliver is best known and most popular. It is easy to operate, yet success hinges upon various little known minor details, as shown by the following notes on its operation at the Globe mill at Dedrick, California, and elsewhere.

Vacuum. Methods of obtaining vacuum are widely divergent. Good work cannot be done with a vacuum less than 20 in., and 25 in. is better. Wherever sufficient water under high pressure is available, the simple water ejector, used in the Grass Valley mills and elsewhere, is most commendable. Installation and operating costs are slight, and a uniformly high vacuum is easily maintained. At the North Star mills, where E. L. Oliver invented and perfected his filter, novel use of an old mine shaft is made to lengthen the draft tube, 60 ft. fall being secured. At the North Star the ejector has both high and low pressure water connections; available water supply determining which shall be used. Average vacuum maintained on the filter is 25 in. of mercury. This approaches perfection as the altitude is 3,000 ft. With the ejector, filtered solution is withdrawn from the vacuum vat by a small centrifugal pump. However, the latter uses but little power, and, since it handles clear solution, repair costs are negligible.

Feed. Pulp is usually fed so as to enter the filter vat at the top centre on the ascending side of the drum. If introduced at one end, an abnormally thick cake results on the adjacent portion of the drum and it is correspondingly thin elsewhere. This is particularly true with fast settling ores and in the case of drums more than 12 ft. long. Generally, for all slimming mills, pulp feed of not less than 1·45 specific gravity (D 2·7) is recommended. Referring again to Grass Valley practice, the Empire pulp is fed at 1·30 specific gravity with good success. However, separate sand treatment is practised and the filter feed is an impervious slime containing no granular material. It will not settle to a density greater than 1·35. With thin feed 1·30 or less in the average mill, rapid settling together with segregation of coarse material results the accumulated material acts as a brake upon the revolving drum and finally throws off the driving belt. When this occurs it is wiser to shut out the vat than to try to force the drum to revolve.

It is desirable to keep alkalinity of feed as low as other considerations will permit, as lime salts are detrimental to the cloth and their precipitation in the canvas will necessitate frequent acid washing.

Agitation of Charge. Methods used to prevent settling of charge in the filter vat are sufficient on the average ore, but in certain cases they are inadequate. Internal air lifts are being used at the Globe with considerable success. There are two air lifts on the rear and three on the front side of the 16 ft. filter at this plant. Possibly the addition of two or three more air lifts would entirely obviate the need of mechanical agitation. The agitator now used consists of a longitudinal shaft, fitted with short paddles, running in stuffing boxes placed midway between the lower side of the drum and the bottom of the filter vat. This shaft is driven at about 20 r.p.m. by sproket and bevel gears. This apparatus is much more effective when a 3 in. angle iron the length of the filter is bolted across the ends of the paddles parallel to the shaft. With the latter arrangement, the shaft need be revolved at much less than the above stated speed. The stuffing boxes of the agitator shaft often give trouble. This may be minimized either by feeding a small stream of solution to each gland, or grease may be automatically forced into them by compression cups. If the agitator shaft is sprung or badly worn or if neither solution nor grease is being regularly fed to the glands, it is wellnigh impossible to prevent serious leaks through the boxes. At the extreme bottom of the vat an air agitator is placed lengthwise of the drum. It consists of a length of perforated canvas-covered 1 in. pipe with pressure air connections. It is intended for emergency use, when for any reason the mechanical agitator is out of commission, and no doubt has some value if started before the charge has become packed. The canvas covering of this pipe should be renewed once in six months.

When material is settling, the cake shows irregular "bunches" of coarse sand. If prompt action is then taken, sluicing and the attendant shut down may sometimes be avoided. This difficulty may arise from one of several causes: (1) inefficient agitation; (2) carrying pulp level too high; (3) specific gravity too low; or (4) poor work in fine grinding machines or classifier. These may be considered in detail.

1. Prevention of settling is absolutely imperative, and this difficulty would seem to have been mastered particularly at Butte, where the Anaconda Company has several Oliver filters dewatering concentrates. Incidentally, this marks a further invasion of cyanide machinery into the field of general metallurgy, the attack having been begun by Dorr and others.

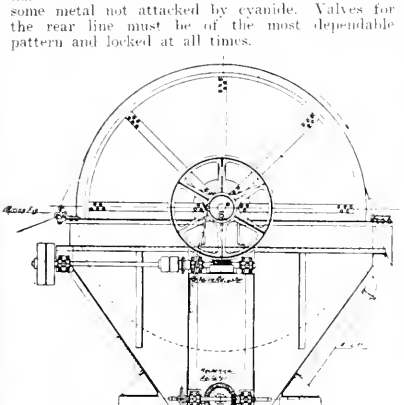
2. The best practice is to carry the pulp level about two feet above the bottom of the drum. With exceptional ores those colloidal in nature—the charge may be safely carried at much greater depths than above stated. When deep charges are permissible, capacity is increased both by the greater vacuum then secured and because a larger cloth area is constantly submerged.

3. As previously stated under the heading "Feed," pulp at from 1·45 to 1·60 specific gravity gives best results. Thinner pulp will usually cause trouble.

4. Screen analyses show that the settled material consists largely of fairly coarse sand. At a certain mill the appearance of coarse segregations in the cake is held to prove that grinding or classification is inefficient. This should not be applied too literally, however, with the idea that careful grinding will make it allowable to ignore other possible causes of settling. The small proportion of moderately granular ore in the ordinary mill circuit sometimes concentrates in the filter vat when the precautions above noted are disregarded.

Washing the Cake. Effective displacement of pregnant solution by wash water is of first importance in slime filtration operations. With the continuous type of filter, the period during which wash may be added is necessarily brief; consequently, thorough displacement can be obtained only by careful arrangement of pipes and sprays, close attention to amount of water, and even distribution of the same. Ordinarily capable operators obtain a tailing with soluble content (both

cyanide and dissolved metal) equally low as obtained with submerged leaf or pressure filters. The use of perforated pipes discharging water through small holes upon the cake, has been superseded by one of two forms of spray nozzles. The whirling spray (represented by the Vermorel and others) is rather more effective than the fan-shaped form sometimes employed (such as the Bordeaux). The tendency is toward the use of a comparatively large number of individual sprays, each with a relatively small aperture. In mills where volume of solution is a constant, it is customary to have the rear line of sprays arranged so that either barren solution or water may be used at will. The second spray line is placed lengthwise over the crest of the drum and water only is added through it. Sprays intended to handle solution must, of course, be constructed of some metal not attacked by cyanide. Valves for the rear line must be of the most dependable pattern and locked at all times.



CROSS-SECTION OF FILTER.

The importance of washing is evidenced by the great amount of experimental work along this line which has been done at various mills. A Nevada plant still uses shaking spray pipes. This plan is said to be especially adapted to ores that form an unusually porous cake; also to cases where a relatively large volume of wash solution is essential. It would seem, however, that the atomizing sprays, above noted, would meet either requirement more successfully and would have the additional advantage of effecting a small increase in total extraction. This increase is probably due to the splendid aeration given the cake while being washed with atomizers. Extraction during washing as obtained with the Oliver is certainly less than that secured by filters such as the Butters. In some plants equipped with the latter, this increase is said to approximate 2% of the total recovery. This comparison is hardly fair to the Oliver because the washing period is many times greater in the Butters.

Proper wash water control is facilitated by using a pressure gauge on the spray pipes. When using atomizers, the minimum permissible working pressure is 25 lb. per sq. in.; water is not well atomized at less pressure. Average working pressure may be stated at from 35 lb. to 40 lb. Best practice in filter cake washing may be summarized as the even distribution of the greatest permissible amount of wash over the entire exposed cake.

Washing the Cloth.—Continuous filter cloths require more frequent acid washing than do those of leaf filters. Scrubbing with 5 in. or 10 in. fibre brushes in a weak hydrochloric acid solution (about 5% HCl) not only restores permeability to the canvas, but also prolongs its life through removal of deleterious lime salts. The benefits of frequent and thorough acid scrubbing are significantly shown at the Black Oak. That mill holds the record for long life of cloth—seventeen months. This cloth was scrubbed weekly with 5% HCl solution.

Sampling and Disposal of Tailing.—The scrapers which remove tailing from the face of the drum should have no more than just enough tension to keep them resting lightly against the wires. Under excessive pressure, wear both of scrapers and wires is greatly increased and may even ruin the canvas by tearing it. The scrapers should be filed to a smooth and properly bevelled edge once each week. A convenient plan is to have a spare scraper on hand and interchange about once in two months; the one taken off can then be dressed at the convenience of the operator, and very little running time is lost. Two men need 15 to 20 minutes to dress the scraper when it is done each week.

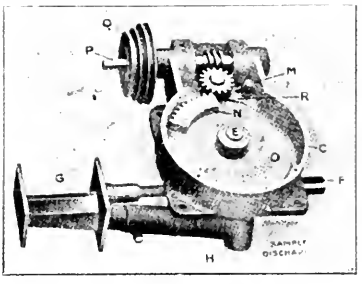
Tailing may be taken away from the filter either by belt-conveyor or sluicing. When ample water is available the latter method is, of course, to be preferred. The 'blow' should be only strong enough to slightly loosen the cake, usually under 10 lb. If too much pressure air is admitted, pregnant solution is blown through the cloth and the soluble losses thereby increased. The practice of taking tailing samples at a point above the section being blown off is to be condemned for the above reason, and samples are uniformly slightly lower in value than the actual loss in the material being discharged. Samples should be taken at intervals of not more than one hour, along the edge of the scraper from end to end of the drum.

Conclusions.—It is believed that this filter may be safely recommended as well fitted for general use in dewatering slime of whatever nature with the possible exception of colloids. The latter can be handled only at a greatly reduced capacity; this also applies to filters of other form. We believe the Oliver to be the best machine for general use with average quartz ores." H. A. Morrison and H. G. Thomson.—*Mining and Scientific Press*, Oct. 10, 1914, p. 554. (G. H. S.)

Automatic Sampling of Mill Pulps.—"The automatic sampler described in this article is the design of Mr. Elmore, and is being built by Messrs. F. W. Brackett & Co., Ltd., Colchester, England. The construction of the Elmore sampler is shown in the adjoining illustration.

It is to be noted that the sampler can be set with driving shaft at any angle in relation to the launder conveying the pulp to be sampled. By simply sliding the adjustable arm on the centre pin, the machine is at once adapted to various widths of launders.

The sampler is put into operation by motion imparted through a round belt on the grooved pulley *Q* which is keyed to the shaft *P*; this in turn rotates the worm engaged with the worm pinion. This worm pinion operates an eccentric (not visible on the illustration) which reciprocates the pawl *B*, causing the main ratchet wheel *O* to advance one tooth at a time. Thus the main ratchet wheel *O* is caused to rotate slowly.



AUTOMATIC SAMPLER.

To the upper face of *O* a toothed segment *N*, equal to about one sixth of the periphery of *O*, is securely attached, the teeth of which are of the same pitch as those of the worm pinion. By the slow rotation of the main ratchet wheel *O* the segment *N* gradually approaches the worm pinion and engages with it, which immediately results in the quick movement of *N* and *O*; and as these carry the pin *E* to which is attached the sampling cup *G* and discharge pipe *H*, it follows that during about five sixths of each revolution (the non-sampling period) the main ratchet wheel *O* is moving at a very slow rate, whereas, for the other one sixth of the revolution (the sampling period) the rate of movement is high, and the sampling cup thus cuts through the whole stream of pulp at a perfectly uniform rate of speed, and always at the same relative angle, during the whole period of sampling.

All the bearings and moving parts are enclosed in a substantial outer casing *C* covered with a dust-tight lid, thus forming an oil bath, and providing perfect lubrication, minimizing wear and tear, and insuring continuity of working." *Mining Journal and Chemical Engineering*, Dec., 1914, p. 790. (G. H. S.)

PROGRESS OF HYDRO-METALLURGY IN 1914. "The author gives details of systems and methods in use at the most important American copper mines.

A noteworthy point is the progress being made in the recovery as by-products of iron, arsenic and sulphur.

At the Nevada Douglas Mine it is said a method has been found of treating sulphide ores without preliminary roasting. The iron is recovered as ferric oxide and the sulphur as sulphuric acid. The management also claim to have a method of clarifying the copper sulphate solutions so that methods used in refineries for the recovery of the copper may be used and electrolytic copper obtained direct.

At the Septae Works a leaching and precipitation plant has been installed for the treatment of the dust from the roasting furnace chambers, along with poor siliceous oxidized ore. The cost of recovering the copper from these materials is about 8 cents per lb.

At the Shannon Copper Company experiments were made for the treatment of a low grade copper ore. The ore was roasted in heaps composed of 100 tons pyrite, then 1,000 tons oxide ore crushed to 2 in., and a covering of fines 1 ft. thick, the heap being sprinkled with waste solution from scrap iron precipitation. The roasted ore was leached with water in filter bottom vats. The ferric sulphate formed in roasting dissolves copper which may not be water soluble. The outflowing copper-bearing solution was run over tailings to use up the last of the ferric sulphate, before going to the precipitation vats. Extraction was 72 to 82%, and the advantages are (1) coarse crushing, (2) cheap installation, (3) no sulphuric acid required." WARREN AIKEN. *Mining and Engineering World*, January 2, 1915, p. 6. (H. R. A.)

NEW PLANT AT MT. MORGAN, QUEENSLAND.—"Considerable alterations have been made at the Mt. Morgan Company's plant during the past two years, and the general manager, M. Magnus, supplied the following data in November:

The old smelter has been entirely replaced by one consisting of three 54 in.×336 in. blast furnaces, fitted with 16 ft. forehearths and auxiliary forehearths. The height of charge possible is 25 ft. 7 in. Each furnace is provided with two outlets, one at each end, for discharge of matte and slag. All tapping blocks on the forehearths are of Australian chrome ore, in place of the usual cast iron water-cooled blocks. Each furnace has a complete bustle pipe around it, and a main bustle pipe on both sides. The water system is in duplicate. The charge floor is airy and light, and devoid of fume. Ore and fluxes are conveyed in side tipping cars, drawn by a 10 ton electric locomotive, to the floor, the furnaces being mechanically fed. Slag is carried in Dewhurst slag bowls, end tippers holding 10 tons, and side tippers 20 tons each. They are drawn by a steam locomotive. Two men handle about 1,000 tons of slag per 24 hours. The flue system is as follows: a down-cast enters an independent dust chamber behind each furnace, connected by a catenary flue, which in turn joins with the main dust chamber on a hill. The latter chamber is 17 ft. wide, 31 ft. high, and 71 ft. long, with a stack 250 ft. high and 11 ft. diameter.

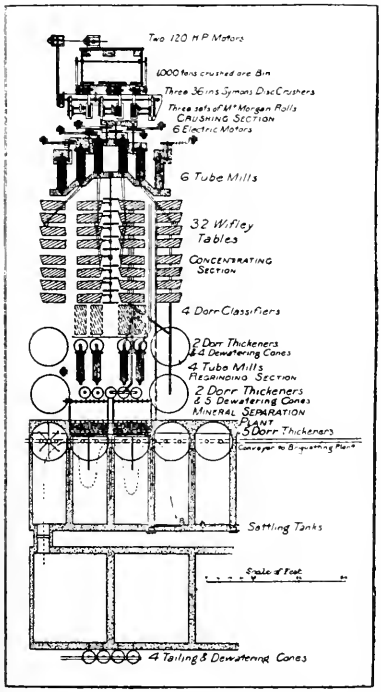
The New Concentrating Mill. The new concentrating mill of 500 ton daily capacity is now in operation, treating the siliceous ores of the mine, not economically treated by smelting. The accompanying cut explains the processes used. At the mine the ore is broken by gyratory crushers and delivered to the mill at 6 in. size. Two jaw-crushers reduce this to 2½ in. Suspended above a belt conveyor which feeds these crushers are two high-power magnets which collect any iron and steel mixed with the ores, and which might damage the machines. The ore is then conveyed to a 1,000 ton bin. Three Symons disc crushers crush to ¾ in., after which come three geared rolls and six Krupp tube mills. The latter produce an 80-mesh product which passes over 32 Wilfley tables. The slime from these is thickened in Dorr

machines, and the resulting pulp is treated by the Minerals Separation flotation process. All concentrates are drained, briquetted, and fed to the blast-furnaces. Basic converters are used.

The power plant for the mine and plants consists of the following: Six Babcock & Wilcox boilers of 20,000 lb. steam per hour capacity, and four smaller boilers; two 4,000 ft. and one 2,000 ft. Belliss & Morcom air-compressors; a Fraser and Chalmers mixed turbo-blower using exhaust steam from the compressors, compressing 10,000 ft. of air per minute to 14 lb. pressure; two Fraser and Chalmers blowers delivering 40,000 ft. of air per minute at 4 lb. pressure to the blast-furnaces; two Curtis turbo-generators of 1,500 kw. capacity, and circulating pumps, condenser, and ice-making machines.

The smelter and power plant were erected on a slag dump, which is 70 ft. high in places.

The workshops are 480 ft. long and 60 ft. wide, and during construction of the new plant, $100,000 of work of all descriptions was turned out. About 200 men are employed in them.



FLOW-SHEET OF MT. MORGAN CONCENTRATING PLANT.

The output in the four weeks ended October 18 was as follows: The concentrating plant treated 6,785 tons of ore, producing 1,771 tons of concentrate containing 137 tons of copper and 1,467 oz. gold; and the smelter handled 21,474 tons of Mt Morgan ore, 5,968 tons of Many Peaks pyrite, and 1,629 tons of concentrate, producing 724 tons of copper and 9,211 oz. gold. The value of this return was gold, $187,000, and copper, $190,000, a total of $377,000."—*Queensland Government Mining Journal.—Mining and Scientific Press*, Dec. 26, 1914, p. 988. (C. H. S.)

Solubility of Sulphides in Cyanide Solutions.—"In his article on continuous counter-current decantation in the *Journal* of Oct. 17, Herbert A. Megraw makes the following statement:—

'Sulphide ores may, and usually do, require previous concentration, but it is nevertheless true that by grinding a sulphide fine enough, its values are very likely to be given up in about the same time required in the ordinary slimes treatments To do this, however, sulphide slimes should be reground until they are in an exceedingly fine state of subdivision. In such case they may go through counter-current treatment along with ordinary slimes and have the metals satisfactorily extracted. Sulphides are, of course, governed by the same general laws as ores, so that what has already been said as to dissolution in the decantation circuit applies to them as well."

The qualifying phrase, that a sulphide's metals are "very likely" to be given up, saves this statement from being incorrect. And still it is so broad that it seems likely to lead to a misconception.

The solubility of the values in sulphide minerals depends upon the nature of the values and also upon that of the sulphides. It seems that gold may be recovered from almost any sulphide mineral by prolonged grinding and treatment, but silver only in exceptional cases. While at the Natividad mine in Oaxaca recently, I conducted a series of experiments on concentrates to determine that very point.

The concentrates consisted chiefly of clean galena and pyrite with a small proportion of blende and chalcopyrite. The galena and pyrite were so clean and bright that they could be taken off the Wilfleys in well defined and separate ribbons. The product of the tables carried from 8 to 10 kgm. of silver and about 300 gm. gold.

Experiments on these concentrates consisted of charging one kilo into an experimental tube-mill with two litres of freshly made cyanide solution and grinding continuously for from one to three weeks, taking a sample of both the pulp and solution daily. In the course of several repetitions with solutions of various strengths and other modifications the following conclusions were reached.

The content of gold in the washed pulp fell rapidly at first, then more gradually until about the fourth day, when it had been reduced to from 3 to 15 gm. After that, even to the end of three weeks, neither the pulp nor the solution showed any further extraction of gold.

Nearly 50% of the total silver dissolved during the first 24 hours. After that it continued in a decreasing ratio until the fifth or sixth day, after which all effect of the treatment ceased. The residue, even after weeks of grinding in solution, never contained less than two kilos of silver, and frequently as much as three kilos. Air agitation at intervals had no effect on the results.

Wet desulphurizing had no effect upon the silver extraction of the concentrates. Previous experi-

ments had shown that this reduction treatment increased the extraction on mill pulp to 85% and 90%, the normal extraction with straight cyaniding being only 40% and 50%. Wet desulphurizing had no effect on the solution of gold in slime or concentrates.

These results seem to show that the gold is all in the metallic state whether associated with the slime minerals or with the sulphides; that the ruby silver minerals, easily visible in the ore, do not enter into the concentrates, but are immediately reduced to powder under the stamps; and that the silver in the concentrates occurs in some form or forms not soluble in cyanide solution even when preceded by a reduction treatment.

It is interesting to find in Mr. Megraw's articles that cyanide men are beginning to doubt the value of air and air agitation as a rule at all. A few of my experiments are illuminating on that point. A kgm. of the sulphide ore from below water level was ground dry in a hermetically sealed tube mill. Sodium hydrate solution and aluminum plates were then introduced and the mill set in motion until a considerable internal pressure, due to hydrogen, developed. Finally, the cover was removed just long enough to remove the aluminum plates and introduce a weighed quantity of sodium cyanide. The mill, tightly closed and containing a few pebbles to affect a mild agitation, was kept in motion for another 24 hours. This procedure, in which air and oxidation were carefully excluded, gave as high extraction as agitation with air pressure.

I have followed the articles on cyaniding by Mr. Megraw with great interest. My interest, at least, is increased by his manner of dealing with first principles. He does not constantly exalt the importance of any particular detail, or of some particular machine. It is pleasant to observe in the development of the process the steady movement towards more simple arrangement and more simple apparatus. The multiplication of complicated machines, each guaranteed to increase the extraction, seems to have ceased. Perhaps some day the cyanide man will not be expected to spend 18 hours a day with a monkey wrench in his hand. When that time arrives he may be allowed to devote his time to his real business.—A. Van Zwalenburg, Mexico, Oct. 29, 1914.

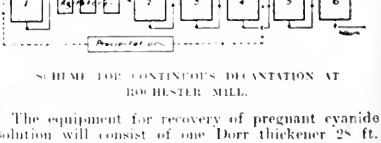
Solubility of the gold or silver contained in sulphides has been discussed by cyanide operators for many years. Mr. Van Zwaluwenburg's objection to my statement that sulphides will usually give up their contained metal to cyanide solution if the mineral is ground fine enough is not surprising, but I am still of the opinion that my original statement is essentially true. The usual sulphide encountered in Mexican silver-bearing ores will give up its silver to cyanide solutions when the mineral is sufficiently fine, a fact which has been demonstrated by experiments at many places. At Guanajuato, Pachuca, Pozos, and many other camps, fine grinding has been found sufficient to insure high percentage extractions on concentrates. Fine grinding in such cases may mean much finer than −200 mesh size. The −200 mesh material often has to be reground, but in view of the friability of the sulphides the operation is not particularly difficult.

It is undoubtedly true that there do exist silver minerals which are not amenable to this treatment. Some of the lead-silver compounds belong to this class. The number of these compounds, however, is small in comparison to those which are amenable to cyanide treatment. As to the economy of such methods, I have said nothing. With some ores profits may be obtained, and with others there may be none. Each one presents a special problem which must be worked out in its own merits. The chemical fact that silver in sulphides may be dissolved by fine grinding and ordinary cyaniding methods, however, remains. Attention may be called to the discussion by G. H. Clevenger on "Mill and Practice of the Nipissing Mining Co., Ltd.," in *Bull. of A. I. M. E.*, July, 1914, in which the opinion is indicated that fine grinding of heavy silver minerals is advisable.

As to Mr. Van Zwaluwenburg's remarks on the necessity of air as an aid to cyaniding, his opinion is becoming general. In some cases air or oxygen will aid in shortening the time of treatment. In a few cases it will increase the ultimate recovery, but in many cases it has no effect.

I heartily agree with Mr. Van Zwaluwenburg's final comment. When the cyanide director is permitted to be a technical director, rather than a roustabout, more of our puzzling problems will be solved.—H. A. Megraw—*Eng. and Min. Journal*, Sept. 25, 1914, p. 925. (G. H. S.).

Cyanide Mill for Rochester, Nevada.—"The design of the new 100-ton cyanide mill for the Rochester Mines Co. at Rochester, Nevada, construction of which is now under way, presents some interesting features, particularly with reference to continuous counter-current decantation. The equipment of the sampling department comprises the following machines in sequence: Blake crusher, elevator, No. 3 Vezin sampler, Dodge crusher, No. 2 Vezin sampler, rolls, and No. 1 Vezin sampler. Metallurgical treatment will begin by crushing in solution with ten 1,550 lb. stamps. The stamp discharge will be classified into sand and slime in a Dorr classifier specially designed to give extra settling area, being deeper and longer than the standard machines. The grinding will be completed in two 5 ft. × 16 ft. tube-mills connected by clutches with a single 100 H.P. motor equipped with silent chain drive. The classifier overflow will contain probably 95% of −200 mesh pulp.



Scheme for Continuous Decantation at Rochester Mill.

The equipment for recovery of pregnant cyanide solution will consist of one Dorr thickener 28 ft. × 12 ft., followed successively by three Dorr agitators 28 ft. × 16 ft., and five thickeners 28 ft. × 10 ft. The agitators will be operated in series, with a drop of 4 in. between successive machines. The last five thickeners have, successively, 2 ft. greater elevation to provide for return of decanted solution by gravity. Diaphragm pumps will be used to elevate the thickened pulp through the series. The pumps will be driven from adjustable eccentrics on an overhead shaft, a 7 ft. rod connecting the diaphragm with the drive. This

scheme is expected to materially lessen the wear on the diaphragms due to lateral motion.

The general scheme for continuous decantation which was finally adopted after considering the conditions peculiar to this proposition, is diagrammatically represented in Fig. 1. It will be observed that there are six thickeners, five following the agitators instead of four as is customary in other installations. It was first proposed to use only five altogether, with double precipitation, precipitating not only the overflow from No. 4, but also that from No. 4. Under this scheme, the theoretical calculations indicated a loss of dissolved gold amounting to four cents per ton of ore treated, and a mechanical loss of cyanide equal to 0·61 lb. per ton of ore. The calculations for the present scheme, however, using six thickeners and single precipitation, indicated dissolved loss of about one and one-half cents per ton of ore, and a mechanical loss of cyanide equal to 0·62 lb. per ton of ore.

Comparison of the figures will show that the extra decantation obtained in the sixth thickener, reduced the dissolved loss and still gave about the same mechanical loss of cyanide. A further reason for the adoption of the extra thickener was the possibility of occasionally treating high-grade ore—up to $50 per ton—in which case the extra decantation would be a protection against loss of dissolved metal. It is interesting to note that a 15 H.P. motor provides power for all operations below the tube-mills."—*Metallurgical and Chemical Engineering*, Nov., 1914, p. 677. (G. H. S.)

AURORA CONSOLIDATED MILL, NEVADA.—"The tendency in milling practice towards heavy stamps, short tube-mills, and individual drive and general simplification is again evidenced in the description of the new mill of the Aurora Consolidated, Nevada, by Percy M. Cropper which appears in the Salt Lake Mining Review. The mill handles 800 tons of ore a day, yet it requires an operating force of only 18 men for 24 hours, including four men for a mill crew, a repair crew of four men and a metallurgist and superintendent.

In design and operation the mill has a number of new features and brings in some new things in mill practice. The combination of machinery is new. The stamps are much heavier than those usually used, being forty in number and driven in batteries of five; the tube-mills are larger in diameter and shorter in length, and worked in units of two. The Dorr classifiers are declared to be the largest thus far constructed. Where possible, all machinery is individually driven by motor, and there is an absence of line shafts.

At the mine trains of twenty cars each are used. As the cars pass over the tramway to the mill each car is weighed automatically as it moves along. The cars dump over steel ore bins for the coarse crushing plant, and are then sent around a circle back to the main track again.

Flow Sheet at the Mill. The ore from the coarse ore bins pass through a No. 7½ Gates crusher and then over a Gates screen. Here everything over 1½ in. passes on to two No. 5 Gates crushers that reduce it to 1½ in. This joins the undersize from the screen on a conveyor belt and goes to the head of the mill, where a Gates elevator carries it to the top of the sampling plant. At the sampling plant the ore is further crushed and resampled, so that a pound sample is secured from each ton, or a 500 lb. sample for each day. The ore from the sampling plant goes on to a cross conveyor that carries it to four steel bins, each having a capacity of 100 tons. From these bins the ore passes through feeders to the stamps which crush to pass through a quarter inch ring.

The stamps weigh 1,650 lb. each, which are probably the largest in use in the United States, although heavier have been used for some time in the gold fields of South Africa. Each battery of five stamps is driven by an individual back geared motor. It is not necessary to hang the stamps up as the motor has sufficient power to start them from rest.

From the stamps the ore passes to Callow sloughing off vats. From there it goes to the Dorr classifiers. These are six in number, and are declared to be the largest ever constructed. Here a line shaft, which is the only one in the mill, geared to a 5 H.P. back-geared motor, drives the classifiers.

The ore goes from the classifier to the tube mills. These are 6 ft. 16 ft., which is a great change from the old style of tube used in this class of work, being larger in diameter and shorter in length. Each, it is declared, is capable of handling 100 tons of feed a day from the stamps and crushing it to 200 mesh. The six tube mills are arranged in units of two. This arrangement allows them to be driven by units, two mills being run by a 150 H.P. motor equipped with outboard bearing and Morse silent chain drives.

From the tube-mills the material passes to a second Dorr classifier, where all 200 mesh is taken out, sending the +200 to a second tube mill. This has a closed circuit, and the material is reground until it all passes 200 mesh.

Cyanide Plant. The material then goes to three Dorr thickeners and is reduced to a consistency of one to one, water and cyanide solution. Then it goes through a series of five Dorr agitators and then to four Trent replacement machines. This process is something new in cyaniding, but has been used with a great deal of success in Arizona. In these the gold values in the cyanide solution are replaced with water. The pulp from the replacement machines then goes to three Trent filters, where the greater part of the cyanide solution is taken out and the pulp is allowed to go to waste.

The gold solutions from the Trent replacement machines and the Trent filters flow by gravity to the lower end of the mill, where they are pumped through Merrill clarifying presses. These are stationed on the thickener floor, from which point they flow by gravity through two Merrill precipitating presses. Here the gold is separated from the solutions and sent to the refinery. The cyanide solution is then pumped to the stock vats at the head of the mill, where it is enriched and started through the mill again.

The pumps are all assembled in one room on the lower floor of the mill, where one person can handle them all. They are all individually driven, as are the thickeners, agitators and replacement machines.

An inclined electric elevator for the use of the men in going up and down in the mill and for the conveying of supplies to the different floors is another feature of the plant. This works automatically, and can be run up and down from any floor. It is on the opposite side from the conveyor." PERCY M. CROPPER. *Mexican Mining Journal*, Dec., 1914, p. 352. (H. A. W.)

EFFECT OF GRAPHITE IN CYANIDE TREATMENT OF ORES. "Graphite in ores has interfered with cyanide treatment at several gold mining districts of the world. At Gympie, Queensland, Australia, there are several large heaps of tailing assaying about 82·50 per ton, but no serious attempt has been made to treat them. The Scottish Gympie Co. has two dumps of tailings, one of 60,000 tons or more, and mostly graphitic, and a smaller lot of 20,000 tons of what is called Inglewood tailing, derived from the Inglewood vein, which runs at right angles to the main lode series. The tailing is white and contains no graphite, and is particularly amenable to cyanide, the only consideration being the degree to which it is ground. The graphitic tailing, on the contrary, is not suitable for cyaniding, only yielding 40% or so after complete sliming. W. Macdonald, who studied the problem there for a year, came to the conclusion that this poor extraction by cyanide was not due to premature or re precipitation of the gold. On mixing the two classes of tailing, and subjecting the mixture to cyanide solution, and comparing the results with the results of similar tests carried out on each class separately, he found the extraction on the clean Inglewood tailing was just as good as when treated separately, showing that the graphite had not caused precipitation of the gold dissolved from the Inglewood ore. There is no chemical difficulty in treatment and the consumption of cyanide in both cases was abnormally low. A characteristic feature of the graphitic tailing is its yield of gold to amalgamation. By grinding very fine, practically all sliming, an extraction of upward of 70% of the gold resulted. The proportion of gold to silver is about 5 to 3, and it is likely that they exist in the form of an alloy." *Mining and Scientific Press*, Oct. 17, 1914, p. 608. (G. H. S.)

ELECTROLYTIC IRON. "Among the papers presented for reading at the abandoned Paris meeting of the Iron and Steel Institute was one by Prof. L. Guillet dealing with the commercial production of electrolytic iron. The author stated that Le Fer Company, of Grenoble, has taken out patents since 1910 for the electrolytic manufacture of iron. The method consists essentially in the use of a revolving cathode in a neutral solution of iron salts, the neutrality of which is maintained by constant circulation about the anode. Periodic additions of oxide of iron effect depolarization by partially removing hydrogen from the surface of the cathode, and permit the use of a current having a density as high as 1,000 amp. per square meter. From a solution made with ordinary pig iron a product can be obtained of a very high degree of purity. After removing occluded gases by annealing the metal contains an average of only 0·004% of carbon, 0·007% of silicon, 0·006% of sulphur and 0·008% of phosphorus. When taken from the bath the metal is hard, very brittle and contains occluded gases, notably hydrogen. Sir Robert Hadfield obtained 28·8 cc. of gases from a sample of the iron weighing 34 gm., having a volume of 4·33 cc., by heating it in a vacuum for four hours at 800° C. to 1,100° C. and for two hours more at nearly 1,400° C. The relative volumes of the gases were, hydrogen 18·8, carbon monoxide 7·4, carbon dioxide 0·2 and oxygen 2·2. The Brinell hardness of the crude iron is 193 with a ball of 10 mm. in diameter and a load of 3,000 kgm., and the micrographic structure is composed of exceedingly numerous fine needles, closely resembling martensite. Annealing in magnesia for two hours at 900° C. reduces the Brinell hardness to 90 and imparts a perfectly normal micrographic structure.

Tensile tests on tubes gave a breaking strength of 30·9 to 32·8 kgm. per sq. mm. (about 20 tons per sq. in.), with an elongation of 16·2%, to 42·1%, in the direction of the axes. Annealed tubes withstand compression tests producing more than ordinary deformation without exhibiting any sign of fracture. One tube 100 mm. in diameter and 0·75 mm. thick when subjected to a pressure of 1,200 lb. per sq. in. showed a deformation as regular as if it had been squeezed in a press. Another portion of the same tube was exposed for $2\frac{1}{2}$ months to a temperature of 120° in a steam boiler and then withstood a pressure of 1,200 lb. per sq. in. without any sign of fracture being perceptible.

Tubes are being manufactured by Messrs. Bouchayer & Viallet, whose plant is equal to an output of 100 lengths per day. The tubes are uniformly 4 metres long, and vary in diameter from 100 mm. to 200 mm. and in thickness from 1 mm. to 6 mm., and the great regularity in the thickness obtained contrasts strongly with the inequalities prevailing in the tubes made in the ordinary way, particularly when the thickness is less than 6 mm. The manufacture of sheets is still in the experimental stage, but hopes are entertained of their direct production without rolling. The quality of the iron is first rate, and as it can be worked with remarkable facility, it is especially suitable for stamping both as black annealed plate and as tinplate. The great purity of the sheets endows them with a particular value in the construction of electrical machinery by reason of the increased efficiency to be secured and the reduction in the weight of the material to be effected.

For the manufacture of steel the electrolytic iron appears likely to compete quite successfully with the best Swedish iron. Its quality is very regular, and its brittleness enables it to be broken into small pieces of any required size, and consequently it lends itself to a more uniform cementation than does ordinary material. Tool steel and special steel giving results at least equal to those obtained with similar steels made from Swedish iron have already been produced at several steel works. Working with a current density of 1,000 amp. per sq. meter a yield of two tons of very pure iron per kilowatt year is obtained; but when the density is reduced to 500 amp. per sq. meter the output per kilowatt year is nearly doubled, and on the latter basis the total cost of production, making ample allowance for waste of pig iron, depreciation, etc., is estimated at from £6 to £7 12s. per ton, according to local conditions.

It appears that the process for the production of electrolytic iron in tubes and sheets described in the paper by Prof. L. Guillet had, according to a statement in the *Times*, already been worked out in England by Mr. S. Cowper Coles, and that works are to be erected in the Midlands for the commercial manufacture of sheets, tubes, etc. In 1908 Mr. Cowper Coles read a paper before the Iron and Steel Institute in which he gave details of the electrolytic production of finished iron tubes and sheets in one operation. In his process he employed a mandrel revolving at a critical to obtain a close homogeneous deposit and to pass between the electrolyte and the deposited metal speed in order to give sufficient "skin friction"

mit the use of a high current density, whilst the solution was kept neutral by circulation over the anode."—*Metallurgical and Chemical Engineering*, Dec., 1914, p. 787. (G. H. S.)

RECENT PROGRESS IN THE DESIGN OF ORE-MILLING PLANTS.—"The authors discuss in detail improvements in ore moving and milling in the Western Mining Districts of the United States.

In milling progress has been made in the regulation of ore supply to the primary crushers. "In order to prevent choking and irregular feeding the bin openings are being made larger, 1 ft. wide by 6 ft. to 10 ft. long; the ore falls on to steel pan conveyors fitted with car wheels.

Belt conveyors are being introduced to remove large tonnages from primary and secondary crushers to fine grinding machines. These make possible the introduction at convenient points of automatic scales or "weightometers" and also of magnetic head pulleys for the removal of loose iron.

Sampling is usually done in connection with the feeding from the bins to the belt. A bucket of known capacity is passed through the stream of ore as it flows over the head pulley of the conveyor.

In connection with fine grinding, improved forms of Chilian Mills have been making some remarkable records.

In concentration, interest in largely centred in flotation processes. Flotation is generally regarded as best suited for the recovery of values from slimes. One large company has, after some months of testing, decided to take the feed for flotation direct from the fine grinders without intermediate treatment.

Various quick filters and separators are coming into use for the reclamation of mill water. The most noteworthy case is an enormous classifier on the Dorr system, 150 ft. in diameter, revolved slowly in a huge concrete vat, which will clarify all the water of a large mill.

For pumping, electric power has become almost universal.

In cyanidation, continuous decantation, including the counterflow system, is attracting most attention. For efficient aeration, vats lined with a minutely porous silicate tile through which air is forced are used.

Instead of building or remodelling a large mill on the basis of more or less uncertain expectations, it has recently become the practice to instal an experimental plant large enough to handle regular unit tonnages of ore. In this way the most efficient methods of treatment for particular ores are arrived at with some certainty."—COLBY M. AVERY and C. A. TUPPER.—*Mining and Engineering World*, Jan. 2, 1915, p. 1. (H. R. A.)

PROGRESS IN GOLD METALLURGY, 1914.—"Perhaps no work of recent years will be watched as closely by metallurgists as will this application of the accepted practice of the porphyry copper mills to the requirements of gold metallurgy. Breaking is to be followed by rolls, roughing tables, classifiers, regrinding plant, and concentrators, the aim being for concentrating efficiency rather than high recovery by amalgamation. Revolutionary as it seems, it may be inspired, and certainly not only the prestige of its sponsor but many strong features of the process itself merit the attention it will receive. To the onlooker this undertaking is particularly timely, since the increase in the fall ing weight of stamps has ceased, both abroad and in this country, where we understand that tests by one of the largest producers have indicated 1,250 lb. as maximum falling weight obtainable without loss of efficiency.

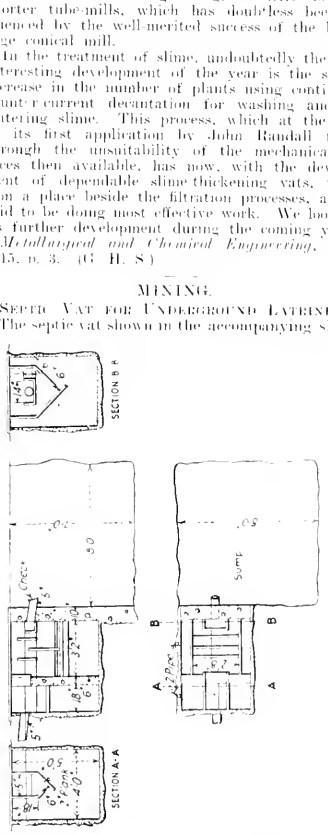
With the continued development and widening application of cyaniding, amalgamation has in many instances been abandoned; in many others reduced to a position of minor importance. It is said that a good mine is a long time in dying; the same may be said of a good process, and we feel that amalgamation, far from being moribund, has its distinct field, doubtless narrower than formerly and yet of sufficient consequence to justify its consideration as a live process.

We note, in fine grinding, a drift towards shorter tube-mills, which has doubtless been influenced by the well-merited success of the Hardinge conical mill.

In the treatment of slime, undoubtedly the most interesting development of the year is the steady increase in the number of plants using continuous counter-current decantation for washing and dewatering slime. This process, which at the time of its first application by John Randall failed through the unsuitability of the mechanical devices then available, has now, with the development of dependable slime thickening vats, fairly won a place beside the filtration processes, and is said to be doing most effective work. We look for its further development during the coming year."—*Metallurgical and Chemical Engineering*, Jan., 1915, p. 3. (G. H. S.)

MINING.

SEPTIC VAT FOR UNDERGROUND LATRINES.—"The septic vat shown in the accompanying sketch



was designed by me for the Mond Nickel Co.'s Frood Extension Mine at Sudbury. The Ontario mining law requires the provision of underground closets,

and the Provincial health laws forbid the discharging the untreated sewage into running streams.

At the best the ordinary box closet is a nuisance, and there are many objections to running untreated sewage through the pumps to the surface. To overcome the objections to these types of closets, it was decided to introduce a septic vat. The stools flush automatically and discharge through a pipe into the vat, which may be located at any point provided there is sufficient fall from the vat to the pump sump. The sketch shows the ideal location, since the excavations for sump and vat may be taken out at one operation.

With septic vats, as ordinarily constructed, it is found necessary to clean out the sediment that is deposited from once to twice a year. To obviate the necessity of doing this cleaning by hand, iron pipes having their ends drawn to flat nozzles are inserted in the concrete lining of the vat. The pipes are bent so as to deflect a stream of water toward the centres of the chambers. A bypass from the pump connects with the wash pipes, and sediment may be washed into the sump, where it will remain long enough in suspension to be discharged by the pump before settling. The vat shown was designed to accommodate a force of one hundred and fifty men per day."—H. G. PICKARD. *Engineering and Mining Journal*, Jan. 16, 1915, p. 149. (G. H. S.)

MISCELLANEOUS.

A PRIMARY BATTERY. "The *Irish Builder and Engineer* writes the following interesting note:—It is reported that a primary battery, in which the negative element consists of plates of lead amalgam, obtained by pouring 10 gm of mercury into 90 gm. of molten lead, and the positive element consists of carbon, is giving some interesting results. The electrolyte is composed of a solution of 80 cub. cm. of sulphuric acid and 120 cub. cm. of nitric acid in 1,000 cub. cm. of water. When in operation the lead amalgam is attacked and lead sulphate formed, which being insoluble drops to the bottom of the cell, in company with some sulphate of mercury and a few mercury globules. The resistance of a cell tested varied with the current strength from 0·03 ohm with 0·7 ampere current to 0·128 ohm and a 5 ampere current, the lead consumption being about 5 gm. per ampere hour. Another test was made on a cell with four positive and three negative plates, the surface of the latter being 900 sq. cm. The internal resistance was found to be 0·022 ohm under a 5 ampere discharge, while the capacity of the cell without the addition of further acid was 112·5 ampere hours. One of the more important features of the battery is its constancy." *Indian Engineering*, Oct. 17, 1914, p. 231. (J. A. W.)

PAINT PROTECTION FOR PORTLAND CEMENT SURFACES. Thirty-five samples of paint were applied to concrete surfaces and exposed in the open for two years. Paints of white lead, or zinc oxide, or both, ground in pure linseed oil, or of both ground in mixtures of raw and boiled linseed oil or in treated China wood oil and thinned with turpentine, in general gave good results. Paints made with oil varnishes containing acid resins, spirit varnishes, and glue or casein paints were unsatisfactory. When the concrete surface is damp, a zinc sulphate priming, to neutralize free lime, should be applied. Oil paints containing wear-resisting pigments were successfully applied to cement floors. H. A. GARDNER. *Journ. Soc. Chem. Ind.*, Oct. 31, 1914, p. 1018. (J. A. W.)

Abstracts of Patent Applications.

(C.) 22514. James Wilshir (1), W. Martin (2). Improvements in hose connectors and the like. 23.5.14.

This invention relates to connectors or couplings for rapidly connecting lengths of pipe, whether metal or hose, or for connecting such pipe to a machine or for similar purposes.

(C.) 71 15. Otto Karl Quast. Improved amalgamating apparatus and process. 29.3.15.

This application refers to improvements in apparatus for the extraction of precious metals from ore by amalgamation. The apparatus consists of means for supplying mercury and cyanide solution to a trough containing the ore and washing same through a vertical separating column containing a hollow screw conveyor. The mixed pulp flows through the conveyor, and the mercury and amalgam collect at the bottom, the residue being elevated therefrom by the conveyor and discharged.

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(COMPILED BY THE SECRETARY, MR. FRED. ROWLAND, F.C.I.S.)

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This paper will be submitted for discussion at the [illegible] to be held [illegible] September, 1915.

RECENT INVESTIGATIONS ON DUST IN MINE AIR AND THE CAUSATION OF MINERS' PHTHISIS.

By James Moir, M.A., D.Sc., F.R.S.S.Afr.

The Committee on Miners' Phthisis Prevention has now been sitting for over three years, and I think the time has come for the publication of such of the new scientific methods of the Committee as I am myself responsible for, together with a synoptic view of the main results achieved. They deal mainly with the size, number, and other properties, of silica dust as found in mine air. This paper was written in May, but publication has been delayed because the Committee had not presented the results to the Minister of Mines, whom I have now to thank for permission to publish at once.

(1) *Rate of falling of quartz particles in air.*—According to Stokes' formula, the velocity of fall, given by $v = \frac{2}{9}\frac{g\delta a^2}{\eta}$ (where g = 980 cm./sec, δ = density = 2·65, η = viscosity of air = $\frac{1·9}{10^4}$ at 20° and 760 mm.) depends on the square of the radius (a) of the particle (assumed spherical). Hence a 1 micron diameter particle (radius 0·00005 cm.) falls at a rate of only 0·0075 cm. per second, whilst a 10 micron particle falls at 0·75 cm. second. Hence to fall through 5 ft., say in a drive, will require 3½ minutes for a 10 micron particle, and no less than 5½ hours for a particle of 1 micron diameter. These are, of course, only approximations for the conditions of the Rand Mines, as the viscosity of air here is probably 5 less than at sea-level, and in addition the effect of the particles not being spherical would increase the friction, and consequently increase the time required to fall; these factors consequently partly cancel one another.

It is obvious that the smallest particles formed in dry-drilling and by blasting cannot possibly settle at all in the eddying atmosphere of the mines, and if they come out of the mines at all, it must be owing to ventilation removing bodily the air containing them; dry-drilling is, of course, illegal.

(2) *Size and weight of quartz-particles found in mine air.*—In order to calculate the weight of such small particles (which are quite unweighable), it is necessary to assume a standard or average shape for them. My assumption is that they are rounded tetrahedra, and that therefore their volume is intermediate between that of spheres of the same diameter and that of tetrahedra of the same greatest diameter (seen in projection under the microscope). Examination of mine dust under the microscope, including determination of the thickness of large particles by their polarization-colours, confirms this assumption, though the polarization of particles smaller than 15 microns is too faint to be of any use. The volume of a sphere of diameter d is approximately $d^3/2$, and the volume of a tetrahedron of projected diameter d is approximately $d^3/6$, so for the dust-particles I strike an average and assume the volume to be somewhere between $0·3d^3$ and $d^3/3$, when d is measured from the most prominent point of the particle to the middle of the opposite side (or curve). Take the case of a quartz particle measuring thus 10 microns (0·01 mm.) across. Its weight in grams will be 2·65 times its volume in cc., which assuming

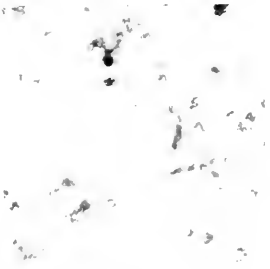


Fig. 1. Dust from lung of miner who died of silicosis, not ignited × 900. The square is 50 microns.

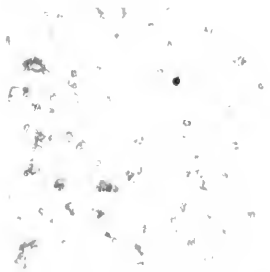


Fig. 2. Blasting dust × 900. The square is 50 microns.

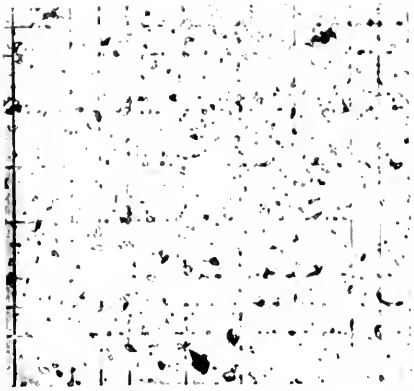


Fig. 3. Blasting dust × 200. Average sample of squares are 50 microns each.

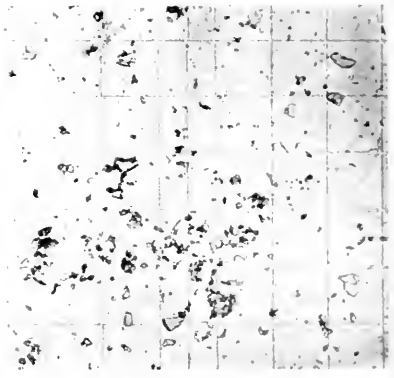


Fig. 4. Dust from atomized mine water used for drills × 200.

$1 - 0.3d$ gives $\frac{2.65 \times 0.3}{10^{3}}$ gm., or, say, $\frac{8}{10^{7}}$ mgm. (or, assuming $1 - \frac{d}{3^{3}}$, $\frac{9}{10^{7}}$ mgm.). Striking an average therefore, there are about 1,200,000 such particles to the mgm.*

In actual mine air the average size of the floating particles is less than 10 microns (probably 3 to 4 microns is a fair average) so the number of particles in a mg. of dust is correspondingly greater. Thus the smallest measurable particle (1 micron diameter) is so extremely light that it would require 1,200,000,000 of them to weigh 1 mgm. It should be noted, however, that many of the smallest particles in mine air, particularly after blasting, are not quartz, but come from the aluminous matrix of the banket; particles of sericite, chlorite and rutile or anatase are comparatively frequent in air after blasting, and when water-blasts are in use seem to resist wetting more than quartz particles do, and consequently remain unlaid. Analysis of this unlaid dust (caught in sugar) shows also a great increase in the percentage of alumina as compared with that present in the ore as a whole.

The very small size of dust particles in mine air is the main reason why respirators are of so little use as a protection underground. I have examined all the commer-

* This is a minimum estimate, as no allowance for flat or narrow particles is made.

† The wavelength of red light is 0.7 micron, consequently owing to diffraction effects the microscope cannot define particles much smaller than 1 micron.

cial makes available here, and find that the openings in the fabrics used are all much wider than 10 microns, one popular article, for example, having apertures up to 250 microns in diameter! Some idea of what a 10 micron particle is may be gained from the fact that 200-mesh screening has apertures of about 70 microns each in diameter, and 260-mesh, the finest made, has 50-micron openings.

(3) *Fall of quartz-particles suspended in water.*—This may again be calculated from Stokes' formula, putting in the viscosity of water for η in the formula (also $\delta - 1$ instead of δ) and modifying the result as before to suit the non-spherical nature of the particles. I find that the formula $V = \frac{7d^2}{100000}$ where V is the velocity of fall in cm. per second and d is the diameter in microns, gives very nearly perfect results. Thus a particle of 20 microns diameter is calculated to fall at 0·028 cm./sec., which agrees exactly with H. Stadler's observations; so a 60 micron particle falls at the rate of 0·252 cm. sec., which again agrees exactly with Stadler's observations. On the other hand, a 10 micron particle is calculated to fall at a rate of 0·007 cm. sec. whereas Stadler, working with Rand slime, found 0·010. I understand that Stadler explains this by a predominance of very fine pyrite in such slime, which through its density would give an abnormally high velocity of fall to such slime; but the measurement of such extremely low velocities is probably not very accurate, so that the discrepancy may not be significant.

Advantage has been taken of this knowledge to invent a process for sorting out mine dust, as caught from the air in sugar-tubes, into large and small particles. It was discovered by Dr. J. McCrae some years ago that all the silica particles observable in the lungs of fatal cases of miners' phthisis were exceedingly minute, generally under 3 microns in diameter, with none larger than 12 microns, and very few above 8 or 10 microns.* Consequently it became desirable in dealing with dust samples from air to effect a separation of all particles larger than 12 microns in diameter which had happened to be caught floating in the air, but which are presumably too large to be retained by the lungs and so cause miners' phthisis.

This process depends on causing the 'larger' particles to settle by allowing the suspension of the dust in sugar solution to stand for a definite time, calculated from the viscosity of the sugar-solution and confirmed by microscopic observation of the sediment. The 'very fine' particles, viz., those of 12 microns and less diameter, remain floating and are then filtered off for gravimetric determination in the usual way. The sugar used to catch the dust is a mixture of 90 very coarse and 10 fine, this does not clog in a moist atmosphere. The quantity of water taken to dissolve the sugar is chosen so as to give a 16 % solution by weight, *e.g.*, 40 gm. of sugar are made up to 220 cc. with warm water.

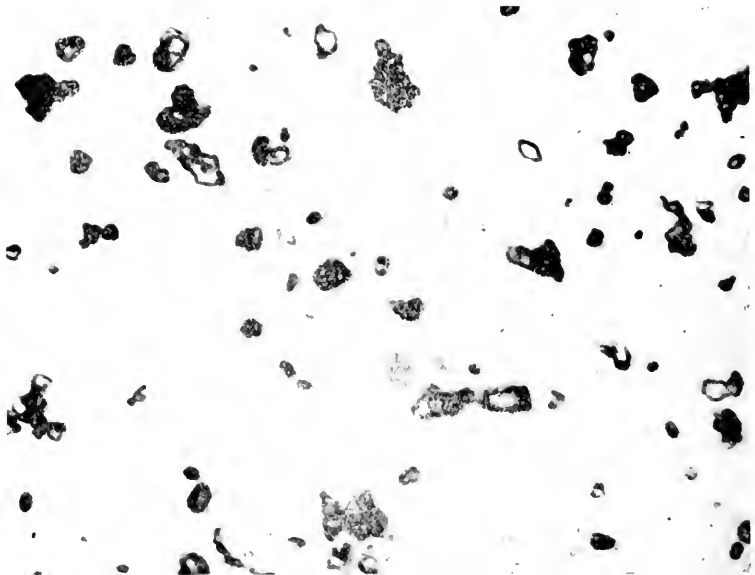
It is next filtered through a 260-mesh screen to remove very large particles over 50 microns in diameter, and the temperature adjusted to about 10° C.; under these circumstances the friction is twice that of water, and after stirring, the solution is allowed to stand for 4 minutes per cm. height of liquid in the vessel, when all the 'large' particles (and some of the finer ones) will have settled, and the upper liquid is removed by means of a siphon previously filled and fitted with a clip. Thereon the sediment is mixed with water up to the original bulk and allowed to stand for 2 minutes per cm. depth, *i.e.*, half the original time, which floats nearly all the remaining 'fine' particles, and the siphon is again used. The second sediment consists almost entirely of particles larger than 12 microns and can be filtered off, ignited and weighed as usual; whilst the two siphoned-off liquors are united and also filtered off to give the weight of 'very fine' dust. If a portion has been removed at the beginning for the purpose of counting the particles on the hemocytometer, an appropriate correction of the gravimetric results must be made. Thus if 5 cc. have been removed from 220, the correction is 5 in 215 or 1 in 43.

By counting the number of particles in the sugar solution from a known volume of mine air made up to a known volume of the sugar solution (the hemocytometer is used, which enables the number of particles in $\frac{1}{16}$ cubic mm. of fluid to be counted) we can calculate the number of quartz particles in the original air; this number varies from 10 per cc. in dusty street air to 90,000 per cc. in mine air after blasting.

If we count the 'very fine' particles and also determine their weight in mg. by

* See Bulletin I of South African Institute of Medical Research, Johannesburg, 1913.

FIG. 5.—BLASTING DUST FROM BLADES OF EXHAUST FAN. ×1160 DIAMETERS.



the above process, we get by division the number in a mgm., whence we can calculate their average diameter, using the formula $D = \sqrt[3]{\frac{1200^*}{N}}$ where N is the number of millions of the particles in a mg. and D is got in mikrons. For example, in air from dry-drilling, the particles are very small, and about 100 million are required to weigh 1 mg.; hence $D = \sqrt[3]{12} = 2{\cdot}3$ microns, which is rather less than one-third of the diameter of a hemocyte (red blood-corpuscle).

The classification into large and small may be carried much farther with the hemocytometer. Thus in one case—drilling under standard watering conditions—I found that 99·10 % of the number of particles were under 6 microns, whilst 0·72 % lay between 6 and 12 microns, 0·16 % by number lay between 12 and 25 microns, and only 0·02% were over 25 microns: the first class amounted to 300 per cc. of air sampled. Going by weight, however, the results were quite different, the percentages of the four classes being 64, 16, 11 and 9 respectively: the calculated number of particles per mgm. weight was respectively 87,000,000, 2,500,000, 900,000, and 120,000, showing that, though very few in number, the large particles appreciably affect the weight of the whole. Corrections were found necessary for dust contained in the sugar used. This was found to contain about 10 million particles of silica in the 40 grams used, which, with the filter-ash, weighed exactly 1 mg. on the average.

Having now described the new experimental methods, I may pass on to some of the practical results. The first and most important of these is an investigation into the question whether the methods in use of 'laying' the dust are capable of laying the very fine particles as well as the coarser ones. The result was satisfactory, and showed that *when the laying devices were in order and conscientiously used*, there was no predominance of 'very fine' dust in the air which had undergone treatment; the percentage of 'very fine' dust, going both by weight and by number, in such air was no higher than in the original untreated air. Thus, in drilling when the point of the drill is wetted by having a jet of water directed into a 'down' hole, the amount of dust getting into the air is only 2 % or 3 % of what happens during dry-drilling, and this is the case whether one goes by weight or by number of particles per cc. of air. In fact, it is surprising to find how little variation there is in the percentage by weight of 'very fine' (under 12 microns) particles: they constitute about two-thirds of the total weight of dust in all sorts of mine samples from different sources, and it is only in certain samples taken above ground, *e.g.*, during dust-storms, that a deviation from this rule can be found, the proportion of 'very fine' dust then being insignificant, which is the main reason why street dust does not cause miners' phthisis, despite its appalling quantity in this country, for, unless the electrical factor is present as in dust-storms, not very much dust is raised (speaking gravimetrically) even by a very high wind.

It should be noted that the practice of playing a jet of water at random over the working-face is not satisfactory. In such a case the smaller sizes of dust are not laid, and their proportion by number in the air becomes twice or thrice what would be expected from their weight, so that even a low gravimetric figure for the dust in such air (*e.g.*, 5 mg. cub. metre) is no guarantee of safety. A similar criticism, of course, applies to coarse sprays of water, and to certain miniature water-blasts used after blasting the 'cut,' which are quite incapable of catching dust particles to the number of, say, 50,000 per cc. of air, of which 99·9 % are of the very minutest size. The thing to remember is that there is practically no class of underground air in which the dust is coarse in character; practically all show that over 97 % of the number of particles observable are of the 'very fine' (and dangerous) class.

What the lungs have sometimes to contend with underground may be seen from the fact that during light work about 7,000 cc.† of air per minute are usually breathed in, and three times as much during hard work. It follows from this that it is difficult to find any place underground where the number of particles inhaled is less than a million per minute, whilst the case of those who go in for dry-drilling is much worse, since they may inhale 60 or 70 million per minute, or 3 or 4 million in every breath! As for the person who goes in immediately after blasting, which used to be quite common, he gets at least 10 million and sometimes 30 million particles per breath, and obviously no lungs can stand that. The air of a dust-storm may treat one to 10,000 particles per breath, and most of us think that is quite-

* This factor is a minimum value and may possibly require to be doubled in practice.

† Haldane puts this figure four times higher.

Fig. 7. UNLAID (*i.e.*, WATERED) DRILL DUST FROM AIR OF MINE. × 1160 DIAMETERS.

The squares are 50 microns each, or $\frac{1}{5}$ $\frac{1}{6}$ inch.

1 Nature of Sample.	2 Number of Silica Particles per cc. of Air.	3 Weight of Dust under 500 in mgm. per cubic metre.	4 Number of Millions of Particles in 1 mgm. Weight.	5 Calculated Average Diameter of Particles in Microns.	6 Weight of "Very Fine" Dust per cubic metre.	7 Percentage by Weight of "Very Fine."	8 Percentage by Number of "Very Fine."	9 Average Diameter of "Very Fine" in Microns.
A. (1) From air of Johannesburg, Fordsburg dip, near dump (1), strong south wind (2), during dust storm ...	(1) 100	25	31	7	4·0	16	87	3·8
	(2) 25	17	12	9·3	0·1	0·5	50	2·0
(2) Street, north wind, no electrical disturbance (twice)	(1) 35	4	9	5·0	2·4	60	91	4·3
	(2) 40	4	10	5·0	2·4	60	90	4·3
B. Mining road air disturbed by motor cars, Germiston (mainly quartzite and dump dust) ...	500	14	35	3·25	7	50	90	2·65
C. Dry-drilling (average of six)	9,500 (3,000–20,000)	124 (60–200)	75 (40–100)	2·52 (2·2–3·1)	89 (50–130)	72 (65–78)	98·9 (97·7–99·4)	2·20 (1·95–2·45)
D. Dry blasting—								
(1) 5 minutes after firing cut ...	86,000	503	171	1·91	470	93·5	99·85	1·87
(2) 15 minutes after firing cut	19,000	170	110	2·22	136	80	99·5	2·05
(3) Inside doors at station (three different mines) ...	62,000	366	169	1·92	331	90·5	99·72	1·86
E. Drilling with jet of water *in* hole, or can-watering of down holes (average of 5)	210 (80–330)	4·1 (2–6·4)	50 (30–55)	2·9	3·1	75	97·5 (95–99·5)	2·13
F. Drilling with promiscuous jet, bottom of hole usually dry (average of 9) (naturally very variable)	1,120 (400–2200	14 (3–30)	80 (33–140)	2·46	8 (2–13)	57 (30–75)	97·3 (94–99·7)	2·07
G. Dry "collaring" of holes (average of 3)	3,700	49	75	2·52	28	57	98·4	2·10
H. Blasting cut—								
(1) followed by 30 minutes best water blast	650	8·1	80	2·47	5·5	68	99·0	2·16
(2) 15 minutes later	350	5·7	61	2·70	4·2	69	99·3	2·44
I. (1) Crushers and sample houses on surface, 1913 ...	1,200	40	30	3·42	27	68	97·3	3·03
(2) Best 1915 conditions	(50–500)	(2–8)	(25–60)	(3–3·5)	(1·2–6)	(60–70)	(96–99)	3
J. Leyner drills (in good order) (average of 10) (very irregular)	210 (40–1000)	3·4 2–5)	62	2·65	2·7	79	99·0	2·50

NOTE.—When latter are defective and cause a mist by atomising the water, the count goes up to 1,500 or 2,000. The feed-water on a certain mine was once found to contain 23,000,000 particles per cc.!

The figures in parentheses are maximum and minimum values observed.

enough, and that our lungs are not constructed to get rid of much more than that.

The air of crushers and sample-houses comes out badly in the table, but I may remark that this kind of dust is distinctly coarser, and in addition is not breathed continuously by the employés; the underground men gets in addition an occasional dose of CO and of tubercle germs, and therefore suffers more from a general sort of dust than a surface man; all this explains the comparative immunity of crushers-house men from miners' phthisis.

Some investigations were also made in connection with my work on carbon monoxide in mine air due to the use of explosives. It is necessary to point out that I was the first to discover this property of gelatinous explosives (see Report of first Transvaal Commission on Miners' Phthisis and this *Journal* for 1903) as I find that text-books tend to ascribe it to Weishopt or Cullen or E. V. Mann, who elaborated the discovery about five years later.

The recent work comprises 17 accurate analyses for CO_2 and CO in samples taken during 1911 at the places where men want to be hoisted after blasting, being part of a general survey of the Witwatersrand mines. The average value of the CO_2 was 0·218 %, and of the CO, 0·00156 % (the iodic anhydride method was used on 3540 c.c. samples), giving an average ratio of 16 for CO_2:CO. The maximum CO_2 met with in such return air was 0·650 % and the minimum 0·11 %; whilst the CO varied between 0·0078 % and *nil*, nearly all being below 0·001 %. The ratio was extremely variable, six samples giving values under 10 (chiefly due to powerful spraying which dissolved the CO_2 and left the CO), whilst certain outcrop mines gave values of 200 or 300. All these high CO's, probably exceptional, be omitted, the average ratio of CO to CO_2 in the returning H mines is about 20.

In the previous year a survey of after-blasting gases was done over the Rand, 63 samples being taken 30 minutes after firing the cut. A water-blast was supposed to be in use, but sometimes was cut out or else too small to be of any use. The average CO found in these 63 samples was 0·316 %, the maximum figure being 1·50 % and the minimum 0·10 %. The CO_2 average was 0·028 % (done by combustion in Lévy apparatus, confirming by the blood test when desirable); the maximum CO found was 0·11 %, and the minimum a trace. The average ratio was 11, varying from 1 to 2·0 or 3·00.

Two or three tests for nitrous fumes were made, all giving blank results, as was to be expected 30 minutes after blasting in the presence of water.

Ten special samples taken 30 minutes after the "cut" in a completely unventilated area of a certain mine gave CO 1·046 and CO 0·12 on the average, which shows that 30 minutes is not long enough to wait after blasting under such circumstances. The worst gas encountered in this set gave CO_2 1·10, CO 0·20, O 17·80, the latter being about the lowest figure for oxygen I have met with in a Rand mine.

Mr. Jas. Gray, F.I.C., has done the most of the difficult gravimetric work in connexion with this research. I also beg to acknowledge most valuable suggestions from Mr. E. N. Kotzé, the Chairman, and from Mr. A. McA. Johnston, my colleague on the Committee.

Obituary Notices.

The following losses by death are recorded:

Mr. H. F. Olds.

Mr. Henry Frances Olds, M.Inst.M.M., was a Life Member of this Society, which he joined in September, 1903.

From 1891-93 he was studying mining at the Camborne School of Mining, Camborne, Cornwall, and completed his training as a mining engineer at the Royal School of Mines, London, where he remained until 1896.

His first professional engagement took him to India, where he was engaged in prospecting work, for about two years. Upon the termination of this appointment he took over the management of the Cape Asbestos Co., Ltd., in Griqualand West, which he occupied for about a year, leaving in 1901 to manage the Rhodesian gold properties of De Beers. Later he had charge of the Bechuanaland properties also. He remained with De Beers until 1908, when, returning to England, he resided in Cornwall for some time.

He left England again in 1910 to go to the Lone Star Mine, Gadzema, Rhodesia, where he remained for about 18 months. In 1911 he transferred his activities to the Kimberley Tyne Gold Mining Syndicate at Hartley, Rhodesia, and towards the end of the following year he left South Africa and returned to England. He again resided in Cornwall, and it was whilst engaged in some mining

operations there that he met with the accident which resulted in his death on November 18th, 1914. His wife and other members of the family have the sincere sympathy of the Society in their bereavement.

Mr. C. A. Ferguson.

Mr. Colin Armstrong Ferguson joined the Society as a member in October, 1910, and died on the 18th July, 1915. Mr. Ferguson was born at Victoria West, Cape Province, and was educated at Rondebosch and in England. After several years' work on the Langlaagte Block B. G.M. Co., he was appointed manager of the Porges Randfontein Mine in 1905, and succeeded to the General Managership of the Randfontein Estates, Ltd., in 1911, which position he held until his death. Mr. Ferguson, who was a brother-in-law to Sir J. B. Robinson, was 42 years of age, and was highly esteemed by all those with whom he came in contact. His death is deeply regretted, and to his wife and family the Society expresses its deepest sympathy in their loss.

Mr. R. G. Bevington.

In Mr. Richard George Bevington the Society loses one of its Foundation Members. He was elected in April, 1894, and since 1902 was continuously a member of the Council. In 1908-9 he held the position of President of the Society, and was always an active member, being especially interested in the Society's finances, and a member of the Finance Committee for many years. Mr. Bevington, who was 55 years of age, was born in London, and came to South Africa in 1886. After several years in the outside districts, he was appointed Mill Manager of the Meyer and Charlton mine, leaving to occupy the post of Mill Manager on the Village Main Reef mine in 1896. He was one of the few men on the Reef to hold one position for nearly 20 years. He was an active Freemason, and an Executive member of the District Grand Lodge, S.C., a member of the Rand Pioneers, a member of the Institution of Mining and Metallurgy.

During the Anglo-Boer War Mr. Bevington was a Lieutenant in the Rand Rifles Mine Guard, receiving the Queen's medal.

During the snowstorm on the 19th July he visited the mill late at night, and in returning home he slipped on the snow, and fell some 40 feet, fracturing several ribs and severely cutting his head, injuries which resulted in his death on the 23rd July. The funeral at Brixton at which both the Masonic and Church of England services were held, was largely attended by members of the Society.

To his widow, his son and daughter, the Society tenders its sincere condolences in the great loss they have sustained.

Mr. F. N. Gelds.

Lieut. Frederick Niven Gelds, A.R.S.M., St.Inst.M.M., joined the Society as a member in April, 1912, when practising as a Mining Engineer at the Cam and Motor Mine, Rhodesia. He left for England in April last, was gazetted as a 2nd-Lieut. in the 176th Tunnelling Company of the Royal Engineers shortly after his arrival, and was killed in action in France on the 1st June. Much sympathy is expressed with his mother and sister in their bereavement.

Contributions and Correspondence.

LOSS OF CYANIDE AND VALUE OF PROTECTIVE ALKALI.

At the request of the senior Vice-President, I attach for publication a table giving the amount of hydrolysis of pure sodium cyanide when dissolved in water. This hydrolysis is due to the weakness of HCN as an acid, which results in NaOH and HCN existing uncombined in any solution of NaCy. The table gives the quantities and percentages of HCN and of NaOH for any strength of total cyanide. In presence of zinc cyanide the hydrolysis is less, but I am at present unable to say how much.

(1) Table giving hydrolysis of pure NaCN into HCN and NaOH (at summer temperature; results 10 to 15 lower in winter).

Let Y = total cyanide as KCN.
x = part hydrolysed to HCN, but calculated as KCN.
Z = part hydrolysed to NaOH calculated as NaOH.

Y	x	$\frac{100x}{Y}$ = % of whole	Z	$\frac{100Z}{Y}$ = % of whole Y
1·0	0·0077	(0·8)	0·0047	(0·5)
0·5	0·0054	(1·1)	0·0033	(0·7)
0·3	0·0042	(1·4)	0·0026	(0·9)
0·2	0·0034	(1·7)	0·0021	(1·1)
0·1	0·0024	(2·4)	0·0015	(1·5)
0·05	0·0017	(3·4)	0·00105	(2·1)
0·02	0·0011	(5·4)	0·0007	(3·3)
0·01	0·00078	(7·7)	0·0005	(4·7)
0·005	0·0005	(11)	0·00033	(6·7)
0·001	0·00024	(24)	0·00015	(15)

(2) In presence of extra alkali:

There is some doubt whether Chemell's process tried on *pure* KCN would indicate the alkalinity due to hydrolysis. Assuming that it does *not* do so, but only *added* alkalinity, the formula $x = \frac{Y}{275 \cdot 01 (W - Z)}$ gives the HCN (calculated as KCN in presence of "Chemell-alkalinity") of W. Thus if W = 0·01 NaOH and Y = 0·5 KCN, Z from above table is 0·0033, total alkalinity W − Z = 0·0133 and x_1 (= HCN calculated as KCN) = 0·5 / 2·75 (0·01 − 0·0033) = 0·5 / 0·0184 = 27·2 KCN. (For hydrolysis in absence of NaOH, see *a* above.) The following table has been constructed from this formula for all common strengths of cyanide.

Strength of Cyanide in KCy per cent.	Observed Alkalinity as NaOH per cent.	Amount of HCN lost by Hydrolysis, calculated as KCy per cent.
0·2	0·03	0·00023
	0·02	0·00033
	0·01	0·00060
	0·005	0·00102
	0·002	0·00177
0·15	0·03	0·00017
	0·02	0·00025
	0·01	0·00046
	0·005	0·00080
	0·002	0·00141
0·10	0·03	0·00012
	0·02	0·00017
	0·01	0·00032
	0·005	0·00056
	0·002	0·00104
0·05	0·03	0·00006
	0·02	0·00009
	0·01	0·00016
	0·005	0·00030
	0·002	0·00060
0·02	0·03	0·000024
	0·02	0·000036
	0·01	0·000072
	0·005	0·000144
	0·002	0·000352

Roughly speaking, the addition of alkali equal to 1·10 of the cyanide causes the recovery of 90 % of the lost cyanide.

JAMES MOIR.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RAPID IODIDE METHOD FOR COPPER.—"Chemists who use the iodide method for copper assays usually employ it on account of its rapidity. This method can be appreciably shortened by avoiding the use of ammonia or sodium carbonate to neutralize the nitric acid and subsequent acidification with acetic acid in the following manner. After solution and removal of all nitrous fumes, by boiling, the solution is diluted to about 250 cc. and about 15 cc. to 20 cc. of a saturated solution of zinc acetate added and the titration carried out as usual after addition of about 10 times the weight of copper present of potassium iodide. The end point with starch can be determined without difficulty, but will not remain permanent for more than five or 10 minutes. E. A. SMALE.—*Eng. and Mining Journal*, Feb. 6, 1915, p. 285. (G. H. S.)

RAPID DETECTION OF CYANIDES, THIOCYANATES, FERRO AND FERRICYANIDES IN MIXTURES.—"Solutions to be tested are treated with an excess of zinc nitrate, warmed, and filtered. Thiocyanates remain in the filtrate, and cyanides, ferrocyanides and ferricyanides can be detected in the washed precipitate by spot tests made on filter paper with small quantities of the precipitate and strong acid solutions of mercurous, ferric, and ferrous (free from ferric) salts respectively. The tests are not affected by the presence of fluorides, phosphates, borates, sulphates, chlorides, bromides, iodides, chlorates, carbonates, nitrates, or silica, but chromates and ammonium salts must first be removed. In presence of borates, mercurous salts produce a green coloration which cannot be confused with the black colour given by cyanides. The test with mercurous salts is useless in presence of sulphides, sulphites, or compounds which reduce the reagent. In such cases cyanides may be detected by warming gently some of the washed zinc precipitate with dilute sulphuric acid in a porcelain crucible covered with filter-paper moistened with ammonium sulphide solution, and afterwards testing the paper for thiocyanates by means of acid ferric chloride solution." F. FEIGL, *Chem.-Ztg.*, 1914, *38*, 1265.—*Journ. Soc. Chem. Ind.*, Feb. 15, 1915, p. 135. (J. A. W.)

METALLURGY.

FLOW OF SAND AND WATER THROUGH SPIGOTS.—"It is common practice in ore-dressing operations to pass pulps of sand and water through spigots, as in jig hutches, classifier pockets, etc. 'With a given quantity of solids per unit time, of a given specific gravity, to be discharged with water under a given head, and with a given ratio of water to solid, what must be the size of the spigot opening?' Such is the nature of a problem investigated by Prof. R. H. Richards, of Boston, Mass., and Mr. Boyd Dudley, junr., of State College, Pa. Their results are published in the January *Bulletin* of the A.I.M.E. If a spigot is to be opened continuously, there are, aside from its form, three factors governing the rate at which it will discharge a given mixture, viz., the head of

water above the spigot, the area of the spigot opening and the viscosity of the material discharged. By the term viscosity as used here is meant the ratio of the volume of pure water that will flow through a given orifice under a given head in a given time to the volume of the material under consideration that will flow through the same orifice under the same conditions.

The head of water above a spigot is usually fixed; so with the head known, in order to determine the size of the spigot required, it is necessary to know the viscosity of the desired mixture, and the co-efficient of discharge of the form of spigot employed when pure water is discharged. From these data the area of the spigot can be calculated from the equation

$$a = c\frac{fq}{\sqrt{2gh}}$$

in which a is the area of the spigot opening, f being the viscosity of the mixture, q the rate of discharge by volume, c the co-efficient of discharge, g the acceleration due to gravity and h the head of water above the spigot. The co-efficient of discharge is the ratio of the actual rate of discharge of pure water to the theoretical rate as calculated from the effective head above the spigot. The value c for forms usually found in classifier spigots under heads of from 1 ft. to 3 ft. is between 0·87 and 0·91, the higher values for lower heads.

The authors studied the relation between the composition and viscosity of sand-water mixtures. The sp. gr. of the sand was 2·72. Experimental results are given in the table.

Relation of Composition to Viscosity of Mixtures of Sand and Water.

Kilos Sand and Water	Kilos Sand.	Kilos and Litres Water	Litres Sand.	Litres Sand and Water.	Per Cent. Sand, Volume	Per Cent. Sand Weight	Viscosity Mixture
9·20	0·00	9·20	0·000	9·20	0·00	0·00	1·01
9·30	0·45	8·85	0·165	9·02	1·83	4·84	1·02
9·35	1·10	8·25	0·405	8·66	4·68	11·8	1·06
9·35	1·40	7·95	0·515	8·47	6·08	15·0	1·09
9·40	1·90	7·50	0·699	8·20	8·53	20·2	1·12
9·40	1·95	7·45	0·717	8·17	8·78	20·8	1·13
9·55	2·20	7·35	0·809	8·16	9·92	22·0	1·13
9·20	2·25	6·95	0·827	7·78	10·6	24·4	1·18
9·05	2·50	6·55	0·920	7·47	12·3	27·6	1·23

It was found by these tests that when the amount of the sand in the mixture exceeded 30% by weight, the spigot would produce a very thick discharge for a short time, but that continuance operation under these conditions was not certain, and clogging would result.

A concrete example illustrating these data is given. It is desired to discharge from the pocket of a classifier 10 tons of sand per 24 hours, together with water in the ratio of 1 sand to 3 water, by weight. The head of water above the spigot is 3 ft. The mean sp. gr. of the sand is 2·81. What must be the diameter of the spigot opening? The area of the opening can be calculated from the formula given.

Taking up the terms on the right hand of the equation in order, f, the viscosity, may be estimated as follows: The weight ratio of water to sand in the mixture to be discharged is 3 to 1. Considering 100 g. of the mixture, the weight of water is 75 g.; its volume is 75 cc. The volume of the sand is 25 g. ÷ 2·81 (the density of the sand) = 8·9 cc. The total volume of 100 g. of the mixture is 75 + 8·9 = 83·9 cc. Hence, the percentage of sand by volume in the mixture is 8·9 ÷ 83·9 = 10%. From the table the viscosity of a mixture containing 10% of sand by volume is 1·17. Therefore $f = 1·17$.

The quantity of sand discharged per 24 hr. is 10 tons. One ton per 24 hr. is 0·651 kg. per minute. Forty tons per 24 hr. is 40 × 0·651 = 25·2 kg. per minute. The volume of sand per minute is 25·2 ÷ 2·81 = the density) = 8·98 litres. The quantity of water per minute is three times that of the sand, 25·2 × 3 = 75·6 kg. = 75·6 litres. The total volume of sand and water per minute is 8·98 = sand plus 76·5 = water) = 85·5 litres. The total volume per second = 85·5 ÷ 60 = 1·43 litres = 1430 cc.

Since the spigot is to consist of a short tube with a cone mouth on the influx end, the coefficient of discharge, c, may be assumed as 0·88.

In metric units g = 980 cm per sec.²; and the head, h (3 ft.) = 914 cm.

Substituting these values in the above equation, gives, for the area of the spigot opening,

$$a = \frac{1·17 \quad 1430}{0·88 \times \sqrt{2 \times 980 \cdot 914}} = 1·42 \text{ sq. cm.}$$

The diameter may be obtained from the relation,

$$d = 2\sqrt{\frac{a}{\pi}} = 1·35 \text{ cm.} = 0·53 \text{ in.}$$

R. H. Richards and B. Dibdin, *American Institute of Mining Engineers: Metallurgical and Chemical Engineering*, Feb., 1915, p. 120–66; H., S.

Effect of Increasing Reduction on Blake Type Crushers.—"At few mills has it been necessary to work crushers up to their maximum capacity. The fact that in most cases the crushers are fed by an amount of material limited by the capacity of the following machinery, is perhaps the most significant reason for this. In other instances, a liberal allowance has been made in the selection of the crusher.

Whether the crusher be worked to capacity or not, a saving in power cost can be effected by the installation of the smallest possible machine. The crusher at its best is a cumbersome piece of machinery with heavy moving parts. It has been found that approximately 70% of the power used on the crusher is used when the machine is running empty. If it is considered that perhaps 30% of the run-of-mine material requires no reduction, on the large first crusher, the possible capacity, indicated by the mouth size of the crusher, is not used, and for a great part of the time the crusher turns over doing no work, and there is a consequent loss of power.

The disregard of these facts has led to the installation of larger crushers than required. To economize this lost power some plants have tried to increase the amount of reduction. As the machine was originally underloaded, no doubt it would handle the load put on it, but this condition was long since expressed by W. P. Blake.

"When importance of getting a fines product leads in many cases to setting the jaws closely together, which, of course, increases the friction and wear upon the lower parts of the faces of the jaws, and it may be said that although by setting the jaws very close together . . . a larger amount of fine stuff can be produced by an ordinary rock

be so—it is not an economical use of the machine, since it was never designed to be so used."

What then are the factors controlling this degree of reduction and along what line should crushers be designed to meet these demands in the most satisfactory way?

Admitting that a crusher working all the time at breaking rock is working more economically than if it were running idle for part of the time, it is essential to have a crusher of the smallest size which will have the required capacity.

Heretofore it seems that mill men have closely connected "mouth size" with the capacity of the crusher, and manufacturers have followed lines that could satisfy this idea. Only vague guesses are given as to the effect of a variation in the size of product upon the output, and in all instances, these guesses favour the machine. Of several standard types, the actual gap fails to meet the stated mouth size, the measurement having been taken between favourable points, and a "trade" standard given. This is more apparent in the larger sizes, where the increase of mouth size greatly increases the amount of iron put into the machine. One 15×24 in. machine was found to measure 11¾ in., and a 15×24 in. measured only 13½ in.

Given a certain size crusher operating at a given speed and power, the capacity of the machine will depend only upon the time necessary for the rock to pass through the jaws. This time depends upon three factors: the time consumed before the rock receives the first break; the breaking character of the rock, etc., the number of pieces formed per break, and the size of the discharge opening.

The relation of the amount of fine stuff going through compared to the amount that must have actual work done upon it, would not influence the result appreciably, for at the high rate of vibration of the machine, 300 r.p.m., the fine stuff cannot cling to the larger pieces, but has free passage.

Obviously in a machine making 300 r.p.m. and travelling but a short distance, the effect of the impact of the jaw is not great enough to break the piece. If the piece is not gripped and squeezed, perhaps the maximum result is breaking of the edges so that this grip can take place. The time taken to have a piece is nipped is of greater importance than is generally recognized. A few seconds at most on one piece will rapidly amount to something as thousands of pieces go through.

Richards, in his "Ore Dressing," Vol. I., p. 91, shows mathematically that with a co-efficient of friction of 0·3, rolls will take a particle when the angle of nip is approximately 16°. The theory holds otherwise in a Blake type jaw crusher when the resultant breaking force passes below the centre of gravity of the piece. It should be remembered that whereas in the rolls the force keeping the piece in the zone of action is the difference of frictional force on the roll and the component of the normal force tending to throw the particle out, the force keeping the piece in the crusher in the zone of action is likewise the difference of the frictional force, holding the piece against its movement up the jaw and the component tending to throw it up.

When it is observed that the material fed to crushers is for the most part wet, as it comes from the mine, or dampened to reduce the dust, it is apparent the water exerts a lubricating effect, which is further augmented should any clayey matter also be present. This condition might at any time bring the co-efficient of friction down to 0·2. Again using Richards' formulas, the angle of nip would have to be 11° or under before a bite would occur.

The great variety of shapes and sizes fed to a crusher, as compared with the rather uniform product to the rolls, would indicate that whereas a roll operating with an angle of nip of 16° is just on the danger point, a crusher so operated would have exceeded it. From this reasoning it would appear correct that the angles between the jaws of a crusher should not exceed 12° to work near its utmost capacity.

By referring to the accompanying table, it is readily seen what degree of reduction under present standard measurements of construction will bring the jaw angle about this limit:

Degree of Reduction and Jaw Angle.

Size of Crusher, Inches	Actual Width opening, Inches	Length Vertical Jaw, Inches	Set to Crush to, Inches.	Angle Between Jaws.
4 × 7	4	12	½	15° 50′
			1	13° 45′
			1½	11° 50′
			2	9° 25′
7 × 10	6¾	17½	1	16° 30′
			1½	15° 0′
			2	13° 15′
			3	10° 50′
9 × 15	8¾	24	1½	15° 25′
			2½	13° 10′
			5	12° 0′
			4	9° 30′
10 × 20	8¾	26	1½	14° 40′
			3	11° 30′
			4	9° 40′
13 × 24	11¾	33	1½	16° 30′
			3	14° 15′
			4	12° 30′
			5	11° 0′
15 × 24	13¾	33	1½	22° 30′
			2	21° 45′
			3	20° 30′
			4	18° 30′
			5	17° 15′
			6	15° 20′
			7	13° 30′

The manufacturers, no doubt, have exceeded this angle because it gave them the mouth size that was sought for the least cost. The direction that has been taken to increase crusher capacity has been to make a wider jaw. It would have been better if the jaw angle had been made smaller, and the additional iron put into the height of the jaw, rather than the width. The second point, the breaking character of the rock, is important, but is a character outside of our control.

It is readily admitted that a decrease in the size of the discharge opening will reduce the capacity. This amount of reduction is, however, greatly underestimated. Extending the principle given by Richards in Vol. I., p. 35, of his "Ore Dressing," we may argue that in a 15×24 in. breaker, if one 15 in. cube reports at the mouth in 125 3in. cubes, then the capacity at the mouth is 125 times that at the throat when breaking to 3in. If, now, the crushing be reduced to 1½ in., there would be 1,000 cubes produced, and the capacity would be 1,000 times greater at the mouth than at the throat.

The capacity, then, in the second case would be theoretically but one-eighth of that in the first case."—M. W. Heller.—*Engineering and Mining Journal*, Feb. 27, 1915, p. 399. (G. H. S.)

Effect of Air in Zinc-Dust Precipitation Systems.—"In precipitating from cyanide solutions by means of zinc dust, the admission of air to the precipitation line and presses must be carefully avoided. Upon starting one Arizona mill there was absolutely no precipitation when the solution was pumped through the presses. The pressure rose to over 25 lb., with the presses handling but 50% of their capacity. A careful examination at the precipitate pump showed that the zinc-feed pipe entered the solution line between the pump and the pregnant sump, permitting a large volume of air to enter the pipe line and press. Upon correcting the piping arrangement there was no further trouble.

Another Arizona plant found that the effluent solution from the precipitation presses instead of being barren was actually 200 to 300 richer than the solution in the pregnant sump. A careful examination showed that at times the pregnant solution sump was being pumped dry, allowing large volumes of air to enter the precipitation system, and re-solution was taking place."—D. W. Minier.—*Engineering and Mining Journal*, March 20, 1915, p. 534. (G. H. S.)

Metallurgy of the Cam and Motor Ore.—"Ore from the Cam and Motor mine, Rhodesia, contains antimony, as has been mentioned in this journal before, and the recovery in the new mill is now only 73%. A great deal of experimenting has been done, and the following will be the flow-sheet: The roasted ore will be first pulped with water in order to wash out the deleterious compounds before mixing with cyanide solution. Coarse sand will be separated for tube-milling by Deister classifiers. The ground product will then be classified into sand and slime by Dorr classifiers, and, as the gold content in the sand is quickly dissolved, it is proposed to treat this product in small tanks with cyanide solution. Finally the sand will undergo a short leaching treatment in vats. The slime treatment will consist of continuous decantation on the Dorr system."—*Mining Press*, Feb. 13, 1915, p. 265. (G. H. S.)

Tungsten Wire Filaments.—"O. Ruff, in *Zeit. Ver. deutsch Ing.*, 1913, L., p. 1615 (through *Journ. Soc. Chem. Ind.*), says that to prepare a dense, ductile wire suitable for electric lamps, pure tungsten is first obtained by reducing the trioxide in hydrogen, and it is then compressed into rods which are hardened by heating gradually in a current of hydrogen in a tube furnace to 1000° C. for one hour. The rods are sintered by heating electrically up to 2850° C., the heating being carried out very carefully and slowly, as the quality of the final product largely depends on this process. The rods are then hammered and rolled at 1200° C. to 1300° C. in a current of hydrogen in a special machine, the operation being repeated until the rods are reduced to the desired diameter. To point the rods before inserting them in the wire-drawing machine the ends of the thicker rods are dipped into molten potassium nitrite, and the thinner ones are made the anode in an electrolytic cell containing potassium cyanide. The finished tungsten wire is silvery white, malleable, tough, ductile and non-magnetic. It is stable in air at ordinary temperatures, but oxidizes at a red heat so that electric lamps with tungsten filaments have to be exhausted."—*Engineering and Mining Journal*, Feb. 6, 1915, p. 295. (G. H. S.)

MINING.

Humidifying Mine Air.—"There are a number of methods of dealing with the dry, dusty conditions that appear in mines during the winter. Humidifying of the intake air is conceded by most to be the best way to deal with this condition. Personally, I feel that it is the only method which insures even comparative safety. Proper humidification of the air not only moistens the dust on the ribs and roof, but also moistens the dust particles floating in the air, causing them to become heavy and fall to the bottom.

Observations taken over a period of several years show that in the eastern section of Kentucky there is a deficiency of moisture in the intake air for practically seven months of the year. With normal weather, artificial means of supplying this deficiency are needed from 1st November to 1st May. During the dry periods preceding November and following May, the mine will normally take care of itself, unless surrounded by exceptionally dry strata. From the observations taken, it was found that the amount of water necessary to properly humidify the air varied from 2 to 4 gallons per 100,000 cubic feet of air, 2 gallons during early fall and late spring, and 4 gallons during midwinter. This amount varies greatly on individual days, but it was found that the mine could be used to some extent as a storehouse, and moisture stored up on warm days to supply the exceptionally dry or cold days following, provided the extreme cold or dry days did not continue for more than a week at a time. Do not make the mistake, which is common, of looking on warm days and moist days as being the same. It is true that cold days are dry days except on those rare occasions when the cold is accompanied by a dense fog, but warm weather closely following cold weather is ordinarily just as dry as the cold weather. For this reason humidifying is necessary, during March, April, and sometimes a portion of May.

Steam was adopted as the easiest method of placing the moisture in the air, and boiler plants were installed at each mine for this purpose. As no work other than that of boiling water was to be put on the boilers, old boilers condemned for high pressure were used, they being much cheaper and just as satisfactory as new boilers. One hundred horsepower return tubular boilers were used. These are about 5 ft. in diameter and 16 to 18 ft. long. They were set in a standard boiler setting with a light wood building as a protection from the weather. Only a short stack is used, as it is not intended to fire them heavily and not much dratt is needed.

The steam is carried direct into the fanway by a 2 in. pipe line, and a branch is carried into each airway. These branch lines are only carried beyond the crop-affected roof near the outcrop.

I should state here that this company uses only blowing fans, having found them more satisfactory for all kinds and classes of mines operated by it. These fans have a reversible fan setting for use in case of emergency. The ventilation is laid out with the intention of using steam, and the air travels from 1,500 ft. to 2,000 ft. before reach-

ing entries where men are required regularly to work. The steam is turned into the air near the bottom and in the centre of the airway. It quickly passes to the roof and spreads out till it fills the entire airway with a dense white fog. A small amount condenses and settles to the bottom as fine mist or rain, but this loss is only a small percentage. By having a large section area in the airway the air travels at a low velocity. After travelling about 2,000 ft. it is heated by the surrounding strata to near the mine temperature. As the temperature rises the moisture or fog previously is absorbed by the air which remains constantly at 100 relative humidity, until the fog is absorbed.

This absorption of moisture is due to the increase in the temperature. This function of air should perhaps be mentioned here, namely, that air has a greater capacity for moisture as the temperature increases. Thus, 100,000 cubic ft of air at 50° F. will be fully saturated by (or could not contain) but 1 gallon of water, while at 60° F., the average mine temperature, it will absorb (or could contain) 10 gallons of water. The fact that makes the use of steam practical is that a large quantity of water in the form of fog may be carried in the air at any temperature, much the same as mud is carried by a river. Thus the air at 50° and a capacity to absorb 1 gallon of water permits the other 9 gallons to be carried along as fog and gradually absorbs the moisture necessary as it is warmed by its passage through the mine. The best results are obtained when the amount of steam is so regulated that the fog disappears at the point where the air reaches the normal mine temperature. This point agrees with the temperature of the intake air and with the velocity of the air in the airways. I have found that under conditions ordinarily met with in coal mines, to be from 1,500 ft. to 2,000 ft. from the intake. This distance makes this method of using steam impractical in the new mine during its first stages of development, due to fogging up the entries and working places of the miners. If steam is to be used in such a mine, a mine air heater should be used with the steam. This shortens the distance the fog travels into the mine. In the mines of which I have been speaking steam was not used in the development stage. Here conditions were taken care of by watering from a sprinkler, which, though not entirely satisfactory, was undoubtedly of considerable value. Also, as the workings were not extensive at this stage, the amount of sprinkling and labour necessary to keep the mine in a moist condition was not difficult to provide.

After installing the steam plants at the mine, it becomes necessary to fire them in such manner as to put the proper amount of water into the air. They started in the fall with 2 gallons per minute, or 2,880 gallons per day for a mine using 100,000 cub. ft. of air. Making allowance for loss of moisture, this becomes about 2½ barrels of water per hour. In the boiler used one barrel is approximately 1 in on the water gauge. It is thus only necessary for the fireman to so regulate his fire that the water is lowered 2½ in on the gauge each hour. This should theoretically put the fog line at its proper place in the mine, but, as in all things relating to a mine, it needs some individual attention. This is given it by the mine foreman. The foreman keeps trace of the fog line inside, and should it penetrate far enough to make working conditions dangerous on the entries, he notifies the boiler tender to this effect and has the quantity of water boiled per hour reduced until conditions are satisfactory in the mine.

I must state that I have found that the fogging up of the entries was more generally due to poor brattice work and leaks along the main airways than to the use of too much steam. In a few instances it was due to the use of too large a quantity of air, which caused the temperature of the air to remain considerably below that of the normal mine.

In this work advantage is taken of the fact that it is easier to boil water at a low pressure than a high pressure, and the valve leading to the mine is left wide open, thus requiring only enough pressure to force the steam into the mine. This varies from 1 lb. to 5 lb. The water supply pipe of the boiler was connected to that used for fire protection, and from which sufficient pressure is obtainable to force the water into the boiler by simply opening the valve, thus doing away with the use of an injector, and making the firing of the boiler so simple that any man around the mine can do it easily.

In the mines in which this system was in use last winter, only a small amount of drying out occurred. It was found in practice necessary to reduce the distance the fog travelled in the air to a point where the air had not yet reached its full mine temperature. Thus, not the full amount of moisture was supplied, yet it was found that the natural production of water by the mine was sufficient to provide most of this extra moisture and that the mine remained damp, drops of moisture standing out on the roof and ribs and the condition known as sweating was maintained both summer and winter over most of the mine. The portions of the mine that were affected by slight drying were remedied by occasionally at night increasing the amount of steam until the entire mine was filled with fog, sufficient of which was precipitated to moisten these dry portions.

At some of the mines only a small temporary boiler was installed last winter. At these mines it was found that though not enough steam was supplied to keep the entire mine moist, yet even a small amount of steam greatly reduced the extent of the dry sections and made watering by sprinkler or water car considerably more effective and lasting.

A great deal in this work depends on the amount of judgment shown by the foreman, as he must keep closely in touch with the beginning of dry conditions, in fact, must anticipate them, and must also prevent the causing of unpleasant working conditions and fogging of the haulways by too much steam. In a very large mine, this can be taken care of mostly by theory, but in the ordinary mine the theoretical amount must frequently be tempered by the use of good judgment. At all times it is essential that the foreman give attention to keeping the brattices in air-tight condition and so regulate his ventilation as not to have too high a velocity of air in the main airways.

With an honest effort by the foreman to accept the theory and apply it with ordinary judgment, working as close to it as the conditions of the mine will allow, very good results were always obtained." sl. W. REED.—*Queensland Government Mining Journal*, March 15, 1915, p. 119. (H. A. W.)

MISCELLANEOUS.

PROTECTION OF PIPE LINES AGAINST ALKALI CORROSION.—"The pipe lines serving the California oil fields have frequently to be carried through alkaline soil. The effect of this alkali is to corrode the pipe and in some cases render it useless in a few years. Various attempts have been made to cover the pipe and prolong its life. Asphaltum paint has not been successful. Crude oil runs away through the soil. Quick setting bituminous enamels put on in a fairly thick layer have had some success, but cannot be applied to a cold pipe or a wet one. The most extensive and successful method devised employs specially prepared roofing paper. Such paper is applied to the pipe in either a spiral wrap or a longitudinal lap. In the spiral wrap the paper is cut at the mill into the desired width, 3 in. to 12in., and furnished in lengths of from 50 to 100 ft. As soon as the pipe in screwed together, and before lowering into the ditch, it is coated with hot asphaltum and the wider strip is wound on spirally before the asphaltum sets. There is a crack left by this spiral and this is again coated with hot asphaltum and a 3 in. strip wound over it as a batten. The asphaltum is a thin grade of the ordinary refined product, applied at about 200° F., of such consistence that it will not set for about five minutes after application. At joints, 3 in. strips of paper stuck together with asphaltum are wound around the pipe on each side of the collar until a shoulder is built up flush with the outer circumference of the collar; the sleeve and shoulder are then coated with hot asphaltum and covered with a 12 in. strip of paper. This last strip is bound down with wire, as are also the ends of the spiral wrap.

The longitudinal lap system has been more extensively used. For this the paper is delivered in rolls about 6 in. wider than the pipe circumference and about 72 ft. long. The paper is applied by coating the pipe with hot asphaltum, wrapping the paper around it and sticking it down with the same material. When the oil is to be pumped hot the paper is not always stuck down, since it is calculated that the effect of the heat from the oil and the pressure of the earth will result in its sticking. It is a question, however, whether wrinkles and cracks will not develop in such cases. The lap should always be made on top of the pipe, with the outer portion looking down, and wires should be placed about every 16 in. in order that the weight of the paper itself will not pull it away from the pipe. The joints are protected by building up shoulders as with the spiral wrap." —*Engineering and Mining Journal*, Feb. 20, 1915 p. 367. (G. H. S.)

ACETYLENE WELDING OF PIPE JOINTS.—"A process of welding joints in the construction of steel and wrought iron mains has been developed to take the place of threaded joints and couplings. Experience in this country, notably at Philadelphia, Chicago and San Francisco, and also in foreign cities, has shown that welding is not only an economical method of making joints, but that a welded main eliminates joint leaks and the maintenance cost due to insecure joints, says the above writer, who is western editor of *The Iron Trade Review*.

A skilled operator produces unions with 80% to 95% of the strength of the pipe and by building up the sections at the weld the strength is increased even beyond that of the pipe. Joints of this strength permit great flexibility in construction and are said to insure absolute tightness.

The method of producing welded joints is simple. The pipes are strung along the surface and the successive lengths are simply butted together and welded by the operator, who is assisted by a helper at each end of the section. The helpers turn the pipe with chain tongs so that the operator is always welding on top, in which position the fastest work can be accomplished. The welding flame, at a temperature of approximately 6,300°, heats the metal on each side of the joint to the fusion point and pure Norway iron wire is fused with the pipe metal to form a fusion weld. The operator thus can build up the weld to any desired thickness.

With pipes cut squarely the sections are butted within 1/16 in. to 1/4 in. of each other, according to the size of the pipe. However, pipe may be obtained, without increase in cost, cut at a bevel, in which case the two pipes are butted together, form a V shaped opening of about 60°. With the ends cut in this manner, the weld can be made more quickly, at less cost and with less filling material.

The great strength of the welded joint has made practical the use of pipe in 40 ft. lengths instead of 20 ft., commonly used with other methods. Much lighter pipe is also feasible as it is not necessary to make allowance in it. thickness to permit threading. This results in a saving in cost, and it is reported that in some parts of Europe, where welded joints have been in widespread use much longer than in this country, it has been demonstrated that pipe of 40% less thickness may be utilized with the same results.

As soon as a section of the welded pipe is finished it is tested for leaks at any desired pressure. After it has been tested it is rolled to the trench and lowered to place. Various methods are followed in doing this work. It is one engineer's practice to drop the section bodily instead of lowering it with ropes or chains.

Although the trench should be graded as carefully as usually is the custom in ordinary practice, no care need be taken to have the pipe absolutely straight. In fact, a few curves are desirable, as contraction and expansion are taken care of in this way. Experience has demonstrated this is the only provision necessary to take care of expansion and contraction.

The length to which the pipe may be welded before lowering into the trench apparently is not limited. In the open country it is not unusual to weld as much as 1,000 ft. at a time, and in one case 100 lengths of 8 in. pipe, each 40 ft. long, were welded before being placed.

A convenience resulting from this method is illustrated, indicating how laterals can be inserted into the main at any angle by the use of the cutting blowpipe in combination with the welding blowpipe. The workmen simply cut out say a 10 in. pipe by making holes 12 in. in diameter in the sides of the drip cylinder. The pieces of 12 in. pipe then can be welded in place.

The Oxweld Acetylene Co., Chicago, which has done much pioneer work in developing this method of welding, has pertected a portable welding unit. It is readily wheeled about from place to place and is compact and simple."—GEORGE H. MANLOVE. —*Mining and Engineering World*, March 20, 1915, p. 553. (J. C.)

Review.

THE CYANIDE HANDBOOK. By J. E. Clennell. Published by the McGraw Hill Book Co., New York, 1915. For sale by the *Mining Press*. Price, 8s. 6d. pp. 417, index.

This is the second edition of Mr. Clennell's well known treatise on the chemistry and practical application of the cyanide process. The volume under review contains 81 pages more than the previous one, and every effort has been made to bring the matter up to date. It was not considered advisable to omit any great part of the descriptions of the older methods, these being generally still in use; and new arrangements often need the adoption of systems previously discarded. Section III. of Chapter I. covers in short the tendencies of modern practice. The chemistry of the progress is given practically the same as before. In part III. of the preparatory treatment of ore for cyaniding, the comparative utility of various crushing and grinding machines is new, including Lane slow-speed Chilian and Hardinge mills. Also, the following discussions are in addition to No. 1 edition: factors influencing rate of settlement of slime, diaphragm cone classifier, Merrill cone, Dorr classifier, Dorr thickeners, amalgamation in conjunction with cyaniding, skimming tanking, dumps, continuous agitation and decantation; the Pachuca, Patterson, Trent, and Dorr agitators; the Merrill press; the modified Butters filter and Oliver filter; zinc dust and aluminium dust precipitation; smelting precipitate; treatment of antimonial and arsenical ores; new applications of aluminium in ore treatment; electrolytic regeneration of cyanide; sources of error in assaying ore; assay of silver ore from Cobalt; additional notes on ore analysis—copper, tin, zinc; method for valuation of caustic lime; and tests to determine value of zinc dust for precipitation. Valuable additions to the existing literature. Cyanide workers will always find something useful in this excellent study of a world-wide used process. *Mining Press*, March 6, 1915, p. 392. (G. H. S.)

Abstracts of Patent Applications.

(C.) 129/14. William Charles Stephens. Improvements in or connected with rock drilling machines. 31/11/14.

This application refers to modifications in the head ends of drill steels for the hammer type of rock drills and to arrangements for admitting water to hollow steels.

(C.) 127/15. Frederick Charles William Ingle. Improvements in means for sampling crushed, granular or loose material, or a mixture of such material and a liquid, or dividing or sub-dividing samples of such material or mixture. 2/6/15.

This application relates to improvements in means for dividing or sub-dividing samples of crushed ore for assay purposes and consists in a device containing a feed hopper, provided with spouts or outlets, which is open and is intended to deliver a representative portion into each of the receptacles arranged beneath.

Alternatively the receptacles may be arranged to revolve instead of the feed hopper and spouts, and an alternative means for rotating the hopper by means of compressed air and fan blades is also shown.

(C.) 128/15. Francis Dey. Improvements in the construction of concrete and other similar walls, roofs, floors, and the like, and apparatus for the use thereof. 11/5/15.

This application refers to improvements in the construction of concrete or other similar walls, floors, ceilings and the like in situ, and has reference to the methods of forming the boxing or moulds required in such work. There are twelve claims.

(C.) 135/15. Philip Blair Fjost (1), Frederick Rings (2). Improvements in the construction of floors, roofs, decks of bridges, foundations of roads, tracks, footpaths, yards, areas, quays and the like. 16/6/15.

This application has reference to beams made of concrete or other like material, the beams consisting of a compressive top member, appreciably narrower than the bottom or tensile member, and also side lugs at intervals along the whole length of the beams so that, when the beams are laid side by side to form a floor, the filling can be introduced between the beams so as to make a continuous slab. This filling may also be of such a nature that it can be used for fastening down flooring boards.

(C.) 141/15. William John Isaac Morcom. Improvements relating to tipping skips. 22/6/15.

This application refers to a proposed arrangement of tipping gear for skips in mine headgears.

Instead of the usual forms of tipping rails, the skip is made to run into a truck provided in the headgear, this truck being arranged to travel on tip rails and thus discharge the skip.

The advantage claimed is that the skip body can be lowered for the wheels, be made of greater width and capacity than with the wide back wheels usually employed for tipping the skip.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

BARRY, R. A., *l/o* Johannesburg; Transvaal G. M. Estates, Ltd., Pilgrims Rest.

BASKETT, E. G., *l/o* Gold Coast; Royal Colonial Institute, Northumberland Avenue, London, W.C.

BELL, H. F., *l/o* Falcon Mines, Umvuma, Rhodesia.

GRAHAM, L., *l/o* Day Dawn; c/o Mundi Mundi, Brookman Street, Kalgoorlie, W. Australia.

HARDY, J. G., *l/o* Mexico; c/o American Smelting and Refining Co., 120, Broadway, New York, U.S.A.

LLOYD, A. M., *l/o* Madagascar; "E" Squadron, East African Mounted Rifles, c/o C. Tidall, Esq., Electric Lighting Co., Nairobi, B.E. Africa.

MAIN, A. E., *l/o* Mexico; c/o The Exploration Co., Ltd., 61, Broadway, New York, U.S.A.

MANCE, J. C., *l/o* India; Capt., Train Conducting Officer, c/o D.A.D.L.T., Havre, France.

WILLIAMS, G. W., *l/o* 33rd Divisional Signal Co., Royal Engineers, British Expeditionary Force.

YATES, A., *l/o* Prestwich; 16a, Harrington Gardens, South Kensington, London, S.W.

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Vol. XVI SEPTEMBER 1915. No. 2

Proceedings
AT
Ordinary General Meeting, September 18, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, September 18th, 1915, Mr. J. E. Thomas (President) in the chair. There were also present:—

35 Members: Messrs. H. A. White, Prof. J. S. Cellier, G. Hildick Smith, C. Toombs, M. Torrente, F. W. Watson, John Watson, A. Whitby, W. R. Dowling, Prof. G. H. Stanley, J. R. Williams (Members of Council), H. R. Adam, R. W. Bennett, H. C. Boydell, L. Colquhoun, W. M. Coulter, J. M. Dixon, H. W. Gill, B. J. Hastings, W. E. John, E. J. Laschinger, G. A. Lawson, H. M. Leslie, J. D. Marquard, G. Melvill, P. T. Morrisby, S. Nettleton, S. Newton, E. A. Osterloh, W. S. V. Price, W. A. C. Tayler, A. Thomas, J. T. Triggs, J. H. Veasey, and M. Weinbren.

11 Associates and Students: Messrs. D. L. Allan, J. Davidson, O. A. Gerber, J. Ingram, A. King, D. Ordman, F. J. Pooler, D. W. Pugh, H. Rusden, H. Ward, and W. Waters.

10 Visitors and Fred Rowland (Secretary).

MINUTES.

The Minutes of the Annual General Meeting were confirmed.

OBITUARY.

DEATH OF MR. R. G. BEVINGTON.

The President: Before proceeding with the business of the evening I deeply regret to have to announce the death of one of our Past-Presidents, Mr. R. G. Bevington. Mr. Bevington met with an accident while attending to his duties as Mill Manager of the Village Main Reef G.M. Co., and died from the results of the injuries he sustained. As you no doubt will have seen in the obituary notice published in the *Journal* for July, Mr. Bevington was one of the Foundation Members of the Society, and had been a Member of Council since 1908.

I especially, as President, shall miss his matured and valuable advice during my term of office.

I would ask all present to rise from their seats as a mark of respect and a token of their sympathy with his widow and family.

All present rose to their feet.

NEW MEMBERS.

Messrs. A. Whitby and J. R. Williams were elected scrutineers, and after their scrutiny of the ballot papers, it was declared that the following candidates for membership had been elected:—

Brink, Cyril Gordon, Fairview Mine (T.C.L.), P.O. Sheba, Transvaal. Reduction Worker.

Brodrigan, Charles B., East Rand Proprietary Mines, Ltd., P.O. Box 57, East Rand. *(Transfer from Associate Roll.)*

Greathead, Cyril Harding, Summer & Jack Proprietary Mines, Ltd., P.O. Box 192, Germiston. Mining Engineer. *(Transfer from Associate Roll.)*

Halbert, Thomas, B.S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Analytical Chemist.

Hutton, Charles Edward, Mines Department, P.O. Box 161, Germiston. Inspector of Mines.

King, Andrew, Knights Deep, P.O. Box 143, Germiston. Cyanide Manager. *(Transfer from Associate Roll.)*

The Secretary announced that since the last meeting of the Society,

Mann, Robert Lloyd, Joker Mine, P.O. Shamva, S. Rhodesia. Reduction Officer.

had been admitted as an Associate by the Council.

General Business.

RESIGNATION OF VICE-PRESIDENT.

The President: Owing to Mr. R. A. Barry having left the Rand and gone to Pilgrims Rest as General Manager of the Transvaal Gold Mining Estates, he has, I regret to say, found it necessary to resign his position as Vice-President of the Society, and the Council has elected Mr. G. Hildick Smith to fill his place as a member of the Council.

Mr G. Hildick Smith *(Member of Council):* I greatly appreciate the honour the Council has done me in having elected me one of their members. I will do everything I can for the Society.

MESSRS. RALPH STOKES AND GERARD WILLIAMS.

The Secretary: I might mention the Society has had acknowledgments of re-election as Corresponding Members of Council from Mr. Ralph Stokes, who is now a Lieutenant with the Royal Engineers in France, and from Capt. Gerard Williams, who is hoping shortly to be in France. I might read an extract from Mr. Stokes's letter from the trenches, as being of interest:

"Although I have been very decidedly at the front for about nine months, you may be surprised to hear that my work has for five months been almost exclusively of a character to justify detailed description in a paper before the Society without going beyond the recognised scope of its transactions. But, Censor or no Censor, I shall not feel tempted to submit the paper before the end of the war!" Mr. Stokes concludes by sending his kindest remembrances to his many old friends in the Society.

DR. HAHN.

Mr. J. R. Williams *(Past-President):* I regret very much that I have been so much tied that I have not been able to give that attention to the Society that I used to in the past. I would like to take the opportunity of congratulating the President on his election, and to congratulate the Society on its wise selection.

Mr. Williams then entered a vigorous protest against Dr. Hahn's name remaining on the list of Honorary Members.

In the discussion which ensued it was pointed out that Dr. Hahn was a born British subject, and ultimately it was agreed that the matter be dealt with by the Council at its next meeting.

A DESCRIPTION OF AN IMPROVED METHOD OF REMOVING THE BROKEN ORE FROM STOPES.

Mr. M. Weinbren *(Member):* One of the most inefficient, if not the most inefficient operation, connected with mining on dips of 25° to 33° is the removal of the broken ore from the stopes, for the following reasons:—

1. The work is carried out by unskilled natives, whose main object is to get through the shift with as little exertion as possible.

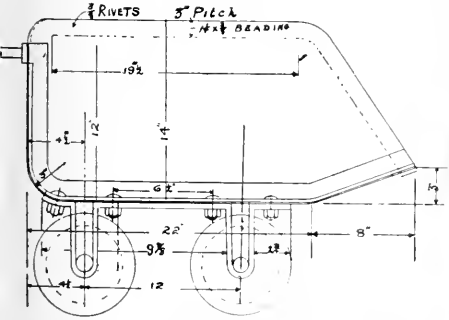
2. The work is difficult to regulate and supervise, when, for instance, say ten natives are shovelling near one another in some part of a stope, it is almost impossible to so place them that they do a maximum amount of work, and the result is, that instead of each native lifting the rock and throwing it as far as possible, he only scrapes it along the foot-wall into the shovel of the native immediately below him. It therefore follows that if excessive shovelling can be avoided, the removal of the broken ore will become more efficient. With this object in view the following system was introduced. A permanent line of chutes was laid down at a safe distance from the faces of the stopes, in many cases one line being sufficient for both faces. Each line of chutes was given a uniform grade, so that the rock should go from the very top of the stope right into the box without choking the chutes or jumping out of them. The chutes were well anchored by putting in props made of stout lagging poles at every fourth or fifth section, to which the sections were chained. It was found better practice to fasten the chains to the chutes by bolts rather than by hooks at the ends of the chains. The advantages of this method were that the sections of the shakers were kept well down on each other, and there was less likelihood of obstructing the rock in going down the chute. A water tap was placed at the top of each chute line.

Trucks of one foot gauge were then put in at approximately every fifty feet along the dip of the stopes, and running from the faces to the chutes. It would at first sight appear that it would be difficult to lay down a number of horizontal tracks in stopes dipping at 33°, but this by no means proved to be the case, and it was very seldom that the foot-wall had to be blasted to obtain a horizontal track.

Blasting was avoided by putting in several props along the length of the track, and then laying lagging on the upper side of the props. One end of the sleepers was made to rest on the lagging and the other end on the foot-wall, and to make the track solid loose rock was shovelled in beneath it.

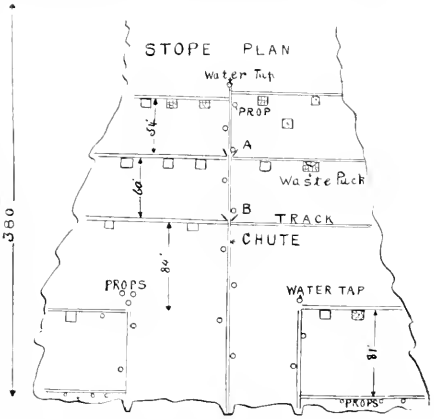
Where possible waste packs were used to support the tracks, which were laid with old rails and sleepers, and as the pipe-fitter's natives were quite equal to the task of laying them down, the whole system did not prove very expensive.

A special type of truck was designed. It is very light, and can therefore be easily placed in a position of safety, behind some waste pack, at the end of the shift. The truck has a capacity of five cubic feet, and as the total height is only one and a half feet from the rails it can be handled quite comfortably in most stopes. It is tipped by pulling the back of the car upwards, the front wheels acting as the fulcrum. Further details of the trucks are shown in the drawing. The total cost of a truck is two pounds ten shillings.



Sectional Elevation Stope Truck, ¼ in. Plate, Width 2 ft.

The accompanying stope sheet very clearly shows the system, and also the amount of labour saved, especially, in stopes with long backs. After the system was introduced in this stope the number of shovellers was reduced by 50%, and yet the stope is kept cleaner than it was before. The stope-sheet shows how waste packs have been utilised to support the tracks, and, further, the barricades A and B, which protect the natives, when tipping on their respective tracks, from stones passing down the chutes from the tracks above them. By means of the barricades the chutes can be divided into sections, so that if any section requires repairing the whole



Stoping Width, 50 in.; Dip, 33°; Scale, 1=600.

line need not be stopped. A half section of rails (9ft. long) is kept ready at each track, so that the rails can always be kept up to the face. As a rule one native shovels the rock on to the track, while another loads it and wheels it to the shaker, as shown in the photograph.

Summarising then the system consists of permanent chutes and tracks running from the face to the chute, roughly, every fifty feet on the dip, with specially designed trucks to overcome the difficulty of small

stoping widths, and the advantages claimed for it are:—

1. By having permanent chutes no time is wasted at the beginning and end of the shift in putting them up and taking down.

2. Gravity is made to assist as much as possible.

3. Instead of having boxes every 20 to 30 feet, they are only necessary every 60 to 70 feet.

Better supervision is possible.

4. A good idea can be obtained of how much each native actually does.

In conclusion, I desire to thank the management of the mine where this method is employed for permission to publish this note.

Mr. G. Hildick Smith *(Member of Council):* I would like to ask Mr. Weinbren if I am correct in having understood him to say there was only one mine on the Rand on which this method was in use.

Mr. M. Weinbren *(Member):* There are other mines which use intermediate tracks, but not to such an extent as the Robinson Deep, the mine refered to. I believe the "State Mines" uses intermediate tracks, but it is unfortunate in having a dip of about 18°, so that boxes have to be sunk at the intermediate tracks.

Mr. G. Hildick Smith *(Member of Council):* Mr. Weinbren is quite mistaken in his statements as to the extent to which intermediate tracks and footwall chutes are used on other mines. For instance, at the Ferreira Deep there are some thousands of feet of intermediate tracks and footwall chutes, and this system of handling ore has been employed throughout the mine for the past three years. The interesting point is, what is the lowest dip at which ore can be flushed down a chute without the necessity of placing shovelling-natives on the chute line? The point is governed to a great extent by the type of chute used; with a chute line constructed of chutes such as shown in Fig. 1, ore can be flushed down a well graded line at dips of from 20° to 21° and upwards. The type of chute shown in Fig. 1 is that used for main lines, into which ore from trucks is tipped direct. Fig. 2, shows a lighter type of chute used for subsidiary lines, and whenever possible to facilitate shovelling. This lighter form of chute can easily be laid near a working stope face at the commencement of each shift and removed again before blasting takes place. It is now the practice to have liners or wearing plates rivetted along the centre of the larger types of chutes. The liners consist of strips from the sides of old chutes. Apart from the fact that the removal of ore from stopes is facilitated by means of a system of intermediate tracks and footwall chutes, this system indirectly has a great influence on the prevention of

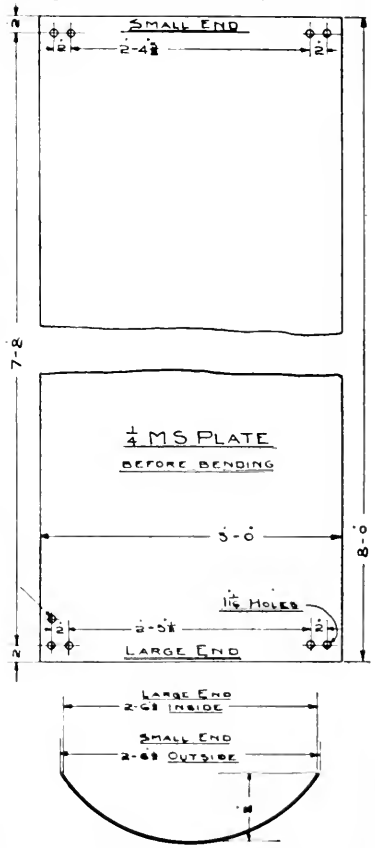


Fig. 1. Large Footwall Chutes.

dust underground. I have asked Mr. Weinbren to visit the Ferreira Deep and inspect our shovelling and tramming arrangements, as it is possible that both the Robinson Deep and the Ferreira Deep may profit by comparing notes on this subject.

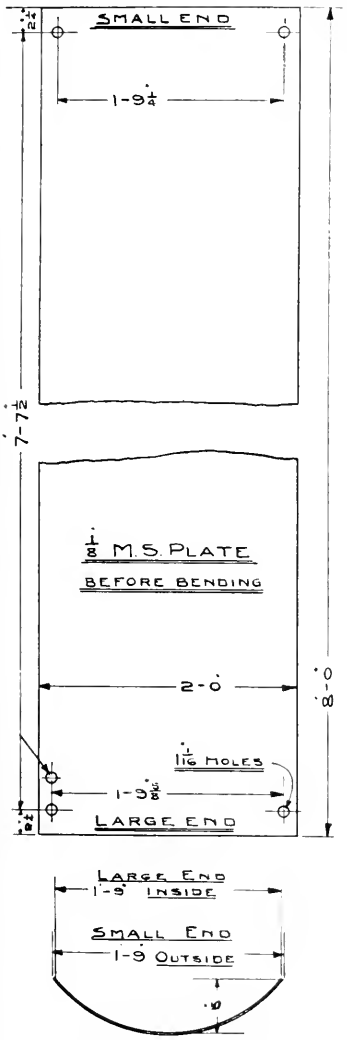


FIG 2. SMALL FOOTWALL CHUTES.

The President: Our thanks are due to Mr. Weinbren for bringing this matter forward. It is one of those little notes that bring out details of ordinary working which are sometimes, to a great extent, lost. Though it may not be novel to some of our members, to others it may lead to a development of quite an interesting idea.

USES FOR WORN TUBE-MILL LINER BARS.

The following was read by Mr. W. R. Dowling:—

Mr. J. M. Dixon *(Member):* In these days of scarcity of material and rising prices, the following notes on further methods of obtaining uses for worn out tube-mill liners of the Osborn type are submitted in the hope that other suggestion for economy on similar lines may be forthcoming.

Launder lining.—In coarse crushing plants endless expense is caused by the excessive wear of launders, and the writer has found relief in employing discarded tube-mill bars for this purpose. The bars are selected to a more or less uniform size and are set in cement, the thickest ends being placed at the lower end of the stream to avoid splash. If insufficient time is available to allow the cement to set, the liner should be completely covered with belting or other suitable material, to afford protection for some days.

Battery Bins.—The wearing faces of battery bins can be covered with worn bars set on end and held in position either by staples or by cross pieces bolted to the bin.

Battery Feeder Chutes, when lined by laying discarded bars cut to suitable lengths side by side, and wedging with wood (cement need not be used) give no further trouble.

The use of steel bars for lining bins and chutes is by no means novel, and the foregoing remarks rather indicate a new source of supply. It may be, however, claimed that the use of worn tube-mill liner bars as liners to the stamp-mill mortar box is new.

Mortar box lining.—Trials made of using this waste material to line the backs of mortar boxes have given very good results. A close selection of bars has to be made in order to get a more or less uniform and suitable thickness. The bars when cut to the desired lengths are bevelled, the bevel corresponding to that of the end liners, and when placed horizontally in the box are held in position by the end liners, small wedges of hardwood being driven in to complete the lock. Nine to twelve pieces form the back liner, according to type of box, and those receiving most of the wear are easily re-

placed by pinching up the top pieces and changing only those worn out. The top and bottom sections are available for three or four liners. The metal left in a cast steel liner is considerably greater than that actually worn away, but the casting nevertheless must be discarded because it does not form a complete protection to the box. These bar liners give quite a good life and when compared with imported manganese steel at 5d. to 6d. per lb. a considerable saving is effected, the cost of cutting on a mine being practically negligible.

Mr. E. J. Laschinger *(Member):* I did not come here to night with the intention of saying anything particular, but this little paper read by Mr. Dowling just now reminded me that there is a movement on foot with regard to the manufacture of materials, which would relieve the pressure upon manufacturers at Home so that munitions could be manufactured there instead of going to the absurd length of trying to manufacture munitions in this country. It would strike any reasonable man, if we could, in this country, manufacture materials which are in everyday use, more especially by the Mining Industry, it would relieve workers in the Old Country. The idea has been mooted and is under consideration by the Technical Committee of the South African Institution of Engineers that this Society should be approached with regard to getting expert advice as to what chemicals and such other things could be manufactured here, so as to relieve the present shortage experienced in a great many lines. I am not quite certain whether the Council of this Society has had a communication yet, or what steps have been taken; but I should certainly think that the Council of this Society could do a great deal by working in conjunction with the Technical Committee of the Institution of Engineers or with any other organised Society or body of men, who have weight or influence in this country. I have no doubt whatever this Society will do all that is possible in order to forward anything which will help towards the successful prosecution of the present war. I only would like to ask whether this Society has done anything yet, or whether the Council received any communication from the Munitions Committee?

The President: I might answer Mr. Laschinger by saying we have heard nothing from the Munitions Committee. Of course, the point with regard to manufacturing various articles in this country has already been brought up by Mr. Watson and others at several meetings.

With regard to the notes that Mr. Dowling brought forward just now, written by Mr. Dixon, I think this is one of the sort of papers I have referred to before as being useful to our members, particularly in these times when we are liable to be short of supplies that we require from overseas.

Mr. C. Toombs *(Member of Council):* Another use to which these old tube liners could be put is as fire-bars for smelting and assay furnaces.

Mr. H. A. White *(Vice-President)* took the chair while Mr. J. E. Thomas (President) read his Address, as follows:—

PRESIDENTIAL ADDRESS.

By Jas. E. Thomas, A.Inst.M.M., M.Am.I.M.E.

In the past it has been the custom for each newly elected President to commence his term of office by giving an address, and it has been decreed by your Council that I shall continue that custom. The choice of a subject becomes increasingly difficult each year, as no doubt our Past-Presidents will agree, and I regret that I have been unable to spare the time since my election to prepare a technical paper to bring before you, so must perforce confine myself to generalities.

I desire to thank you for the great honour you have conferred on me by electing me your President, and to assure you that I shall do my utmost to show my appreciation of that honour by doing my best to further the interests and usefulness of the Society and, at the same time, endeavour to prove myself worthy to follow in the footsteps of my predecessors, the Past-Presidents.

The Society is to be congratulated on the Vice-Presidents and Council elected for the present year, and I feel very gratified indeed at the prospect of having their valuable advice to guide me during my term of office.

It is at present useless, I am afraid, to expect the same activity and interest in the Society's doings that was displayed a few years ago, owing to the terrible war now raging in Europe, but it is clearly our duty to do what we can to promote the advancement of technical knowledge in the meantime, so that at the conclusion of hostilities we may be able to do our share towards the economic recuperation of the Empire.

This war has brought home to us, as probably nothing else could have done, our dependence on other countries for the supplies necessary for carrying on our great mining industry, on which so much depends in this great crisis. One can only hope that the lesson will not be lost and that strenuous efforts will be made towards making us, at least to some extent, self-supporting in the future.

It has been successfully demonstrated that it is no longer necessary to import lead acetate for cyaniding purposes as locally made lead nitrate is cheaper and more efficient.

I remember the consternation in cyaniding circles a few years ago caused by a shortage of lead acetate, due, it was alleged, to the sinking of a few barges, containing that salt, in some inland waterway in Germany, and also how it was sought to overcome the difficulty by having natives roll barrels containing litharge and acetic acid all over the veld to produce lead acetate.

There is also a great possibility that our requirements in zinc may be met by the recovery of zinc from waste solutions, augmented by zinc from ore mined in the country.

I sincerely trust that these efforts may meet with the success they deserve and that we may, in the near future, be favoured with a paper on the subject.

These two items alone show the importance of encouraging the base metals industry in this country, a subject that has not hitherto received the attention that is its due.

It is a common saying amongst the younger members of our profession on the Rand that these professions are overcrowded. Let me remind them, however, that the same cry was raised years ago when the Society first came into existence, and also of the remarks made in 1895 by one of our Past-Presidents, Mr. A. F. Crosse, "That no profession is overcrowded with really hard-working, energetic and capable men."

In my opinion the base metal industry affords as good, if not better, opportunities to our younger members as the gold mining industry did to our older members in the early days of the Rand.

In those early days most of us had to learn by our mistakes and by the opportunity of exchanging and discussing ideas afforded by the Society. Now we have a most excellent School of Mines and Technology at which a thorough grounding in chemistry, metallurgy and mining may be obtained at very little cost, but that, I maintain, in no way detracts from the usefulness of the Society. No better evidence of this could be obtained than the fact that some of the professors of the S.A. School of Mines and Technology are amongst our most useful and active members.

I do not propose to review the doings of the past year, as that has already been done in the Annual Report submitted by your Council at the Annual General Meeting held on the evening of the 26th June and published in the *Journal* for that month. I must be permitted, however, to remind our members that the Society needs their whole-hearted support to enable it to carry on its work in these times of stress.

Your Council has studied the question of finances most thoroughly, and has resolved to economise in every possible manner that will not unduly affect the efficiency of the Society.

The abstracts continue to be a valuable feature of your *Journal*, giving, as they do, a concise description of practice and improvements in other countries. Much time and labour is devoted by your Council to the preparation of these abstracts, and I sincerely trust that it will not be found necessary to seriously curtail this portion of our activities.

Although Government has not called on us as a Society to undertake any particular work to aid in the successful prosecution of the war, individual members are doing their best to ensure that there shall be no lack of the gold needed to back the credit of the Empire.

I venture to say that it is indeed fortunate, in these times, that the Empire possesses such a tremendous asset as the gold mines of the Rand.

In the early stages of the war it was found that there might be a shortage of gold coinage required to pay native wages, and arrangements were made to have the Mint at Pretoria in readiness to re-start operations. The mining houses were asked if they could produce gold suitable for making coins of

standard fineness, and, by the efforts of some of our members, succeeded in doing so.

Although it has not been found necessary to mint money here, thanks to our Navy, it is gratifying to know that it can be readily done should occasion ever arise.

Many of our members are taking active part in the war both in Africa and Europe, and amongst these I would mention Lieut. Ralph Stokes and Capt. Gerard Williams, both Corresponding Members of Council, and both in the Royal Engineers.

In the past it has been the fashion in inaugural addresses to give the Government a metaphorical "prod" and point out to them how lacking they are in their duty of encouraging and supporting the precious and base metal industries. I think however, that you will all agree that reminders of that sort may be well left to more peaceful and appropriate times, and that it is "up to us" at present, individually and collectively, to support the Government and to help bring about such a state of affairs in South Africa as will enable them to give the Empire all the aid and support this country is capable of giving.

As there are two interesting papers to be read before you this evening, I will not detain you longer, but before concluding I would once more appeal to our members, who are actively engaged in chemical, metallurgical and mining pursuits, to bring to the notice of the Society any improvements, observations or difficulties that may arise in the course of their daily work. Notes and papers of this description would be of great assistance to many of the younger members of the Society, some of whom are apt, I fear, to consider our proceedings as being above their heads and so lose interest in the Society.

On behalf of the Society I wish to take advantage of this opportunity to welcome back those of our members who have been on active service with the Forces in ex-German South-West Africa, and to congratulate them on the successful conclusion of that campaign and their safe return.

Mr G. Hildick Smith (*Member of Council*). I rise to propose a very hearty vote of thanks to our President for his Address to us this evening. I also thank him very much indeed for his kind words of welcome to those of us who have returned from active service in the South-West campaign. I heartily agree with Mr. Thomas's remarks with regard to the war. Every one of us at this time should think out carefully as to what is our immediate duty at the moment, as to whether or not each of us is doing most for the Empire by helping in the output of gold to furnish the hard cash necessary to win the war, or by volunteering for active service wherever the Empire requires men. Each man's decision will be influenced according to the nature of the work in which he is engaged. Whatever decision each of us comes to, we must all do our very utmost either on the mines or at the front, always remembering that every little will eventually tell and assist our Empire in the crisis which it is at present passing through.

The vote of thanks was agreed to by acclamation.

CYANIDE CONSUMPTION ON THE WITWATERSRAND.

By H. A. White, M.Inst.M.M. (Vice-President).

The annual report of the Chamber of Mines for the year 1914 gives figures for purchases of cyanide and total tons of ore milled, and assuming no great increase in the stocks on hand, it may be calculated that the average consumption of cyanide was 0·10 lb. per ton of ore treated, while zinc figured at 0·32 lb.

As our annual requirement of sodium cyanide is therefore, roughly, 5,000 tons, costing at present prices well over a million sterling, it is of great importance to ascertain exactly where all this goes and what are the possibilities of effecting any significant saving.

With the latter object in view, the Buying Committee of the Chamber of Mines recently called together the Consulting Metallurgists of the Rand, who, however, agreed that no application of known methods could be relied upon to materially reduce the cyanide consumption in an economic manner. The results of special efforts made, when the onset of war threat-

ened supplies, showed a loss of gold more than counterbalancing any saving of cyanide.

As, however, no published data were available giving details of the exact incidence of the known sources of loss, a sum of money was voted to enable experiments to be made with this object in view, under the control of a sub-committee consisting of W. A. Caldecott, K. L. Graham, E. H. Johnson, and the writer.

By the courtesy of the Committee, the results of all the experiments specially made and previously unpublished work in this direction are available for inclusion in this paper, and it is hoped that a full discussion of the matter will be of some benefit to the industry.

The bulk of the experiments were made in the Laboratory of the Consolidated Gold Fields of South Africa by Mr. C. A. Meiklejohn, to whom thanks are due for the careful and accurate work done, but the full responsibility for deductions made and opinions expressed is assumed by the author.

Preliminary Observations.

A theoretical limit to the minimum consumption of cyanide is set by the fact that the ore, crushed in water, carries into the collected charges a large amount of water as moisture: this water must be brought up to the cyanide strength required for extraction.

Example:—assume 50 % sand at 15 % moisture to 0·05 % KCN.
50 % slime at 40 % moisture to 0·01 % KCN.

This would give a loss of 0·12 lb. KCN per ton of crushed ore, and the exact figure will depend on proportion between sand and slime and the moisture with each.

Another way of looking at this (in reality less exact) is to note that as the solution in stock is always kept at a certain average KCN strength, the water brought into the plant as moisture must be brought up to that strength by the addition of KCN, one way or another, and this KCN must go to the residues or be dissipated elsewhere. This dilution effect is, of course, minimised by crushing in cyanide solution.

If we add to this the cyanide converted by contact to KCNS, $K_4Fe(CN)_6$, etc., we get a limit which cannot be improved upon in practice, and indeed we must also add the self-decomposition of the works solution while standing in sumps.

The cyanide going to the residue dumps in various forms in reality must represent the dilution loss, less the decomposition and HCN or other gas losses. Any saving in these two direction must therefore be partly discounted by increase in cyanide sent to residue dumps.

No attention need be paid to cyanide used in dissolving Cu, Ag, Au, Hg, or Zn, which merely have the effect of temporary accumulators.

The theoretical minimum cyanide consumption for Geduld, based on 60 % of the pulp, treated as slime, collected in water as a 40 % pulp (that is, 40 tons of water with 60 tons of dry slime) and 40 % of sand of 15 % moisture (that is 15 tons of water to 85 tons of dry sand) after allowing for cyanide required to raise the associated water to 0·3 lb. NaCN per ton for slime and to 1·0 lb. per ton for sand, amounts to 0·19 lb. per ton of ore treated. To this must be added 0·07 lb. converted into unavailable products as detailed in Table B. The irreducible minimum of 0·26 lb. NaCN per ton is thus arrived at, and this shows a possible saving in this case of only 0·06 lb. per ton if there were no losses in the form of gas. (See Table B.)

Of course, a saving could be effected if lower cyanide strengths were possible without counterbalancing losses of gold in the residues, but that is not generally the case, and the point must be emphasised that any saving of cyanide by means of increased alkalinity or in any other way of reducing gas losses must be partly offset by increased amount of loss in residue moisture.

It is, however, a theoretical possibility that the CO_2 of the boiler flue gases might be utilized to displace the HCN in our residues both of sand and slime, and in the former case the use of cyanide destroyers previous to sand filling would thus be avoided and the HCN collected in alkali instead.

Experimental Results (C. A. M.).

M. 1. *Estimation, as NaCN, of the amount of total cyanide, ferrocyanide, and sulphocyanide in sand residues as discharged at Simmer Deep.*

The sample of liquor in which the determinations were made was taken at leaching pipe from sand vat 15 minutes before the commencement of discharging. A moisture sample was taken on a freshly exposed face of sand during discharge.

Analysis of Solution.

NaCN.		KCN Per cent.	KCN per cent.	KCN per ton of Pulp in Sand.
0·0259	Total cyanide	0·019	= 0·019	= 0·0334
0·0042	Pot. ferrocyanide	0·0029	= 0·0031	= 0·0054
0·0043	Pot. sulphocyanide	0·0048	= 0·0032	= 0·0056
0·0344			0·0253	= 0·0444

Percentage of moisture in sand as discharged, calculated on dry sand 16·73 %.

Monthly percentage for June in total tonnage treated

Sand 52·19 %.

$0{\cdot}0253 \times 0{\cdot}5219 \times 20 \times 0{\cdot}1673$

0·0111 lbs. KCN per ton of pulp in sand treated.

0·0311 lbs. NaCN per ton of pulp in sand treated.

Sample from sump of weak solution used for last washes on sand vats

KCN = 0·022 %

M. 2. *Simmer Deep. Estimation, as NaCN, of the amount of total cyanide, ferrocyanide and sulphocyanide remaining with settled charge of slime after treatment and before same is discharged.*

31 cub. ft. of settled pulp = 1 ton dry slime.

Percentage of solution to dry slime = 59·81 %.

Monthly percentage for June of slime in total tonnage treated 47·51 %.

Analysis of Solution.

NaCN.		KCN Per cent.	KCN per cent.	KCN per ton of Pulp in Slime.
0·0925	Total cyanide	0·021	= 0·021	= 0·1193
0·0119	Pot. ferrocyanide	0·0026	= 0·0027	= 0·0153
0·0175	Pot. sulphocyanide	0·0050	= 0·0040	= 0·0227
0·1219			0·0277	= 0·1573

Solution sample taken immediately before discharge.

$0{\cdot}0277 \times 0{\cdot}1751 \times 20 \times 0{\cdot}5984$

0·1573 lbs. KCN per ton of pulp in slime treated.

0·1219 lbs. NaCN per ton of pulp in slime treated.

M. 3. *Estimation same as No. 1 on sand residues as discharged at Rose Deep.*

The sample of liquor on which the various determinations were made was obtained by taking a sample of the sand in vat during discharge into a sample bottle containing a known volume of lime water, and the results calculated on total liquor present.

Analysis of Solution.

NaCN.		KCN Per cent.	KCN per cent.	KCN per ton of Sand.
0·0489	Total cyanide	0·0115	= 0·0357	= 0·0631
0·0049	Pot. ferrocy.	0·0011	= 0·0034	= 0·0036 = 0·0064
0·0045	Pot. sulphocy	0·0015	= 0·0047	= 0·0032 = 0·0057
0·0583			0·0425	= 0·0752

$0{\cdot}0125 \times 0{\cdot}58 \times 20 \times 0{\cdot}1525$

0·0752 lbs. KCN per ton of sand.

0·0583 lbs. NaCN per ton of sand.

Percentage of moisture in sand as discharged, calculated on dry sand, 15·25%.

Weight of liquor present as moisture in sample of sand taken for testing ... ... 285 gm.

Weight of liquor added ... ... 600 gm.

885 gm.

$0{\cdot}0115 \times \frac{885}{285}$ 0·0357

$0{\cdot}0015 \times \frac{885}{285}$ 0·0047

$0{\cdot}0011 \times \frac{885}{285}$ 0·0034

M. 4. *Rose Deep. Estimation, as NaCN, of the amount of total cyanide, ferrocyanide and sulphocyanide remaining with settled charge of slime after treatment, and before same is discharged.*

31·9 cub. ft. of settled pulp = 1 ton dry slime.

Percentage of solution to dry slime = 72·50 %.

Monthly percentage for June in total tonnage treated slime 12·0%.

Analysis of Solution.

NaCN.		KCN Per cent.	KCN per cent.	KCN per ton of Slime, lb.
0·0519	Total cyanide	0·0110	= 0·0110	= 0·0670
0·0046	Pot. ferrocyanide	0·0009	= 0·00095	= 0·0059
0·0088	Pot. sulphocyanide	0·0028	= 0·00188	= 0·0114
0·0653			0·01383	= 0·0843

$0{\cdot}01383 \times 0{\cdot}42 \times 20 \times 0{\cdot}725$

0·0843 lb. KCN per ton of slime.

0·0653 lb. NaCN per ton of slime.

Sample of solution taken shortly before discharge.

M. 5. *Estimation, as NaCN, of total nitrogen in a sample of zinc-box precipitate from Simmer Deep Mine.*

The sample examined was received from the Simmer Deep Reduction Works, and was taken by them during mid-monthly clean up.

Moisture	51·34%
Total weight of precipitate for half month (wet)	35,738 lb.
Estimated for month at (wet)	71,476 lb.
Estimated for month at (dry)	34,780 lb.
Total tonnage sand and slime treated for June	63,100 tons
Nitrogen in dried sample of precipitate as NaCN	0·98

=0·0054 lb. NaCN per ton treated.

M. 6. Examination of cyanide solutions from Simmer Deep and Rose Deep Mines for total nitrogen present.

(1) Sample of last drainings from sand vat. Taken 15 minutes before commencement of discharge. (Simmer Deep.)

(2) Sample of last few inches of solution above settled slime charge after treatment and just prior to discharge. (Simmer Deep.)

(3) Solution from sample of sand residue. Sample taken into measured volume of water. (Rose Deep.)

(4) Same as No. 2. (Rose Deep.)

No. 3 was the solution obtained from a sample of sand residue. Dry weight, 1 lb. 4 oz. Moisture, 14·18 on dry sand.

cc. water in sand residue ...	274
cc. water added	600

	Sand S.D.	Slime S.D.	Sand R.D.	Slime R.D.
	1	2	3	4
Moisture present on dry residue	15·62	59·84	14·18	72·50
KCN per cent. ...	0·0200	0·0200	0·0110	0·0120
KCNS as per cent. KCN	0·0038	0·0042	0·0010	0·0023
K_4FeCy_6 as KCN...	0·0024	0·0024	0·0013	0·0021
	0·0262	0·0266	0·0133	0·0169
Total nitrogen as per cent. KCN	0·0272	0·0278	0·0136	0·0179
Previous tests ...	0·0263	0·0277	0·0141	0·0138

Percentage of nitrogen indicated (as KCN) in excess of that present as KCN, KCNS and $K_4Fe(CN)_6$.

1.	2	3	4
0·001	0·0012	$0·0003 \times \frac{874}{274}$	0·0010
		0·00096	

of other known loss

4	1	2½	6

The results from above six sets of experiments are gathered together in Tables "A" and "B", together with the average of 10 similar sets at Geduld added for the sake of comparison. (See next page.)

In Table "A" the results are arranged to show losses in sand and slime residues separately, while in Table "B" the arrangement is on a chemical basis only.

It will be observed:

(I.) That an increased proportion of the pulp treated as slime results in increased cyanide consumption owing to the greater amount of cyanide-bearing moisture in slime residues.

(II.) That smaller loss per ton of slime at Rose Deep is due to lower cyanide strength required for treatment at that plant.

In Table "B" it will be noted that from 40 to 50 of the total loss of cyanide is due to the escape of HCN as gas into the air as determined by difference, the smaller amount at Geduld being probably due to the higher alkalinity maintained there.

EXPOSURE OF SUMP CYANIDE SOLUTIONS TO ATMOSPHERE.

W. 7. Experiments were conducted in uniform glass jars, placed on a board above the sump. Jars held about 300 cc. and gave a surface exposure of 84 sq. ft. per ton of solution, compared with 5 sq. ft. in a sump 6 ft. deep.

No correction is made for evaporation of water.

The results appended show clearly (see Table "C" on next page):—

(1) Loose or incomplete covering is practically useless in preventing loss of alkali or cyanides.

(2) With weak solutions no loss of HCN as total cyanide and but little loss as free cyanide occurs while available protective alkali is present.

(3) With strongest solutions the loss of HCN both as free cyanide and total cyanide becomes serious when available protective alkali is below 0·01 NaHO.

(4) Comparing results published by Williams, this *Journal*, Vol. IV., p. 501, it is evident that the presence of zinc enormously decreases the loss of HCN by hydrolysis, etc.

TABLES A. AND B. ANALYSIS OF CYANIDE LOSS AS NaCN. June, 1915.

A MINE.	Sand	lb. per ton Sand as Solution in Residue	lb. per ton pulp due to sand.	Slime	lb. per ton Slime as Solution in Residue.	lb. per ton Pulp due to Slime.	lb. per ton in Zinc Boxes.	lb. per ton Ammonia Compounds, etc.	lb. per ton as HCN gas by difference.	as HCN, etc.	Total lb. per ton Pulp.
Simmer Deep ..	52·5	0·0655		47·5	0·2566	—	—	—	—	—	—
	—	—	0·0344	—	—	0·1219	0·0054	0·0066	0·1367	44·7	0·305
Rose Deep ...	58·0	0·1005		42·0	0·1555	—	—	—	—	—	—
	—	—	0·0583	—	—	0·0653	(0·0054)	0·0064	0·1326	49·5	0·268
Geduld ...	20·0	0·0900	—	60·0	0·2480	—	—	—	—	—	—
	—	—	0·0360	—	—	0·1488	(0·0054)	(0·0065)	0·1233	38·5	0·320

B. MINE.	Loss as Cyanides	Loss as sulpho-cyanides	Loss as ferro-cyanides	Loss as Ammonia compounds, etc.	Loss in Zinc Boxes	Evaporated as HCN gas (by difference)	loss as HCN gas etc.	Total loss as NaCN per ton.
Simmer Deep ...	0·1484	0·0248	0·0161	0·0066	0·0054	0·1367	44·7	0·305
Rose Deep ...	0·1008	0·0133	0·0095	0·0064	(0·0054)	0·1326	49·5	0·268
Geduld ...	0·1274	0·0574	Tr.	(0·0065)	(0·0054)	0·1233	38·5	0·320

(Brackets enclose figures not directly determined but interpolated from results from the other mines.)

TABLE "C."

No.	Sump.	Type.		Initial	1 Day	2 Days	3 Days	4 Days	5 Days	6 Days	7 Days
I.	Slime	A	Free KCN	0·010	0·010	0·010	0·009	0·010	0·010	0·010	0·010
II.			Total KCN	0·015	0·015	0·015	0·015	0·015	0·015	0·015	0·015
III.			NaHO	0·0164	0·0161	0·0160	0·0160	0·0160	0·0160	0·0160	0·016
IV.		C	Free KCN	0·010	0·010	0·008	0·008	0·010		—	0·005
V.	(Open)		Total KCN	0·015	0·015	0·015	0·015	0·015		—	0·011
VI.			NaHO	0·0164	0·0116	0·0080	0·0032	0·0016			Acid
VII.	Medium	A	Free KCN	0·018	0·017	0·018	0·019	0·017	0·014	0·018	0·018
VIII.			Total KCN	0·032	0·032	0·032	0·032	0·032	0·032	—	0·032
IX.			NaHO	0·0196	0·0192	0·0188	0·0180	0·0186	0·0180		0·012
X.		B	Free KCN	0·018	0·018	0·018	0·019	0·016	0·014	0·012	0·012
XI.			Total KCN	0·032	0·032	0·032	0·032	0·031	0·033		0·033
XII.			NaHO	0·0196	0·0148	0·0112	0·0072	0·0032	Trace		Acid
XIII.		C	Free KCN	0·018	0·017	0·017	0·018	0·016	0·012	—	0·008
XIV	(Open)		Total KCN	0·032	0·032	0·032	0·032	0·032	0·032		0·035
XV.			NaHO	0·0196	0·0152	0·0101	0·0072	0·0012	Acid	—	Acid
XVI.	Strong	A	Free KCN	0·096	0·098	0·098	—	0·097		0·092	0·095
XVII			Total KCN	0·118	0·118	0·118		0·118		0·118	0·118
XVIII			NaHO	0·0156	0·0156	0·0152	—	0·0160		0·0152	0·0152
XIX		C	Free KCN	0·096	0·097	0·091	—	0·057		0·071	0·073
XX.	(Open)		Total KCN	0·118	0·118	0·115		0·106		0·100	0·096
XXI.			NaHO	0·0160	0·0116	0·0032		0·0016		0·0088 (checked)	0·0084 (checked)

A—tightly closed. B—lightly covered. C—Fully open. (Evaporation loss up to 20% in 7 days.)

(5) The recrudescence of alkaline strength observed in experiment XXI. was independantly checked and can possibly be accounted for by change of KCN to K_2CO_3.

G. 8. *Laboratory Experiments: Open vessel against closed vessel. Atmospheric temperature. 9 in. evaporating dishes used.*

Solution No. Remarks.

1. Strong stock sand solution (in open vessel).
2. Strong stock sand solution (in air-tight bottle).
3. New strong solution in open vessel (no protective alkalinity).
4. New strong solution in stoppered bottle (no protective alkalinity).
5. Mill slime solution (in open vessel).
6. Mill slime solution (in air-tight bottle).
7. New weak solution in open vessel (no protective alkalinity).
8. New weak solution in air-tight bottle (no protective alkalinity).

Solutions tested for cyanide:

(1) To formation of white opalescence.

(2) To formation of silver iodide precipitate in presence of excess caustic soda.

The figures given for "KCN after allowing for evaporation" represents the percentage of KCN that should have been present at the conclusion of the experiment, had no decomposition taken place.

The alkalinity is expressed in terms of NaOH:

(1) Without addition of $K_4Fe(CN)_6$.

(2) With addition of $K_4Fe(CN)_6$.

The experiments were conducted under varying conditions.

(a) At ordinary temperatures (indoors).

(b) At temperatures as near 50° F. as possible. Solutions kept outside on verandah during the day, and inside at night.

(c) At temperatures as near 100° F. as possible.

	(1)	(2)	(3)	(1)	(5)	(6)	(7)	(8)
Total cyanide	0·16	0·16			0·018	0·018		
Free cyanide	0·11	0·14	0·136	0·136	0·017	0·017	0·022	0·022
Protective alkalinity as per cent.								
CaO	0·014	0·014			0·006	0·006	—	—

NOTE.—The solutions were allowed to stand in the vessels without stirring.

TABLE "D." DETERMINATION OF FREE CYANIDE.

Solution.	Original Strength.	6 Hours.	23 Hours.	29 Hours.	47 Hours.	53 Hours.	71 Hours.	77 Hours.	95 Hours.
1	0·140	0·122	0·089	0·080	0·070	0·066	0·056	0·050	0·043
2	0·140	0·139	0·139	0·138	0·138	0·136	0·134	0·134	0·152
3	0·136	0·126	0·100	0·085	0·063	0·046	0·024	0·016	0·007
4	0·136	0·135	0·134	0·134	0·134	0·134	0·133	0·133	0·133
5	0·017	0·017	0·013	0·009	0·009	0·008	0·006	0·006	0·004
6	0·017	0·017	0·017	0·017	0·017	0·017	0·017	0·017	0·016
7	0·022	0·018	0·009	0·005	0·005	0·004	0·003	0·003	0·002
8	0·022	0·021	0·021	0·021	0·021	0·020	0·020	0·020	0·019

No allowance for loss by evaporation.

M. 9. *Exposure to the atmosphere, under varying conditions of working cyanide solutions.*

The vessels used were large glass beakers 4 in. diameter.

Volume of solution at commencement: 1000 cc. in each case.

Surface area at commencement, per ton of solution, 80 sq. ft.

(d) At ordinary temperatures, with varying cyanide and varying alkalinity. Solutions kept outside on verandah during the day and inside after 5 p.m. (16 solutions).

(e) With percentage KCN same, but with varying alkalinity in each case. The alkalinity being brought up to strength daily.

TABLE "E."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ...	0·004	0·0092	0·0032	0·0076	0·0030	0·0064	0·002	0·0052
Do. at finish ...	—	0·002	—	0·002	—	0·002	—	0·002
KCN at start ...	0·070	0·085	0·051	0·063	0·033	0·043	0·015	0.002
KCN after allowing for evaporation ..	0·0784	0·0923	0·057	0·070	0·0369	0·0477	0·0164	0·0244
KCN at finish ...	0·031	0·043	0·017	0·028	0·011	0·0195	0·0065	0·015
KCN decomposed ...	0·0474	0·0493	0·040	0·042	0·0259	0·0282	0·0099	0·0694
KCN loss per cent. in 8 days ...	60·5	53·5	70·2	60·0	70·2	59·1	60·8	38·5

(a) Four solutions of cyanide from Simmer Deep Mine as received. Test lasted eight days. Loss by evaporation about 7 . Average temperature of solution 50° F.

TABLE "F."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ...	0·0040	0·0092	0·0032	0·0076	0·0030	0·0064	0·0020	0·0052
Do. at finish ..	—	0·0030		0·0030		0·0020	—	0·0020
KCN at start ...	0·070	0·085	0·051	0·063	0·033	0·043	0·015	0·022
KCN after allowing for evaporation ...	0·0784	0·0923	0·057	0·070	0·0369	0·0477	0·0164	0·0244
KCN at finish ..	0·032	0·048	0·018	0·033	0·012	0·021	0·0085	0·0150
KCN decomposed .	0·0464	0·0443	0·039	0·037	0·0249	0·0267	0·0079	0·0094
KCN loss per cent. in 8 days ..	59·2	47·9	68·4	52·9	67·5	55·9	48·2	38·5

(b) Samples of same solutions as in Table "E". Experiment lasted eight days. Loss by evaporation about 7 . Average temperature of solution, 17° F.

TABLE "G."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ...	0·0040	0·0092	0·0032	0·0076	0·0030	0·0064	0·0020	0·0052
Do. at finish	0·0024	0·0260	0·0040	0·0170	0·0046	0·0048	0·0040	0·0024
KCN at start	0·070	0·085	0·051	0·063	0·033	0·043	0·015	0·022
KCN after allowing for evaporation ..	0·308	0·374	0·2288	0·2816	0·1452	0·1892	0·066	0·0968
KCN at finish .	0·058	0·105	0·320	0·066	0·009	0·021	—	0·010
KCN decomposed ...	0·250	0·269	0·1968	0·2156	0·1362	0·1682	0·066	0·0868
KCN loss per cent. in 3 days .. .	81·0	71·0	86·0	76·5	93·8	88·9	100·0	89·6

(c) Samples of same solution as in Table "E" and "F". Experiment lasted three days. Loss by evaporation about 66 . Average temperature of solution 90° F.

TABLE "H."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ...	—	0·084	—	0·049	—	0·0184	—	0·0112
Do. at finish ...	—	0·030	—	0·008	—	0·0070	—	0·0070
KCN at start ...	0·068	0·083	0·068	0·084	0·068	0·083	0·069	0·083
KCN after allowing for evaporation ...	0·0806	0·0984	0·0806	0·0996	0·0806	0·0984	0·0818	0·0984
KCN at finish ...	0·075	0·087	0·068	0·079	0·056	0·065	0·044	0·053
KCN decomposed ...	0·0056	0·0114	0·0126	0·0206	0·0246	0·0334	0·0378	0·0454
KCN loss per cent in 8 days	6·9	11·6	15·6	20·6	30·5	33·9	46·2	46·1

(d) 1. KCN 0·068—0·083 . Alkalinity as NaOH 0·084, 0·049, 0·0184 and 0·0112 . Average temperature, 55° F. Experiment lasted eight days.

TABLE "I."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ..	—	0·104	—	0·055	—	0·0304	—	0·0128
Do. at finish ...	—	0·0304	—	0·008	—	0·0080	—	0·0064
KCN at start .	0·053	0·064	0·053	0·064	0·053	0·064	0·057	0·063
KCN after allowing for evaporation ...	0·063	0·0749	0·063	0·0759	0·063	0·0759	0·0676	0·0748
KCN at finish ...	0·057	0·069	0·053	0·0645	0·039	0·053	0·027	0·036
KCN decomposed ..	0·006	0·0069	0·010	0·9114	0·024	0·0229	0·0406	0·0388
KCN loss per cent in 8 days	9·5	9·1	15·9	15·0	38·1	30·2	58·1	51·3

Loss by evaporation, 13·2 %.

(d) 2. KCN 0·053—0·064 . Alkalinity as NaOH, 0·104, 0·055, 0·0304 and 0·0128 . Average temperature, 55° F. Experiment lasted eight days. Loss by evaporation, 13·2 % .

TABLE "J."

	1		2		3		4	
		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.		With Ferrocyanide.
Alkalinity at start ...	—	0·085	—	0·048	—	0·026	—	0·008
Do. at finish ...	—	0·029	—	0·008	—	0·004	—	0·0048
KCN at start ...	0·027	0·033	0·027	0·032	0·027	0·033	0·025	0·031
KCN after allowing for evaporation ...	0·0323	0·0395	0·0323	0·0385	0·323	0·0395	0·0299	0·0371
KCN at finish ...	0·0260	0·363	0·0260	0·0330	0·019	0·0250	0·0110	0·0110
KCN decomposed ...	0·0063	0·0035	0·0063	0·0055	0·0133	0·0145	0·0189	0·0261
KCN loss per cent. in 8 days	19·5	9·7	19·5	14·3	41·1	36·7	63·2	70·3

(d) 3. KCN, 0·027—0·033 . Alkalinity as NaOH, 0·085, 0·048, 0·026 and 0·008 . Average temperature, 55° F. Experiment lasted eight days. Loss by evaporation, 13·2 %.

TABLE "K"

	1		2		3		4	
		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide
Alkalinity at start ...	—	0·087	—	0·049	—	0·028	—	0·009
Do. at finish ...	—	0·0312	—	0·0072	—	0·0048	—	0·0048
KCN at start ...	0·014	0·022	0·013	0·0225	0·013	0·0225	0·013	0·022
KCN after allowing for evaporation ...	0·0168	0·0263	0·0156	0·0269	0·0156	0·0269	0·0156	0·0263
KCN at finish ...	0·0170	0·0250	0·015	0·0230	0·0120	0·0210	0·0110	0·016
KCN decomposed ..	nil	0·0013	0·0006	0·0039	0·0036	0·0059	0·0046	0·0103
KCN loss per cent. in 8 days	nil	4·1	0·4	12·6	23·1	21·9	29·5	39·1

(d) 1. KCN, 0·013–0·0225 . Alkalinity as NaOH, 0·087, 0·049, 0·028 and 0·009 . Average temperature, 55° F. Experiment lasted eight days. Loss by evaporation, 13·2 .

The conclusions to be drawn from a careful consideration of the above exposure experiments, which are sufficiently comprehensive and consistent when account is taken of the various sources and exposure conditions, may be briefly stated as follows:

(1) The loss is incomparably greater in pure synthetic solutions, even with added alkali, than is the case with ordinary working solutions.

(2) Tables "E," "F," and "M" illustrate the very heavy loss shown in the presence of little or no protective alkali, when determined with addition of ferrocyanide. This suggests that it is safer to omit the addition of ferrocyanide in works tests.

TABLE "L"

	1		2		3		4		5		6	
		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide		With Ferrocyanide
Alkalinity at start ...	0·036		0·028		0·024		0·0172		0·011		0·0076	
Alkalinity at finish ...	0·024		0·016		0·010		0·0092		0·001		0·0061	
KCN at start	0·038	0·054	0·038	0·054	0·038	0·054	0·038	0·054	0·038	0·054	0·038	0·054
KCN after allowing for evaporation	0·0421	0·0604	0·0421	0·0604	0·0421	0·0604	0·0421	0·0604	0·0421	0·0604	0·0424	0·0604
KCN at finish	0·035	0·051	0·036	0·051	0·036	0·049	0·036	0·049	0·037	0·048	0·036	0·047
KCN decomposed ...	0·0074	0·0094	0·0064	0·0094	0·0064	0·0114	0·0061	0·0114	0·0054	0·0124	0·0064	0·0134
KCN loss per cent. in 5 days ...	17·4	15·5	15·0	15·5	15·0	18·8	15·0	18·8	12·7	20·5	15·0	22·2

(e) Six Solutions.—KCN 0·038—0·054 in each case. Alkalinity as NaOH, 0·036, 0·028, 0·024, 0·017, 0·011 and 0·0076 . Alkalinity kept as near as possible to original strength by daily addition of fresh lime. Loss by evaporation made up at same time. Experiment lasted five days.

TABLE "M."

	3 ft. Pipe (1)	Top (2)	3 ft. Pipe.	Bottom.	6 ft. Pipe (1)	Top (2)	6 ft. Pipe (1)	Bottom (2)
Alkalinity at start ..	0·0068	0·0100	-	—	0·0068	0·0100	0·0068	0·0100
Do. at finish ...	0·0020	0·0064	-	-	0·0010	0·0056	0·0010	0·0056
KCN at start ...	0·070	0·087	-	—	0·0070	0·087	0·070	0·087
KCN after allowing for evaporation ...	0·0747	0·0929			0·0723	0·090	0·723	0·090
KCN at finish ...	0·050	0·070	—	—	0·050	0·069	0·050	0·069
KCN decomposed ...	0·0247	0·0229	-	-	0·0223	0·021	0·223	0·021
KCN loss per cent. in 12 days	33·0	24·6	—	-	30·8	23·3	30·8	23·3

(1) and (2) have same significance as in M. 9.

M. 10. *Exposure to atmospheric outside of cyanide solution in two iron pipes, 6 in. diameter, and 3 ft. and 6 ft. in length.*

Temperatures from freezing to .. ... 80° F.
Alkalinity, as NaOH ... ... 0·0010

	3 ft. pipe	6 ft. pipe
Loss by evaporation	6·8	3·4
Surface exposed per ton of solution ...	10·6 sq. ft.	5·3 sq. ft. 10·6 sq. ft.

Experiment lasted from 23rd June till 5th July.

(3) In Table "G" the increased temperature effect is strikingly apparent, but alkali protection is low.

(4) Tables "H," "I," "J" and "K" show the much improved results obtainable when the alkali strength (without ferrocyanide) is not allowed to get below 0·030 NaHO. The results shown in Table "C," where working conditions were attempted, more nearly indicate that decomposition is small as long as real protective alkali is present.

(5) In Table "L" the attempt was made to keep the alkaline strength up by daily additions of lime, but of course the drop during the 24 hours' exposure could not be avoided.

If reference be made to Dr. Moir's valuable note (C.M.M.S., Vol. XVI., p. 9) in our last *Journal*, it will be clear that the addition of alkali is of considerable benefit in reducing hydrolysis, which, however, is of secondary importance compared with the protection it gives against decomposition by atmospheric carbon dioxide, which is the principal source of the loss of HCN gas.

G. 11. *Losses in working plant: In decantation slime vat.*

In order to confirm the great loss of cyanide on weak slime solutions (shown in Table "D"), determinations were made in a decanting vat. Results are as follows: —

Slime D treatment vat after first transfer from collectors.—Tests for loss of KCN and alkali by standing after being transferred and during period of decanting gold solution to clarifiers.

TABLE "N."—STOPPERED BOTTLE USED ON MEASURING ROD.

Depth from Surface	Free KCN	Alkali	Total KCN	Time taken after finish of Transfer
Ft.	Per cent.	Per cent. CaO	Per cent.	Hours.
0·5	0·025	0·004	0·027	19
3·0	0·025	0·004	0·027	19
6·0	0·025	0·004	0·027	19
0·5	0·025	0·004	0·027	39
3·0	0·025	0·004	0·027	39
6·0	0·025	0·004	0·027	39
0·5	0·024	0·004	0·025	48
3·0	0·024	0·004	0·025	30
6·0	0·023	0·004	0·025	54
8·0	0·023	0·003	0·025	57

(A much smaller decomposition is evident.)

M. 12. *Estimation of amount of cyanide consumed during the treatment of a charge of collected slime from the time of leaving the collectors till immediately after transfer to the first settlement vat.*

Simmer Deep Mine.

31 cubic ft. of settled slime pulp = 1 ton of dry slime.

Percentage of solution to dry slime ... ... 59·84

Weight of dry slime in charge... 289·6 tons

Weight of water in charge ... 173·3 tons

	KCN	Total Tons.
Cyanide solution in intermediate vat before transfer	0·045	751·65
Cyanide solution used to transfer charge from collector to intermediate vat	0·0245	135·29
Cyanide solution used to transfer charge from intermediate to first settlement vat	0·0245	255·56

Solid cyanide added to intermediate vat, 100 lb. NaCN = 129 KCN.

Summarizing the above there is therefore present to commence with

tons		
751·65)		KCN.
135·29)	*at* 0·0252	559·8 lb.
255·56)		

Total 1142·50

Solid sodium cyanide added 100 lb. 129·0 lb.

688·8 lb.

	tons.
And present in first settlement vat	1142·50
Plus water with charge ...	173·30
Total	1315·80

1315·8 tons *at* 0·0026 = 684·2 lb.

∴ cyanide consumed = 4·6 lb

4·6 : 289·6 :: 0·0158 lb. KCN per ton = 0·0122 lb. NaCN per ton of slime treated.

Loss of cyanide indicated during second transfer (to second settlement vats)

Tons solution left with pulp before transfer from first to second settlement		203·36 tons
Tons solution in second settlement vat before transfer ...		912·00 tons
Tons solution used in transfer		190·00 tons.
203·36 tons	0·025	101·68 lb.
912·00 tons	0·026	474·24 lb.
190·0 tons	0·0255	98·90 lb.
1305·36 tons		672·82 lb.
1305·4 tons *at* 0·0255 total KCN		665·74 lb.
Difference		7·08 lb

7·08 : 289·6 :: 0·0243 lb. KCN per ton of slime treated = 0·0188 lb NaCN per ton of slime treated

The loss as shown above is :

From collectors to first settlement 0·0122 lb. NaCN per ton of slime.

From first to second settlement 0·0188 lb. NaCN per ton of slime.

It should, however, be pointed out that it is extremely difficult to arrive at a satisfactory figure for the consumption of cyanide in this part of the process, *e.g.*, the percentage of KCN in the first settlement vat should have been 0·02617 , whereas that actually found tested 0·026 . This shows a loss of 0·00017 KCN = 0·0122 lb. NaCN per ton of slime treated.

In the transfer from the first to second settlement the loss indicated is 0·00027 KCN = 0·0188 lb. NaCN per ton of slime treated.

A considerable loss of cyanide takes place during the settlement and decantation in the settlement vats. In each vat the solution loses 0·001 KCN during the time the charge is under treatment.

KCN in solution in first settlement vat full	0·026
KCN in solution in first settlement vat before transfer to second settlement	0·025
KCN in solution in second settlement vat	0·0255
KCN in solution in second settlement vat, full just before re-discharge	0·0245%
Ratio of solution to slime, first settlement	4·545 : 1
Ratio of solution to slime, 2nd settlement	4·505 : 1

It will be seen that the loss of cyanide during this part of the treatment is considerable, and if this is taken as representing the loss by evaporation of HCN during the slime treatment, it will account for upwards of 0·11 lb. NaCN per ton of slime treated.

M. 13. *Rose Deep.* It was intended to calculate the loss, similarly, on a charge of slime at the Rose Deep Mine, but, owing to a mishap to the sample of solution used to transfer the slime charge from the slime collector to the Trent agitator, this was not done. Accepting, however, the figures as supplied by the Rose Deep Reduction Dept., for percentage of KCN in this solu-

tion, the consumption of NaCN per ton of slime treated works out at 0·0442 lb. per ton, of which nearly 50% takes place during the agitation in the "Trent".

This 0·0442 lb. NaCN per ton represents cyanide consumed.

(1) During transfer from collector to "Trent".

(2) During agitation in "Trent".

(3) During transfer from "Trent" to first settlement vat.

M. 14. *Action of Mill Service water on working cyanide solutions.*

(1) A sample of the overflow from the return sand cones at Simmer Deep Mine was filtered, and the filtrate mixed, in equal volumes, with a cyanide solution from the "strong" sump.

(2) A second sample was taken from the water supply at mortar boxes and treated similarly.

The purpose was to ascertain what action, if any, the mill service water would have on the cyanide in solution.

Percentage of KCN Present.

Before mixing	0·084
After mixing	0·042%

The mixed solutions were kept in stoppered bottles, and tested (*a*) at the end of 42 hours, and (*b*) at the end of 72 hours.

No decomposition of the KCN in solution could be observed at the end of 72 hours.

M. 15. *Examination, for total nitrogen, of samples of sand before and after treatment with cyanide.*

The samples were received from the Simmer Deep Reduction Works. They were taken (1) at sand table during the charging of a sand vat, and (2) during the discharge of same material from sand vat after treatment, and were *washed* and dried before examination for nitrogen.

In both cases a small amount of nitrogen was indicated. If anything, the sample taken after treatment showed less than the sample taken before treatment, but the amount present in either case was very small.

G. 16. *Regenerative Experiments in Model Brown Vat:*

Pulp ratio, 2½ to 1.

Test 1.—To determine Cy. loss in open vat without regeneration.

Test 2.—To determine Cy. loss in open vat without regeneration.

Test 3.—To determine Cy. loss in open vat without regeneration.

Test 4.—To determine Cy. loss in closed vat with regeneration.

Test 5.—To determine Cy. loss in closed vat with regeneration.

In Test 5, the cyanide regenerated 28·9% of total cyanide lost during agitation. In Test 3 "open", where alkalinity was maintained the results are even better than "closed" Tests 4 and 5.

G. 17. *Regenerative Experiments in Laboratory: in confirmation of above using absorption towers.*

It was found that the total amount of cyanide lost during 10 hours' air agitation was 53·1% of original contents of solution.

Of this 53·1%, 53·8% was recovered by using absorption towers, which equals a recovery of 28·6% of the total cyanide (used).

W. 18. *Regeneration of cyanide from cyanates and sulphocyanides.*

TABLE "O."

Vat.	Test.	Original Solution.		KCN Loss in Hours Pounds per Ton Slime.				Remarks.
		KCN	Alkaline.	8 hrs.	12 hrs.	24 hrs.	36 hrs.	
Open	1	0·030	0·0125	0·10	0·10	0·20	0·50	Alkalinity allowed to go down
Open	2	0·023	0·0125	0·14	0·14	0·35	0·60	Alkalinity allowed to go down
Open	3	0·025	0·0100	0·05	0·10	0·20	0·25	Alkalinity kept up by fresh additions of lime
Closed	4	0·030	0·0110	0·10	0·10	0·20	0·30	Alkalinity kept up by fresh additions of lime
Closed	5	0·022	0·0060	—	—	0·20	0·30	With low alkalinity kept up to 0·006 during agitation.

(1) *In the cold.*—Solutions containing 0·1 of NaCNO and KCNS respectively were submitted to the action of an equal bulk of lead-coated zinc shaving each for one hour and for 24 hours. The resultant solutions were examined for cyanide both with silver nitrate and the prussian blue reaction, and in all cases no distinct trace of cyanide could be observed. In the case of the sulphocyanide no reaction for sulphuretted hydrogen could be obtained from the zinc shaving used. A further test using the very delicate reaction with phenolphthalein compared before and after zinc) showed that no cyanide is formed. As hydrococrulignone, benzidine, tolidine, etc., all give reactions with cyanates and sulphocyanides of this strength, no deductions could be drawn from the comparative experiments made.

(2) *Boiling hot.*—The above experiments were made with solutions kept boiling for 10 minutes, with a negative result.

It may be noted that the cyanate was partly reduced to formamide as shown by the rapid blackening of the silver nitrate test (after zinc), especially on boiling.

The President: In proposing a very hearty vote of thanks to Mr. White for his exceedingly valuable paper, I would like to say that this paper should be carefully studied by all our members who are in any way connected with cyaniding. In these times we should do all we can to reduce the consumption of cyanide: in fact, to save in any way. I will not ask for any discussion on this paper as yet, as we have another paper on "The Prevention of Hydrolysis in Cyanide Solutions," by Mr H. M. Leslie, and it has been decided to take the discussion on the two papers together. No doubt you will remember that Mr Leslie gave us a paper a few years ago on "Cyanide Practice on the Kolar Goldfields, India." Mr. Leslie was for some years our Corresponding Member of Council for Southern India.

The vote of thanks was agreed to

THE PREVENTION OF HYDROLYSIS IN CYANIDE SOLUTIONS.

By H. M. Leslie, A.Inst.M.M. (Member).

"As a matter of fact, a cyanide solution undergoes gradual decomposition, with evolution of hydrocyanic acid, even when an excess of caustic alkali is present, so that the protection afforded is only partial and temporary, and not absolute." This quotation appears in Clennell's book, "The Chemistry of Cyanide Solutions," 1910, and is stated as a recognised fact, but its full significance as applied to cyanide practice in general seems to have been missed.

In ordinary practice it has always been recognised that a great deal more cyanide is used for dissolving out the precious metals than is theoretically necessary, and various reasons have been put forward at various times to account, in individual cases, for high consumptions, but I think I am right in stating that no one has ever yet put forward a reason to account for this large excess of cyanide which is necessary in present practice, and which theory could be applied to cyanide practice in general.

It is with this object I write this paper, as the results of my work in this direction seem to warrant it.

The words I have quoted at the beginning contain three essential statements:

(1) A cyanide solution undergoes gradual decomposition.

(2) A cyanide solution evolves hydrocyanic acid.

(3) This decomposition is not prevented by the use of excess alkali.

The evolution of hydrocyanic acid means loss or consumption of cyanide in working practice, and it is this loss which I contend is accountable for the large excess of cyanide which has to be used over and above that which is necessary for efficient extraction.

Referring to our text books, we find that HCy is a colourless, volatile, inflammable liquid, is extremely poisonous when inhaled or taken in liquid form, and mixes readily with water. Its boiling point is 26·5° C. These properties of HCy must be admitted to have a very great bearing on the case. Compare the boiling points in the table given below. HCy has the lowest boiling point of all, and yet it is the product of the decom-

position of cyanide solutions, and practically no safeguards have ever been taken to prevent its escape. I might almost say that its smell has been recognised as part of the cyanide process for as many years as the process has been in existence.

Table 1.

Name.	Boiling Point.
Hydrocyanic acid	26·5° C.
Ether	35·0° C.
Carbon-disulphide	47·0° C.
Acetone	56·0° C.
Methyl alcohol	60·0° C.
Bromine	63·0° C.
Alcohol (absolute	78·0° C.
Benzene	80·0° C.

It is recognised that HCy can be evolved by reactions which take place when a cyanide solution is brought into contact with the ore to be treated, the extent depending upon the cyanicides present which are capable of doing this, also in water introduced into the circuit, but the point which we are dealing with has nothing to do with these, but refers only to cyanide solutions existing as such. Take a simple solution of KCy in water, the gradual decomposition of all such solutions must be due to a reaction which could apply generally, and this can only be accounted for by hydrolysis, which can be shown by the equation

$$KCy + HOH = KOH + HCy.$$

That this reaction correctly represents what does occur, my experiments have proved, and that such is possible is admitted by Clennel's statement originally quoted, viz., that "this decomposition is not prevented by the use of excess alkali".

I will now quote a few experiments in detail which demonstrate that the hydrolytic action can account for this gradual decomposition

Table 2

Shows the loss which takes place in a fairly strong simple cyanide solution at low temperatures.

Interval. Hours.	KCy Test. Per cent.	KCy Loss. Per cent.	Temperatures. Max. Fahr	Min. Fahr
4	0·368	7·56	46	43
24	0·232	37·00	45	42
22½	0·116	50·00	49	46
23½	0·028	75·80	46	43

Table 3

Shows the effect of higher temperatures on three strengths of simple cyanide solutions.

Interval. Hours.	KCy Test. Per cent.	KCy Loss. Per cent.	Temperatures. Degrees Fahr.
1st Solution, original strength 0·626 % KCy			
4½	0·474	24·28	77
20	0·240	49·30	53 to 62
2nd Solution, original strength 0·131 % KCy			
4½	0·063	51·90	77
20	0·000	100·00	56 to 62
3rd Solution, original strength 0·079 % KCy			
1½	0·020	74·70	77

Table 4

Shows the loss which takes place by agitating with air in Model Brown Agitators in two working strengths of simple cyanide solutions.

No. 1 Original Strength 0·212 per cent.			No. 2 Original Strength 0·108 per cent.		
Interval. hrs.	KCy Test. per cent.	KCy Loss. per cent.	Interval. hrs.	KCy Test. per cent.	KCy Loss. per cent.
¾	0·212	—	¾	0·107	0·910
½	0·211	0·471	½	0·105	2·777
½	0·208	1·886	½	0·105	2·777
½	0·208	1·886	½	0·104	3·705
½	0·206	2·827	½	0·102	5·555

It is generally conceded that the weaker the solution the more active is the hydrolytic action, and it will be noted in the foregoing experiments that the weaker the solution the greater is the percentage decomposition, and the following series of tests emphasise this in a very marked degree.

Table 5

Shows the decomposition of simple cyanide solutions of varying strengths. The solutions, 6,500 grains in each case, were contained in six shallow glass vessels, each 6 in. diameter by 1 in. deep, and were exposed to the atmosphere. Samples were withdrawn at intervals for tests, and average temperatures during period of exposure noted.

(1)

4 Hours Exposure. Temperature 16½° C.

No.	Per cent. KCy before.	Per cent. KCy after.	Difference.	Per cent. KCy Loss.
1	0·208	0·190	0·018	8·6
2	0·138	0·123	0·015	10·8
3	0·101	0·089	0·015	14·1
4	0·075	0·060	0·015	20·0
5	0·052	0·041	0·011	21·1
6	0·043	0·033	0·010	23·2

(2.)

18¾ Hours' Exposure.			Temperature 15° C.
1	0·190	0·154	0·036	18·9
2	0·123	0·094	0·029	23·5
3	0·089	0·065	0·024	26·9
4	0·060	0·038	0·022	36·6
5	0·041	0·022	0·019	46·3
6	0·033	0·016	0·017	51·5

(3)

4 Hours' Exposure.			Temperature 17½° C.
1	0·151	0·138	0·016	10·4
2	0·094	0·082	0·012	12·7
3	0·065	0·054	0·011	16·9
4	0·038	0·028	0·010	26·3
5	0·022	0·015	0·007	31·8
6	0·016	0·010	0·006	37·5

(4)

20 Hours' Exposure.			Temperature 16° C.
1	0·138	0·098	0·040	29·0
2	0·082	0·052	0·030	36·5
3	0·054	0·026	0·028	52·0
4	0·028	0·010	0·018	64·3
5	0·015	0·004	0·011	73·3
6	0·010	0·002	0·008	80·0

TABLE 6 hereunder,

Shows the protection afforded by varying quantities of alkali added to a solution of simple cyanide. Experiment conducted in similar vessels to 5.

(5)

1 Hours' Exposure.			Temperature 17½° C.
1	0·098	0·075	0·023	23·5
2	0·052	0·035	0·017	32·7
3	0·026	0·016	0·010	38·4
4	0·010	0·004	0·006	60·0
5	0·004	0·000	0·004	100·0
6	0·002	0·000	0·002	100·0

(6)

18 Hours' Exposure.			Temperature 16° C.
1	0·075	0·035	0·040	53·3
2	0·035	0·012	0·023	65·7
3	0·016	0·004	0·012	75·0
4	0·004		0·004	100·0
5	—	—	—	—
6	—	—	—	—

(7)

1½ Hours' Exposure.			Temperature 18° C.
1	0·035	0·022	0·013	37·0
2	0·012	0·008	0·004	33·3
3	0·004		0·004	100·0
4	—	—	—	—
5	—	—	—	—
6	—	—	—	—

These six solutions at the end of the experiment were tested for alkali, and it was found, after making the necessary allowance for evaporation, that this alkali agreed in all cases with that originally added as cyanide, and decomposed according to the equation

$$KCy + HOH = KOH + HCy.$$

Interval	1 NaOH Nil		2 NaOH 1 lb. per ton		3 NaOH 2 lb. per ton		4 NaOH 3 lb. per ton		5 NaOH 4 lb. per ton		6 NaOH 5 lb. per ton	
	Strength of KCy at Start											
	0·06 per cent.		0·06 per cent.		0·06 per cent.		0·06 per cent.		0·06 per cent.		0·06 per cent.	
Hrs.	Test per cent	Loss per cent	Test per cent	Loss per cent	Test per cent	Loss per cent	Test per cent	Loss per cent	Test per cent	Loss per cent	Test per cent	Loss per cent
24	0·03	50	0·012	30	0·06	nil	0·06	nil	0·06	nil	0·06	nil
24	nil	100	0·007	88·4	0·014	76·7	0·016	73·4	0·022	63·4	0·028	53·4
24	nil	nil	nil	100	0·002	96·7	0·003	95·0	0·005	91·7	0·008	86·7
24	nil	nil	nil	nil	nil	100	nil	100	nil	100	nil	100

These experiments demonstrate very clearly the following points:—

(1) That simple cyanide solutions decompose by the hydrolysis of the solution

(2) That the percentage loss by this reaction is greater in a given time, the weaker the solution.

(3) That increased temperature accelerates this decomposition.

(4) That the alkali formed as a product of the hydrolytic action has little or no protective action on the remaining cyanide, so that hydrolysis goes on until all the cyanide is destroyed.

(5) That the protection afforded by the addition of an excess of caustic alkali is by no means complete, and that the protection is only of a very temporary nature.

Table 7

Shows the protection afforded to a simple cyanide solution by means of a cover. In one the contents of the vessel were exposed to the atmosphere, and in the other the vessel was covered with a luted cover. Eight litres in each case testing 0·25 KCy. Samples were withdrawn at intervals for test. Area in each case 265 sq. in.

Interval. Hours	KCy Test per cent		Loss Open Vessel		Loss Closed Vessel.	
	Open Vessel.	Closed Vessel.	Per cent. on Interval	Per cent on Original	Per cent. on Interval.	Per cent on Original
24	0·234	0·25	6·4	6·4	0·000	0·0
24	0·225	0·248	3·85	10·0	0·800	0·8
24	0·208	0·248	7·55	16·8	0·000	0·8
24	0·186	0·246	10·57	25·6	0·807	1·6
24	0·168	0·246	9·66	32·8	0·000	1·6
48	0·140	0·245	16·66	44·0	0·405	2·0
24	0·126	0·245	10·00	49·6	0·000	2·0
24	0·111	0·244	11·19	55·6	0·408	2·4
48	0·083	0·242	25·22	66·8	0·816	3·2
24	0·072	0·242	13·25	71·2	0·000	3·2
48	0·056	0·240	22·22	77·6	0·826	4·0
24	0·045	0·239	19·60	82·0	0·417	4·4
24	0·037	0·238	17·77	85·2	0·418	4·8
24	0·031	0·238	16·22	87·6	0·000	4·8
24	0·023	0·238	25·80	90·8	0·000	4·8
24	0·016	0·237	30·43	93·6	0·420	5·2
48	0·008	0·233	50·00	96·8	1·700	6·8
24	0·005	0·233	37·50	98·0	0·000	6·8
48	nil	-	-	100·0	-	-

The temperatures in this experiment were noted and averaged 42° F. minimum to 51·5° F. maximum daily. The lowest temperature recorded being 38° and the highest 57°. The loss in the covered vessel is very small compared with that in the open vessel, and the loss in the covered vessel might have occurred when withdrawing the samples, as same were withdrawn through a hole in the cover, which was kept open while the test was being made. The remaining liquor in the open vessel was carefully measured and tested for alkali. The alkali found, after allowing for evaporation, was calculated to its equivalent in KCy, and was found to equal 0·25 KCy on the original liquor, confirming that the loss was due in accordance with the equation.

$$KCy + HOH = KOH + HCy.$$

From the foregoing it will be at once apparent that a very great loss of cyanide must occur in actual practice where no precautions are taken to safeguard this evaporation of HCy, and that the covering of the vessels show that protection can be afforded by this means, and which I will hereafter call the "Closed" system. This point was very thoroughly investigated in the laboratory on samples of ore from different parts of the world, and the laboratory work was further checked by working tests in a Model Plant on lots varying from 100 lb. to 1½ tons. Under every circumstance this loss by hydrolysis was confirmed, and it was clearly proved that it could be prevented by the "Closed" system so far as these tests went.

These tests, although conclusive to me, knowing how they were conducted, I recognise would not serve to convince others, as the proof of such a theory must be made under actual working conditions, and in a locality where the cyanide process is in active operations.

As there is no place in the world where the cyanide process is used to such an extent as the Witwatersrand, I will pass on to the work which has been done here.

Early facilities were kindly given by the Rand Mines, Limited, for this work to be carried out, and at the Village Deep Mine a start was made. The proposition was, first, to prove that there really was a loss of cyanide going on which could be prevented by the "Closed" system, and, second, if so, was it sufficient to justify any modification in the plant.

The work was therefore confined in the first instance to ascertaining what loss of cyanide by evaporation was taking place in the plant as it stands. Except on the slime plant (decantation method), it was found impossible to do this with any degree of accuracy, owing to the difficulty of getting any data over any lengthy period from which calculations could be made, due to the intermittent treatment and mixing of the liquors in circuit.

The slime plant is briefly as follows:—

1. The slime collectors, used for collecting all the slime preparatory to cyanide treatment.

2. The Trent Agitator, to which the collected slime is transferred for agitation with cyanide solution.

3. Decantation vats, first series, to which the charges from the agitator are pumped along with cyanide solution, and allowed to settle, and the liquors decanted off.

4. Decantation vats, second series, to which the settled slime from the first series is transferred along with some fresh cyanide liquor, and the slime allowed to settle. The

clear liquor being used partly for making up the strong liquor in the first series decantation vats, and partly for the liquor used in the agitator.

5. The clarifiers, to which the liquor that is continuously being decanted off the first series decantation vats is passed, and filtered before reaching the extractor boxes.

6. Six extractor boxes are used for the liquors from this plant.

7. Sumps. No sumps are kept separate for this plant. The sumps are common to both the sand and slime plants.

The treatment was followed from day to day, and regular tests were made of the liquors employed from the time the charges from the agitator were transferred to the decantation vats, and until they were discharged to the slime dam. It was found, as a result of these tests, that there was a considerable loss of cyanide going on continuously, which was entirely due to evaporation, but as the liquor in the slime plant, other than that actually in use in the decantation vats, was in a state of constant circulation, it was impossible to arrive at a satisfactory figure representing the total loss due to evaporation, during the whole course of the treatment. A definite figure was, however, obtained in so far as the decantation vats were concerned, during the time the liquor in these vats was settling and decanting.

To arrive at this an exhaustive series of tests was carried out, and it was found that, during this period of treatment, there was a loss of cyanide going on, due to evaporation, amounting to 0·0013 % of KCy per 24 hours, in all the liquors in use at this stage of the treatment. As there are three vats in each series charged per 24 hours, and each vat receives 580 tons of liquor per charge, the estimate of the loss of cyanide due to evaporation, at this stage of the slime treatment, will amount to 32,572 lb. of cyanide (100 % strength) per annum.

This figure does not take into account the losses, due to evaporation, which must of necessity occur in the other parts of the plant, viz., the clarifiers, the agitator, the extractor boxes, and the sumps, on which, owing to the continual changing and mixing of the liquor, it was impossible to determine the amount with any degree of accuracy.

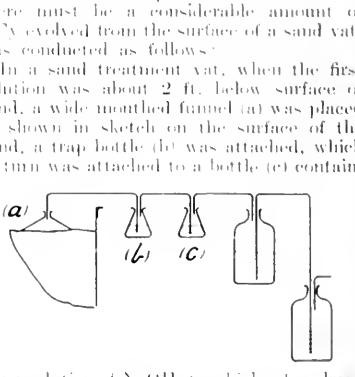
An approximate idea of the losses occurring in the clarifiers may, however, be arrived at by taking the following data into consideration. All the liquor from the first series decantation vats passes through the clarifiers. There are three of these decantation vats charged daily, and the total surface area exposed to the atmosphere amounts to 58,905 sq. ft. The liquor from this first series amounts to 1,740 tons per 24 hours, which is half the liquor on which the former calculation was based. There are two clarifiers, the total surface area of which is equal to 3,927 sq. ft., and taking the same percentage of loss per 24 hours as on the decantation vats would show on this tonnage of 1,740 tons per 24 hours a loss per annum of 10,857 lb. of cyanide (100 % KCy).

If to the observed loss of 32,572 lb., 10,857 lb. are added, a total loss of 43,429 lb. of cyanide per annum is shown. This figure, although representing a large percentage of the total loss of cyanide by evaporation in the slime plant, does not represent the total loss from this cause. It will serve to demonstrate that the loss caused by the hydrolysis of the cyanide solutions, and the consequent volatilisation of the HCy thus formed is a very important factor in the economy of the cyanide process, and one which if guarded against should bring about a very considerable saving in the cost of treatment.

The work was extended to the liquors used in the sand plant, and although it was shown that loss by evaporation is continually taking place, it was found impossible to get at any reliable figures on which to base a calculation, but the tests made, all indicated that the loss, caused by the formation and volatilization of HCy, was taking place all through the process.

An interesting experiment, showing that there must be a considerable amount of HCy evolved from the surface of a sand vat, was conducted as follows:

In a sand treatment vat, when the first solution was about 2 ft. below surface of sand, a wide mouthed funnel (a) was placed as shown in sketch on the surface of the sand, a trap bottle (b) was attached, which in turn was attached to a bottle (c) contain-



ing a solution of NaOH, to which a few drops of KI solution, and 0·1 cc. $AgNO_3$ was added, which gave the solution a yellowish-green

tint. Two aspirator bottles (winchesters) were attached and arranged so that air could be drawn from (a) through (b) and (c). After about 10 winchesters of air had been drawn through in this way the AgI was dissolved, due to HCy forming NaCy in the bottle (c). This test is only qualitative, as there is no ratio between it and actual work, but it serves to show that HCy is evolved in measurable quantities in a given time.

The results of these preliminary tests indicating such a considerable loss, the next step was to demonstrate, that if this loss was due to the reaction claimed, that it could be prevented by the "Closed" system.

This work was carried out in a small demonstrating plant, similar to the one previously mentioned. It was made of steel, and was capable of treating slightly over a ton of sand or about 600 lb. of slime at a time, and so arranged that the whole cycle of treatment common to the cyanide process could be carried out therein in the same manner as in large scale practice. It was erected on a convenient site for charging, and had connections fitted, so that the actual liquors used in large scale work could be run into this plant when required. The measurements of all liquor run into, or out of, this demonstration plant were all checked by the cyanide staff of the mine, and check samples of all solutions taken by them for independent test.

The following experiments give the details of the work done in connection with the sands, and are given in the order in which they were carried out (see table on next page):—

"A" Test.—Open system (present method).

"B" Test.—Closed system. (new method).

These two tests were conducted in the following manner: "A" as an open or ordinary cyanide treatment test; "B", exactly as in "A" except that the vats were all covered in before starting the test. The endeavour was to follow the ordinary treatment as applied to the sand plant on Village Deep, and was as follows:—

The percolating vat was charged with sand taken from one of the sand treatment vats as lowered from the sand collectors. Sufficient (A liquor) strong cyanide solution, was run into the sump and measured. Samples being taken whilst running in. This liquor was then pumped on to the sand and allowed to drain slowly into the empty sump, at the same rate as in large scale work. When drained dry the same solution was pumped back on sand and the draining repeated. This in turn was again pumped back on sand and drained as before. It was then measured and sampled and afterwards discharged. The object of these repumpings was for these two washes to take the place of the "B" liquor as used in large scale practice. Sufficient "C" liquor as used in the sand plant to supply four "C" washes was then run into sump. This was measured and sampled and then pumped into the reservoir, and added to the sand in four separate washes, the rate of washing and draining following as closely as possible the times as occupied in large scale work. After all the "C" liquor had run through the sand it was measured and sampled and the sand discharged.

In the following three tests: C, "Closed System"; D, Open System (present method) and E, "Closed System", the procedure was the same as in the two previous tests, except that the "A" liquor run into sump was pumped on to sand, and then "B" liquor, sufficient for two washes, was run into the empty sump, and measured, and pumped to reservoir ready for applying to the sand as washes instead of circulating the spent "A" liquor for this purpose as in former tests. The "A" and "B" liquor drainings were collected together, measured, sampled, and then discharged, and the "C" liquor run in as before.

From the tabulated results, if the consumptions of cyanide obtained in the "Open" and "Closed" systems are averaged, the following mean figures are obtained.—

Average Consumption of Cyanide.

	Cyanide KCy.	Per ton.
Open System	100	0·6375 lb.
"Closed" System ...	100	0·4283 lb.
Open System	130	0·490 lb.
"Closed" System ...	130	0·329 lb.

This shows a difference of

0·2092 lb. per ton 100 KCy, or 33

0·1640 lb. per ton 130 KCy, or 33

in favour of the "Closed" System per ton of sand treated.

It will be noted that the cyanide consumption shown by these tests is higher than is obtained in actual large scale practice. This is a point only to be expected, as the chances and opportunities for decomposition by the formation of insoluble cyanides are very much greater than obtain in large scale work. The plant used was not protected by any cyanide proof paint, and the surface of iron exposed to its action was very much greater

SUMMARY OF SAND TESTS.

	A Open	B Closed.	C Closed.	D Open.	E Closed.
Inches A liquor added	10½	10½	10¾	10¾	10
Pounds liquor added	694	694	711	711	661·35
Test of A liquor	0·210	0·190	0·201	0·199	0·210
Cyanide in A liquor ... lb.	1·458	1·318	1·429	1·4149	1·388
Inches B liquor added	nil	nil	15⅞	16	16
Pounds B liquor added	nil	nil	1,050	1,058	1,058
Test of B liquor	nil	nil	0·041	0·042	0·031
Cyanide in B liquor ... lb	nil	nil	0·430	0·4413	0·328
Inches C liquor added	31½	31½	31⅜	31¾	31⅞
Pounds C liquor added	2,083	2,067	2,100	2,100	2 108
Test of C liquor	0·012	0·012	0·0115	0·0115	0·010
Cyanide in C liquor ... lb.	0·250	0·218	0·241	0·2415	0·211
Inches 1st solution recovered... .	7½	8½	23⅛	21½	24⅛
Pounds 1st solution recovered	196	562	1,534	1,105	1,595
Test of 1st solution...	0·062	0·048	0·082	0·084	0·075
Cyanide in 1st solution ... lb.	0·307	0·269	1·257	1·1802	1·1962
Inches 2nd solution recovered...	30⅜	31½	31¼	32⅜	31½
Pounds 2nd solution recovered ...	2,008	2,083	2,067	2,141	2,083
Test of 2nd solution	0·029	0·038	0·0175	0·013	0·0135
Cyanide in 2nd solution ... lb.	0·582	0·791	0·361	0·2783	0·281
Total cyanide *added* ... lb.	1·708	1·566	2·100	2·1007	1·927
Total cyanide *recovered* ... lb.	0·889	1·060	1·618	1·4585	1·477
Difference—cyanide consumed ..	0·819	0·506	0·482	0·6412	0·450
Dry weight of charge	2,147	2,104	2,175	2,118	2,130
Cyanide consumed per ton 100% KCy, lb.	0·669	0·420	0·413	0·606	0·422
Cyanide consumed per ton 130% KCy, lb.	0·514	0·322	0·340	0·466	0·324

in preparation. Fresh oxidation was continually taking place, and it was observed that prussian blue was formed thereon, this was particularly noticeable when the vats were emptied. In the case of a large plant, such as the Village Deep, which has been in every day use for a considerable time, this loss should not take place to any extent, as the interior of the vats becomes coated with a hard deposit, which by preventing contact between the iron and the cyanide has a tendency to prevent any loss occurring from this source.

The consumption of cyanide on the Village Deep cyanide plant is somewhat over 0·300 lb. 130% KCy per ton, or 0·390 lb. 100% KCy. If these figures are used as representing the actual conditions, and the "Closed" System consumption reduced by the difference between these actual conditions and the actual consumption as shown for the Open System by these tests. (The difference representing excess consumption due to small scale plant), the comparative figures will be as follows:—

Average Consumption of Cyanide.

	Cyanide KCy.	Per ton.
Open System	100%	0·390 lb.
"Closed" System ...	100	0·1806 lb.
Open System	130	0·300 lb.
"Closed" System ...	130	0·139 lb.

These figures thus show a saving in cyanide brought about by the protection afforded by the "Closed" system of:

0·2092 lb. per ton 100% KCy, or 54%
0·1610 lb. per ton 130% KCy, or 54%

This saving of 0·2092 lb. 100% KCy is a very considerable figure when multiplied by the number of tons treated in even one month, and it makes no allowance for any cyanide which can be recovered when the gases are expelled from the plant at the time of running on the various cyanide solutions washes. The reason this was not estimated was on account of it being found that the pressure created by the running on of, say, a solution to the leaching vat, was too slight to cause any gas to be forced over into the regenerating apparatus provided. The only time this did

occur was in running in liquor into the empty sump, and as this was mainly air, there was only a trace of cyanide recovered, too small to be taken into account. This would not be the case in large scale practice, where the sumps are continually in use and where large volumes of liquor are used. This figure would amount to quite an appreciable quantity per month. Further, in this series of tests a valve had to be opened in the cover whenever a measurement of liquor had to be made, thus breaking the circuit and releasing any pressure in the covered vats. The main figures only are given, and do not show the amount of testing and checking of which these results are the outcome.

One very interesting and significant figure was noted which has a very important bearing on the effectiveness of the "Closed" system. The liquors in use were continually being tested at the various stages of the treatment of the sands. In the Open system it was observed that the "C" liquor lost strength right up to the finish of the treatment, the last washes losing 0·00175 and 0·003% respectively. In the "Closed" system, however, it was observed that as the "C" liquor was applied no loss was apparent in the liquor, as it came through the sand at the same strength at which it was applied, showing that the ore and plant had taken up all the cyanide they required, and owing to the covers, the loss was gradually stopped, these last two washes simply acting as a wash for soluble gold, without any decomposition.

The saving as indicated by the foregoing tests which could be effected by the introduction of the "Closed" system at the Village Deep, is shown and admitted to be between 50% and 60% of the total amount of cyanide used on that mine, and if this is based on the 0·3 lb. per ton consumption previously mentioned it would mean if the price of cyanide is taken at, say 1s. 1d. per lb., a total annual saving on a tonnage of 50,000 tons per month, equal to £6,600, or 2·64d. per lb.

A further series of trials were carried out to determine the loss of cyanide during agitation with air in Brown vats, and to ascertain what saving could be effected it protection was given by covering in these vats. It should be noted, however, that in all these tests the agitation was much more violent than is used or necessary in ordinary large scale work, as it was very difficult to regulate the supply of air on such a small plant, from the compressed air main. The results are, however, comparative, and again demonstrate that by the adoption of the "Closed" system a very considerable saving can be effected.

A number of trials were first carried out to determine the proper charges, slime to solution, air pressure, etc. In these tests it was observed that with excessive agitation the consumption increased, as for example, the following test will show it compared with the next open test.

Excessive Agitation Open Test.

Water added	160 lb.	
Slime added, 30 buckets ...	802 „	
A Solution added ..	116 „	
A Solution Test 0·207 „ KCy	0·24 lb.	cyanide
Pulp ratio	71·48	water
Total weight of charge ...	1,378 lb.	
Weight of water present ...	985 „	
Calculated test	0·02438	KCy
Test at start of agitation ...	0·021 „	„
Test after 17 hours agitation	0·0055 „	
Loss per charge	0·1858 lb.	
„ „ ton (100 KCy) ...	0·945 „	„ cyanide
„ „ „ (130 KCy) ...	0·727 „	„

In the following three tests the air was reduced to the minimum consistent with efficient agitation, and the air valve so regulated that each was supplied with the same pressure of air in each case.

Loss 130° KCy per ton after	1 Open. lb.	2 Closed. lb.	3 Closed lb.
5 min.	0·141	0·048	0·054
35 min.	0·217	0·075	0·062
1 hr.	0·217	0·084	0·080
1½ hr.	0·256	0·094	0·080
2 hr.	0·256	0·091	0·088
3 hr.	0·296	0·121	0·125
4 hr.	0·331	0·158	0·125
5 hr.	0·110	0·158	0·114
6 hr.	0·127	0·158	0·161
22½ hr.	0·677	0·328	0·332
18 hr.			0·194

Saving Shown by "Closed" System over Open System.

Saving per ton 100° after	No. 3.	No. 4.	Average
5 min.	0·121	0·111	0·1175
35 min.	0·184	0·201	0·1925
1 hr.	0·172	0·177	0·1745
1½ hr.	0·211	0·228	0·2195
2 hr.	0·211	0·218	0·2145
3 hr.	0·227	0·222	0·2245
4 hr.	0·226	0·268	0·2470
5 hr.	0·329	0·317	0·3380
6 hr.	0·350	0·315	0·3475
22½ hr.	0·154	0·148	0·1510

Saving per ton 130 KCy after	No. 3.	No. 4	Average.
5 min.	0·0093	0·0087	0·0090
35 min.	0·0142	0·0155	0·0148
1 hr.	0·0133	0·0137	0·0135
1½ hr.	0·0162	0·0176	0·0169
2 hr.	0·0162	0·0168	0·0165
3 hr.	0·0175	0·0171	0·0173
4 hr.	0·0173	0·0206	0·0189
5 hr.	0·0252	0·0266	0·0259
6 hr.	0·0269	0·0266	0·0267
22½ hr.	0·0349	0·0345	0·0347

Summary of Agitation Tests.

	1 Open.	2 Closed.	3 Closed.
	lb.	lb.	lb.
Water added ..	470	470	464
Slime added ..	993	1,051	1,064
V Solution added	165	165	182
	per cent.	per cent.	per cent.
V .. Test KCy	0·202	0·189	0·166
	lb.	lb.	lb.
Cyanide added	0·333	0·311	0·302
	per cent.	per cent.	per cent.
Pulp Ratio water	71·25	70·00	70·00
	lb.	lb	lb.
Total weight of charge	1,628	1,686	1,710
Weight of water present	1,160	1,180	1,197
	per cent.	per cent.	per cent.
Calculated Test KCy .	0·0287	0·0263	0·0252
Test 5 min. after start	0·025	0·025	0·02375
.. 35	0·023	0·02425	0·0235
.. 1 hr.	0·023	0·024	0·023
.. 1½	0·022	0·02375	0·023
.. 2	0·022	0·02377	0·02255
.. 3	0·021	0·023	0·02175
.. 4	0·020	0·022	0·02175
.. 5	0·018	0·022	0·02125
.. 6	0·0175	0·022	0·02075
.. 22	0·011	0·01725	0·016
.. 48		–	0·0115
Loss per charge 5 mins.	0·013	0·016	0·018
.. .. .. 35 ..	0·066	0·025	0·021
.. .. .. 1 hr.	0·066	0·028	0·027
.. .. .. 1½ ..	0·078	0·031	0·027
.. .. .. 2 ..	0·078	0·031	0·030
.. .. .. 3 ..	0·080	0·040	0·042
.. .. .. 4 ..	0·101	0·052	0·042
.. .. .. 5 ..	0·125	0·052	0·048
.. .. .. 6 ..	0·130	0·052	0·054
.. .. .. 22½ ..	0·206	0·108	0·111
.. .. .. 48 ..			0·165
Loss per ton 100 KCy 5 min.	0·184	0·063	0·070
.. .. 35 ..	0·282	0·098	0·081
.. .. 1 hr.	0·282	0·110	0·105
.. .. 1½ ..	0·333	0·122	0·105
.. .. 2 ..	0·333	0·122	0·115
.. .. 3 ..	0·385	0·158	0·163
.. .. 4 ..	0·431	0·205	0·163
.. .. 5 ..	0·531	0·205	0·187
.. .. 6 ..	0·555	0·205	0·210
.. .. 22½ ..	0·880	0·426	0·432
.. .. 48 ..		–	0·643

In the tests it was found possible to work the regenerators, and with the pressure of air regulated to suit the speed of agitation the air and gas evolved during the trials could bubble through the regenerators continuously.

As there was too little liquor to allow of as many samples being taken as in the case of the cyanide liquor samples, the regenerators were only tested at the end of 6 and 24 hours in Nos. 3 and 4 tests, and 18 hours in No. 4, and they are as follows:—

Recovery of Cyanide in Regenerators.

Cyanide recovered per charge after	No. 3. lb.	No. 4. lb.
6 hr.	0·0065	0·00588
22½ hr.	0·0192	0·01924
18 hr		0·047
Cyanide recovered per ton 100 KCy		
6 hr.	0·0256	0·0229
22½ hr.	0·0758	0·0750
18 hr.		0·1832
Cyanide recovered per ton 130 KCy		
6 hr.	0·0197	0·0176
22½ hr.	0·0583	0·0577
18 hr		0·1409

To ascertain what the "Closed" system has accomplished in these slimes tests, the above figures must be deducted from the losses previously given, and the saving thus shown is proportionately greater.

Losses of Cyanide in Agitation Tests.

Allowance being made for recovery. Loss per charge after	Open No. 2. lb.	Closed No. 3 lb.	Closed No. 4. lb.
6 hr.	0·130	0·0155	0·04812
22½ hr.	0·206	0·0888	0·09176
18 hr.	–	–	0·118
Loss per ton 100 KCy			
6 hr.	0·555	0·1791	0·1871
22½ hr.	0·880	0·3502	0·357
18 hr.	–	–	0·4598
Loss per ton 130 KCy			
6 hr.	0·127	0·1383	0·1431
22½ hr.	0·677	0·2697	0·2743
18 hr.			0·3531

Saving Shown by "Closed" System over Open System.

Allowance being made for recovery.	No. 3.	No. 4.	Average.
Saving per ton 100 KCy			
	lb	lb	lb.
6 hr.	0·3756	0·3679	0·37175
22½ hr.	0·7295	0·523	0·52625
Saving per ton 130 KCy			
6 hr	0·2887	0·2836	0·28615
22½ hr	0·1073	0·1027	0·105

The foregoing work having been completed the next step was to demonstrate on a larger scale, and it was arranged that the test should be carried out at the Modder "B" Mine. On the score of economy it was decided to confine the trial to the Brown Agitators, and for this purpose two of these vats were covered in. This arrangement as it turned out was unfortunate, as my investigations showed that the conditions prevailing in that company's plant were altogether abnormal when compared with the rest of the Rand. This was due entirely to the method of precipitation in use, which necessitated the use of a very high alkalinity being kept up in the liquors. When I say unfortunate I mean unfortunate so far as the test was able to demonstrate an actual saving in practice, but the results arising out of the investigations only go further to prove that the loss by hydrolysis and the very partial and temporary protection afforded by the use of caustic alkali are two very important factors in large scale practice.

The first thing I noted on going over to Modder "B" was that the daily cyanide tests of the charges to the Browns and the tests of same taken just prior to their being discharged to the stock vats showed almost invariably no loss of cyanide. By using a weaker standard solution I was, however, able to get a closer end-point, and found that there was invariably a small loss (about 0·001%) which was confirmed in the works when they used a similar strength of solution.

The covers being duly finished and a regenerator provided the plant was tested, but the result showed only a trace of cyanide caught, showing that very little HCy was being given off, and that the loss indicated by the forementioned tests was not due primarily to evaporation of HCy.

The Browns were tested separately, and, as will be seen from the following table of tests, there is not much difference between the Open and "Closed" Browns. The difference is, however, in favour of the "Closed" system, but it is too small to be of much practical use. This small difference could only be noted by using two or three times the usual volume of solution for each test and dividing the result by two or three as the case might be. This explains the number of decimal places to which these tests were carried.

Note.—The closed Browns were not airtight, and it was found difficult with the arrangement provided to pass all the air displaced through the solution in the regenerator at the time of filling the Browns, as the pressure at that stage was so great. Further, after filling, it was difficult, owing to the leakage in the cover and the varying pressure of air to keep a stream of air (introduced for agitation purposes) from these vats constantly passing through the regenerator. Sufficient air was, however, passed through to test whether or not it would be of any advantage to have regenerators installed, with the result as stated.

Losses in Brown Tanks at Modder "B."

(1) Open.			(2) Closed.			(3) Closed.		
Start.	Finish.	Loss.	Start.	Finish.	Loss.	Start.	Finish.	Loss.
0·015	0·0145	0·0005	0·01575	0·015	0·00075	0·017	0·017	0·000
0·017	0·0155	0·0015	0·017	0·016	0·001	0·017	0·016	0·001
0·018	0·165	0·0015	0·018	0·0165	0·0015	0·0175	0·01633	0·00117
0·0193	0·018	0·0015	0·019	0·01775	0·00125	0·019	0·01775	0·00125
0·017	0·0155	0·0015	0·0185	0·01725	0·00125	0·0165	0·016	0·0005
Average		0·0013	Average		0·00115	Average		0·000784

Note.—"Closed" in this table means only partially closed, as there were holes left in the cover which allowed escape of air.

The small loss by evaporation of HCy had already been foreseen in my preliminary examination of the liquors before the covers were completed. This was entirely due to the very high alkalinity of the liquors used, which, taking the average over the period included in the foregoing table (and which is about the ordinary strength registered) alkali in terms of the CaO equal to 0·0254 . The strength of the cyanide used over the same period averaged 0·01715 . With liquors of such high alkalinity, hydrolysis is

to a very large extent prevented in so far as the time under observation is concerned, and while this test indicates that protection is afforded the time of observation was too limited to determine what the actual loss throughout the whole time the liquors are in use would be. It might be advanced that in the case of a Brown vat with its constant agitation and constant changing of surface, that a test of this nature would be most suitable to test the "Closed" system. This I grant in the case of solutions or pulp having little or no free alkali, but not with such as are dealt with in this case. Against this ever-varying surface in an agitator must be put the fact that by means of this same agitation the alkali is also being brought into constant touch with any HCy which is being liberated. This does not occur to anything like the same extent in still solutions, where the protective action of the alkali has to depend entirely on diffusion, so that at the surface of a still liquid the evaporation is bound to be greater (where protective alkali is present) than with the case of a moving and changing liquid where the alkali is being constantly brought into action on the surface, the only possible place where any HCy can escape. It should also be noted that the period of agitation was only about 8 or 9 hours, and the slime is only under treatment for 24 hours at the longest. Further, in considering this question of the evaporation of HCy, it should be remembered that the evaporation is entirely dependent on the decomposition of the cyanide in solution, and that it is not evaporated in the same sense as water is evaporated.

The question as to whether or not any loss of cyanide could take place at any other stage in the treatment was investigated. First in the transfer, where the pulp from the slime collectors is transferred to the Browns with cyanide liquor (alkali being introduced at this stage in the form of lime). It was found that by calculating the amount of cyanide added, and allowing for the dilution by the water contained in the pulp as transferred, that the calculated strength of what should be in the Browns at the commencement of the agitation, worked out the same as that found by actual test, showing that there was no loss of cyanide at this stage. Further, after the treatment in the Brown vats was completed and the pulp had been transferred to the stock vat, and the Butters filter, it was found that the liquor drawn off the Butter's leaves from this pulp, tested the same as the works test at the end of the agitation in the Brown vats.

To further satisfy myself on this question of the alkalinity of the solution in use, I took samples from all the cyanide liquor storage vats, with the following results:—

	CaO		CaO
No. 1	0·02214	No. 6	0·02576%
No. 2	0·02181	No. 7	0·01200%
No. 3	0·02128	No. 8	0·03248%
No. 4	0·02700	No. 9	0·01648%
No. 5	0·02214	No. 10	0·02200%

No. 11 0·01588 CaO.

It will be apparent from the foregoing table that a very large amount of alkali (lime) must be used to keep up this standard of alkalinity. This is caused by the use of the Merrill zinc dust precipitation process, which is used for the recovery of the gold. In ordinary extractor box practice the use of alkali to such an extent is neither expedient nor necessary, as it interferes adversely with the precipitation of the gold in the same manner as the want of it does in the Merrill process. This is quite a different proposition to the Village Deep one, where extractor boxes are in use, and it was unfortunate that out of the 50 odd mines on the Rand where ordinary extractor box practice prevails the first large scale test of the "Closed" system should have been made on one of the two plants (Modder "B" and Brakpan) where such unfavourable conditions exist.

At first sight it would look as if this loss by hydrolysis was not a practical point, and that any step towards the covering in of plant would be quite unnecessary, but I think I can show that even with the protection afforded by the use of such highly alkaline liquors there is still a loss on these plants going on which can only be accounted for by hydrolysis, and further, that the apparent advantage in using such liquors does not benefit cyanide practice to any extent in so far as consumption of cyanide is concerned.

At the time these tests were made I was informed that, roughly speaking, 800 tons of slime and about 620 tons of sand were treated per day, which amounts to 42,600 tons treated per month, and that the consumption of cyanide for this tonnage was about 54 cases, each containing 224 lb. 130 % cyanide, or equal to 0·3691b. 100% KCy per ton on the total tonnage.

As I have already shown, the consumption of cyanide on the slime only amounts

to about 0·001 , but to allow for error I put it at 0·002 . The weight of liquor used in the agitation of the slime per day is approximately 1,100 tons. This figure was found by averaging the daily specific gravities of the tests made, measuring each charge, and working out the daily average from Caldecott Slime Tables. The consumption thus works out to 0·055 lb. per ton of slime treated, or, say, 0·031 lb. per ton on the total tonnage. The state of the sand and slime as it is discharged to the waste dump was investigated, as with liquors containing so much protective alkali, the waste liquors of such a plant must run very much higher in cyanide than is usually found in ordinary practice elsewhere. It was impossible to get a proper sample of the slime pulp, as it is blown off the leaves of the Butters Filter into the hoppers with cyanide solution, and discharged to the dam, but the moisture of the pulp as it is discharged may be taken at 35 %. The strength of the solution used for washing cakes and blowing them off was averaged over 26 days, and worked out at 0·0261 KCy. This shows a loss of cyanide to the dam of 5,238 lb. 130 KCy per month, or 0·2837 lb. per ton of slime, equal to 0·1598 lb. per ton on the total tonnage. A sample of sand going to the dump was taken, the moisture estimated, and the cyanide in same determined. It showed 13 moisture, testing 0·0282 KCy, which equals a loss to the dump of 1,213 lb. 130 KCy per month, or 0·084 lb. per ton of sand, equal to 0·037 lb. 100 KCy per ton on the total tonnage.

Working out the consumption per ton on total tonnage from the above data, the following is the result:—

	lb. per ton.
(1.) Consumption by slime during treatment based on loss in strength of cyanide solution used at 0·002 KCy ...	0·031
(2.) Ditto, ditto by sand on same basis...	0·021
(3.) Cyanide in slime discharged to dam	0·1598
(4.) Cyanide in sand discharged to waste dump	0·037
Total	0·2578

This consumption of 0·2518 lb. per ton, deducted from 0·369 lb. (the total consumption) leaves 0·1172 lb. per ton to be accounted for; equal to 3,843 lb. 130 KCy per month. This cannot be put down to any loss in strength caused by the treatment which has been allowed for, it is based on the slime consumption, and a very liberal margin allowed, viz., double the loss found. It can therefore be only due to the hydrolysis of the solutions in use, as it has already been demonstrated that this is a very potent factor effecting the loss of cyanide in solutions.

It will thus be seen that when taking the consumption on this plant, and making allowance for the large amount of alkali which has to be kept up in these liquors, there can be very little, if any, saving effected in so far as actual working costs are concerned. Further, it will be conceded that the above figure of 0·1172 for loss by hydrolysis is a very moderate estimate, 1st, when the fact that there is a considerable quantity of cyanide brought back into circuit from the slime dam, which ought to be deducted from the figures given as being lost in discharge, and which, when allowed for, would increase the loss due to hydrolysis in proportion; 2nd, when it is noted that no allowance has been made for the ordinary evaporation of the solutions which takes places throughout the year.

This completes the investigations so far, and the results I think more than warrant the necessity for a remedy such as the "Closed" system being applied. The extent of this loss to the gold mining industry, where the cyanide process is in use, cannot be accurately determined, but its magnitude may be gauged by considering the following figures applying to the Rand alone. The tonnage amounts in round numbers to some 27,000,000 tons cyanided per annum, and if the cyanide consumption is averaged at the very moderate estimate of 0·300 lb; 130 KCy, a saving of 50 of this amount would mean a total of 1,250 tons of a value of £162,000 per annum (cyanide at 1/4 per lb). Or if 50 % is taken as the amount which could be saved, it would mean 2,025 tons of cyanide, or £270,000 per annum. These are very big figures, but in my opinion they show conclusively the enormous saving which can be effected by the introduction of the "Closed System for the Treatment of Ores by Cyanide."

The President: In proposing a vote of thanks to the author for his paper, the remarks that apply to Mr. White's paper apply also to his. Both these papers should give much food for thought amongst cyan-

ade men, and although we may not all be able to carry out experiments in the way these have been done, I have no doubt the chemical members of our Society will see into the matter. A great many of our cyanide members do not believe in a high protective alkali; and I think it is their duty to come along and tell us why they do not.

Mr. C. Toombs *(Member of Council):* A weak point in these papers, and one which may lead to some confusion of ideas, has occurred to me whilst listening to them being read, and that is that the same cyanide units have not been kept to throughout - in one paper we have units of sodium cyanide, and in the other units of potassium cyanide. The same remark applies to protective alkali units, caustic soda being given in one place and calcium oxide in another. If it is possible, I think this should be altered before publishing.

Mr. H. A. White *(Vice-President):* I certainly sympathise with Mr. Toombs over the appearance of confusion in these papers caused by referring results to cyanide of sodium in some cases and in others to cyanide of potassium. As far as my own paper is concerned, the testing of all works solutions is reported in terms of potassium salt to enable comparisons to be made with the figures derived in ordinary practice here. In the same way the figures for *consumption* of cyanide are given in terms of the sodium salt (taken as 129 KCN) as that is the salt generally used and the figures for which are most familiar in that connection.

It is more or less a coincidence that we have two papers covering very similar ground read at one meeting, and it must be made clear that they are not antagonistic to one another, nor are they intended to be complementary they are quite independent.

It is of great benefit to this Society to have Mr. Leslie's experiments and ideas so clearly put before us, and our thanks are due to him for so freely giving us the results of so much work and so much thought. He has established a *prima facie* case in favour of covering in our vats and working apparatus, and his work deserves, and I hope will receive, a thorough and critical examination. I intend to confine my attention principally to a discussion of the results and opinions expressed, leaving on one side the questions of cost and practicability of the necessary apparatus—his paper does not give sufficient detail on these important matters on which to base useful criticism.

Generally, the facts so far presented do not warrant the opinion that Mr. Leslie's process is bound to be successful, nor, on the other hand, that there is no chance for it. Beside the question of capital cost, the possibility of interference with extraction in the absence of free air surfaces with continued use of the same solutions, must also be considered. As will be remembered, in the early days of the process the solution sumps were frequently partly or entirely covered in, and the words "foul solutions" are frequently met with in our early Proceedings. At the present time all our solution sumps are fully exposed in the open air and sunshine and that ugly expression is heard no more. There is now no doubt that oxygen is an essential to the correct working of the cyanide process; an appreciation of that fact caused the covers to be removed from solution sumps, and the proposal made by Mr. Leslie has therefore a certain amount of reasonable prejudice to overcome.

The only decomposition of cyanide solutions considered by Mr. Leslie is the plain hydrolysis to hydrocyanic acid and alkaline hydrate, and he ignores the probably more important action of carbon dioxide and the known hydrolytic decompositions involving the production of ammonium formate or (with the aid of oxygen) ammonium carbonate.

The exposure tests made with pure synthetic solutions ignore the effect of the presence of zinc, as was pointed out by Virgoe (C.M.M.S., Vol IV., p. 633, etc.), so long ago as August, 1903. Deductions made from tests of this kind require careful scrutiny, and unless working conditions are as nearly approximated as possible, will lead to erroneous conclusions. The alkalinity found by Mr. Leslie to correspond with loss of HCN gas, and tacitly assumed to be alkaline hydrate requires some explanation, under the conditions, unless the error of using methyl-orange as the indicator was made.

This naturally leads to the curious opinion expressed that alkaline protection is in practice useless, an opinion adhered to throughout the paper up to the point where abnormal conditions are referred to in explanation of the failure to show saving of cyanide in the Modder B. agitators. The "abnormal condition" is the presence of

adequate protective alkali, which fully bears out the result of experiment with Model Brown vat referred to in my paper. The exposure tests at Geduld also show that little or no HCN gas is evolved from a working slime solution which has sufficient protective alkali. I will now detail the results of some experiments I have recently made with the idea of determining the relative importance of the gas losses in the form of HCN and in the form of ammonia. The research is not easy with limited time and improvised apparatus, and the results, while not conclusive, are certainly suggestive.

Experiment I.—Compressor air was passed through (a) 20 NaHO; (b) saturated lime water; then through (c) 2 litres of "slimes sump solution", followed by (d) 500 cc. weak H_2SO_4 about *N* 50, then (e) 500 cc. *N* 65 $AgNO_3$. The air was passed through for 18 hours at the rate of, roughly, 750 cc. per minute, and no trace of a precipitate was found in the silver nitrate.

Before experiment, solution tested:—Total KCN, 0·018; free KCN, 0·011; NaHO, 0·015.

After experiment:—Total KCN, 0·018; free KCN, 0·011; NaHO, 0·0160.

This shows the solution to be practically unchanged under these severe conditions when the CO_2 is kept out.

The 500 cc. weak acid showed a loss of acidity to methyl-orange equivalent to 0·006 gm. NH_3, or to the equivalent of 0·0011 KCN reckoned on the 2000 cc. sump solution.

Experiment II.—Air at the rate of 1 litre (approx.) per minute was passed through (a), (b) and (c) as in Experiment I. for 24 hours, finally passing through 500 cc. sodium picrate solution.

In this case a faint but certain darkening of the picrate proved that a slight amount of HCN gas had come off, but this was too small to affect the titration of the solution, which gave the same results as before the test.

Passing air for a further 16 hours, this time without first removing the CO_2, caused the complete disappearance of protective alkali and a loss of 0·001 in the free and total cyanide tests. In this case the air was passed through *N* 65 silver nitrate, after the 2000 cc. sump solutions and 500 cc. weak acid, and a slight but distinct precipitate of AgCN was shown. Air still further 24 hours under like conditions showed a further loss of 0·005 KCN, which yielded a heavy precipitate in the $AgNO_3$ solution.

The above two experiments show clearly that practically HCN evaporation from alkaline solutions is trifling, but that CO_2 can effect heavy losses in the absence of protective alkali.

Experiment III.—Compressed air at the rate of about 1 litre a minute was passed through (a) 250 cc. of 10 NaHO, (b) 2 litres of slimes sump solution (total KCN 0·015, NaHO 0·0152) then (c) 5 cc. of Nessler's solution in 500 cc. NH_3 free water. The solutions examined after passing the air for 18 hours showed a distinct yellow reaction in the Nessler test, and while no loss was shown in total cyanide, the alkalinity had diminished to 0·0068 (proving that some CO_2 got through the caustic soda solution of this strength and amount). Air was again passed through under the same conditions for a further 24 hours, and the solution had now become acid and showed a loss of 0·001 KCN. The Nessler solution had now become completely decolourized by the HCN evolved. This experiment would suggest that ammonia is much more readily evolved than HCN from alkaline solutions in actual works practice, but it remains a difficult problem to determine exactly what proportion of the gas losses (about 50 of total cyanide consumption) is due to ammonia and what to the other possible sources, namely, hydrocyanic acid and methylamine.

Two possibilities are suggested. First that enclosing the vats might reduce the ammonia hydrolyses by preventing the escape of ammonia, and, second, that these hydrolyses might be checked by the addition of salammoniac to working solutions. It is also correct to point out that Mr. Leslie's apparatus suggests the possibility of using the carbon dioxide of boiler flue gases to remove the HCN from residues before disposal to dam or dump and, incidentally, evade the use of cyanicides for sand-filling, beside saving cyanogen which is now thrown away.

Mr. C. Toombs *(Member of Council):* I should say that the presence of ammonia points to potassium or sodium cyanate having been formed. The cyanates of sodium and potassium are very unstable in solution, rapidly breaking up into the corresponding carbonates and ammonia. This may be a source of loss not taken into consideration by the authors, but put down to evolution of hydrocyanic acid through hydrolysis.

SAFETY MEASURES IN THE USE OF EXPLOSIVES.

By Wm. Cullen (Past-President)

(Printed in Journal, March, 1915.)

DISCUSSION.

Capt. Gerard W. Williams *(Corresponding Member of Council).* In the May *Journal*, p 297, Mr. J. D. Marquard makes the following remarks on "Overcharging":—"We use twice as much explosives here per ton of rock broken as in Australia, and our stoping conditions are somewhat easier; there is something to look into for an explanation."

This statement is only partially correct. Certainly the amount of explosive for a ton of ore broken in the leading mines of Australia is quite possibly only half the amount per ton of ore broken on the Rand, but the conditions of ore-breaking are entirely different. Almost without exception the mines producing the biggest tonnage in Australia are mines working with big stopes. Kalgoorlie stopes average about 40 feet wide, and the Broken Hill stopes extend up to 80 ft. or 90 ft. in width; the average stoping width would probably exceed 20 feet. The same remark applies, only on a larger scale, to Mount Morgan and Mount Lyell, and most of the copper-producing mines in Queensland will vary in stoping width from 40 to 50 feet. In these cases, it is therefore possible to put in very long "uppers" and side holes and, breaking either on the "flat back" or rill systems, very large tonnage can be broken down on a single round of firing. There is also a material difference in the resistance to explosives furnished by the banket reefs and the wide lodes in Australia.

I regret I have no data available at present of actual amounts of explosive used on the various mines, but I remember distinctly that on a small quartz lode, on which I was once working, and which in width and characteristics was not greatly different from Rand mines, my consumption of explosives per ton broken was approximately equal to figures shown in Rand mining statistics at that time.

I quite agree with Mr. Marquard that overcharging is quite as bad, and indeed worse, than undercharging, but I merely wish to draw attention to the dangerous generalisation with which he commences his remarks on "overcharging."

With reference to storage underground, in the same issue, p. 300, I cordially endorse Mr. R. A. Barry's remarks. I have known of at least one accident directly traceable to the use by prospectors of fuse that has been left for some time in an empty stope in the small mine that they were working in. On a large and well-run mine there should be no possible opportunity of stale fuse lying about in the fuse magazine, but on small outback mines fuse and primers have a habit of being left in stopes by the party working there, and some weeks or months later another party enters the stope and uses the stores left there. Of course, this is contrary to all regulation, but on small, scattered prospecting mines worked by small parties, this evil not infrequently occurs.

The President: I will now call on Mr. Waters to read Mr. Cullen's reply to the discussion.

Mr. Wm. Cullen *(Past-President):* Before giving my reply, I think that it would make it more clear and interesting if I give a general summary of the discussion.

SUMMARY OF POINTS MADE IN THE DISCUSSION.

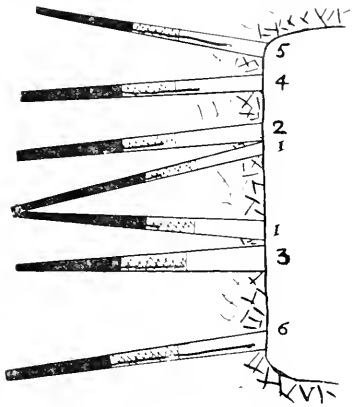
(1) Partial re-blasting of cut (Mr. Price).
(2) Saliva in detonators (Mr. Price).
(3) Placing the detonator well down the charge (Mr. Price).
(4) Crimping the fuse causing misfires (Mr. Weston).
(5) Electric blasting (Mr. Weston).
(6) Tearing off paper at ends of cartridge (Mr. Weston).
(7) Slitting of cartridges longitudinally (Mr. Weston).
(8) Frozen explosives (Mr. Weston).
(9) Use of cheap fuse of irregular burning speed (Mr. Weston).
(10) Deterioration of fuse on storage (Mr. McArthur Johnston).
(11) Rate of burning speed of fuse (Mr. Waters).
(12) Tying of fuse with detonator to primer cartridge (Prof. Cellier).
(13) Primer cartridge at bottom of charge with water tamping (Prof. Cellier).
(14) Nowag clip (Mr. Pam).
(15) No. 6 and No. 8 detonators (Mr. Pam).
(16) Weak detonators (Mr. Marquard).
(17) Damp detonators (Mr. Marquard).
(18) Sawdust in detonators (Mr. Marquard).
(19) Working ends of detonators pointing wrong way (Mr. Marquard).
(20) Stacking of cases of explosives (Mr. Marquard).
(21) Excessive coiling of fuse (Mr. Marquard).
(22) Copper sheathed tamping rods (Mr. Marquard).
(23) Quality of fuse (Mr. Barry).
(24) Natives firing holes (Mr. Barry).
(25) Cut-out shots (Mr. Meyer).
(26) Lag dyn. primers (Mr. Pam).

SUMMARY OF REMARKS MADE IN THE DISCUSSION.

Mr. W. S. V. Price.—Referring to the point made by Mr. Waters, namely (1) the partial reblasting of the cut soon after the first blast, Mr. Price and others stated that this practice, being now prohibited by law, cannot take place. Both he and others also thought that (2) misfires might be caused by saliva finding its way into the detonator tubes when the sawdust was being removed by blowing into them. He defended the practice of (3) placing the detonator well down the charge, and claimed good results from same.

Mr. E. M. Weston.—This gentleman mentioned that (4) crimping the fuse tight sometimes causes misfires, and he reaffirmed his original criticism with reference to (5) electric blasting. He asked the author's opinion about the advisability of (6) tearing off the paper at the ends of cartridges in order to obtain closer packing. He did not agree with Mr. Price's device for getting a cartridge to accommodate itself to an enlarged hole by twisting it so as to soften it, and he asked the author's opinion on his own practice of (7) slitting the cartridge longitudinally so that it collapsed and accommodated itself to the hole. Referring to the author's statement that frozen explosives were seldom met with on the Rand, Mr. Weston mentioned that (8) he had seen frozen explosives on several occasions when these (presumably surplus) had been brought up wet in sacks and stored in this way in the magazine. The author considers Mr. Weston's remarks on (9) the use of cheap fuse, etc., to be of such importance as to call for their reproduction in extenso:—

"I do not think the author, or Mr. Price, has dealt with what I consider the greatest cause of misfires on the Rand, and also, of course, because of that, one of the greatest causes of accidents. It is, I believe, due to a mistaken wish to economise, and this has led to the increased use of cheap fuse with an irregular burning rate, although Mr. Cullen says that the fuse on the Rand is generally good. Fuse is now supplied for development work which has a marked burning rate of 90 to 110 seconds per yard. A fuse like that is absolutely, in my opinion, unsuitable for development work on the Rand at all. Such fuse is, say, the 3d. to 3½d. fuse, whereas by paying another 1d. or so one can get a fuse with a regular burning rate. The reason for accidents is not far to seek. Turning to the rough sketch of a development end, we see that, taking the holes in pairs to be fired in rotation, even in soft ground with only one caser, there are six holes to be fired in rotation as numbered. Now, with the 90 to 110 seconds fuse, assuming we find



Rough Sketch of a Development End.

a very much smaller difference of burning rate between different coils and packets, we require, in order to make sure that the holes will go off in proper rotation (if they do not this will cause great loss and probably also misfires and dangerous sockets) we must cut our fuse in differences of length varying from, say, 4 in. to 6 in. This means that at the time the cut goes off, No. 6 hole has unburnt fuse in from 2 ft. to 3 ft. long. If there are more holes and casers, the condition is worse. I have tried to show what this means by drawing the fuse in the sketch. It is this *protruding unburnt* fuse that gets cut off and is the cause of more misfires on the Rand than anything else. Were the miner given fuse he could depend on when cut, say, with only ½ in. difference of length, misfires would nearly disappear.

"Now, we understand why some miners put their primers at the bottom of the hole, a practice which the author quite rightly condemns. Mr. Price tells us he puts his primer under two sticks. This is bad, because the detonator should point towards all the charge, not away from any of it. He does it because he is given poor fuse and

wedges to prevent fuse being cut off. The wedges occupy about 2½ ft. of the length of the hole, and, as he admits, it is quite unusual to get 3 ft. of the mouth of a hole cut off by the explosion of other holes, so that if the fuse is burnt well down to the detonator there is no need to do this. If, however, there is 12 in. or 18 in. unburnt fuse sticking out, it is cut off if only 1 ft. or 2 ft. (which is common) of the hole is blown away."

Mr. A. McARTHUR JOHNSTON.—This gentleman emphasized the fact that (10) fuse deteriorated on storage.

Mr. W. WATERS.—This speaker referred to (11) the rate of burning speed of fuse, and stated that the question of cost does not necessarily control the regularity of the burning speed. He mentioned that he had proved by personal tests that the highest priced fuses were not, as a rule, more regular than fuses costing half their price.

Prof. J. CHILLER.—This gentleman, while agreeing with the author that Clause 106, sub-section 19, of the Mining Regulations was consistently broken, did not apparently think that there is any great necessity to observe it, namely (12) to tie the fuse with detonator to the primer cartridge. He stated that when sinking the Witwatersrand G.M. Co.'s vertical shaft the (13) primer was always put at the bottom of the hole with water tamping.

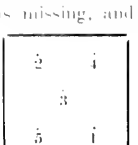
Mr. E. PAM.—This gentleman stated that (14) the Nowag Clip would fulfil the purpose for which it is intended as soon as it can be obtained. With reference to (15) No. 6 and No. 8 detonators, he said that when on a previous occasion he asked the author's advice, he, the author, stated that with Modderfontein explosives, No. 6 detonators would probably do well enough, but where various makes of explosives were used on the same mine, it would be safer to use No. 8 detonators.

Mr. J. D. MARQUARD.—This gentleman mentioned 35 different causes of misfire, most of which were covered by the author and by Mining Regulations. Among the causes which he specified, he dealt with 16 weak detonators, 17 dampdetonators, 18 sawdust in detonators, 19 working ends of detonators pointing the wrong way. Then when referring to the question of oxidation, he emphasized the necessity of (20) proper stacking of cases of explosives, which is a most important point. When dealing with fuse, he stated that short cementing could be caused by (21) excessive cooling of fuse, but I could not follow his argument under the two headings of "*Explosives burning and charges being smothered*," and "*Over-charging*". I shall have something to say in connection with his criticism of my recommendation to do away with (22) copper-sheathed tamping rods.

Mr. R. A. BARRY.—I consider Mr. Barry's contribution to be excellent, but I think that most of his points were made by other speakers. I propose, however, to deal with his remarks on the regularity of burning speed, which he gives under the sub-heading (23) "*Quality of fuse*". He also raises another very important issue under the heading (24) "*Natives firing holes*".

Mr. E. C. J. MEYER.—In the author's opinion, Mr. Meyer's remarks on (25) "*Cut out shots*" are of extraordinary interest, and he considers that the following paragraph should receive the widest publicity:

"*Cut out shots.*—I have had the fortunate experience of discovering that not only are primers cut out by the breaking away of the collar of the hole by other shots, but also by the concussion caused by the explosion of adjacent holes. A bench with five holes of 6 ft. length each was charged up in the usual way, but only No. 1 hole was ignited, and 30 minutes after the explosion the bench was again examined. To my great surprise I found that the primer of No. 3 hole was missing, and on searching



FRONT VIEW OF BENCH.

the footwall of the stope, I found the primer with the detonator and fuse intact, the barrel of the hole not having been damaged at all. This proves that a certain amount of compression in the rock had taken place, which was sufficient to force the primer and tamping out of the hole, much in the same way as paste is squeezed out of a tube."

Mr. E. PAM.—This gentleman, in his second contribution, discussed the influence of 26 ligdyn primers, on the detonating wave, and stated that so far nothing very definite had been achieved.

REPLY TO DISCUSSION.

I consider that the response to the paper has been interesting, instructive, and intensely practical, and I can only trust that it will be the means of still further reducing the accident rate. I was glad to notice that several contributors spoke quite hopefully on this point. The discussion naturally groups itself around the various issues which I raised, and it will simplify matters if I deal with it in a general rather than in a detailed manner.

Fuse was referred to by Messrs. Weston, McArthur Johnston, Waters, Marquard and Barry, who spoke of quality, price, rate of burning, crimping, and deterioration on storage. I did not say much on the question of deterioration, as fuse is seldom stored in this country for any length of time, but that the coating does deteriorate through prolonged storage is well known, and this is irrespective of whether the cases are tin-lined or not. Of course, the worst kind of storage is, unfortunately, that which is most common, viz., a galvanized iron building where the extremes of temperature must of necessity be very great. If care is taken to replenish stocks frequently, and to always use up the oldest fuse first, the grades of fuse at present on the market should really suffer no real deterioration. Since certain brands have been eliminated as a result of the war, the Witwatersrand is really very well catered for, as all the fuse now hails from Great Britain, where the climatic conditions are almost ideal for manufacture, other things being equal. Fuse made in Great Britain should always be the best.

Reference has been made to the so-called cheap fuse in connection with the rate of burning. Now all fuse, whether cheap or dear, has to pass the Government test, and though the margin of speed variation is wide, I cannot hold out any hope that it will ever be materially contracted. It will suffice to state that the difference between the two extremes could be accounted for alone by climatic changes. Fuse made on a hot, dry day would be "quicker" than fuse made on a cold, damp day. In Great Britain, as I have already said, the climatic conditions are almost ideal. With regard to regularity, therefore, a so-called cheap fuse can be quite as good as an expensive one, but in regard to other qualities, the dearer fuse must always be the best. For instance, the coating will deteriorate less rapidly, and when practical men realise the influence this has on short-circuiting and side-spitting—within the limits of commonsense greater care will be taken in selecting. I should, for instance, never recommend gutta fuse for ordinary mining operations any more than I would recommend our ordinary Rand brands for submarine work. It was suggested that I might have said more on this point in my paper, because the rate of burning has naturally a great influence on mining results and on safety, but the only practical suggestion which I could make, and which I reiterate, is to allow for a long time interval between the shots. In other words, do not attempt to economise by cutting down the time interval. This will not obviate cut-out shots, as both Messrs. Meyer and Weston show, but it will minimise them. It is because of this danger that I still advocate electric blasting by means of delay action fuses, and nothing which has transpired since the paper on that subject was written, has materially altered my views. The use of these delay action fuses is the only practicable way of reducing "cut-outs" to a minimum, though, as Mr. Meyer shows very clearly, they can never be entirely eliminated.

Then with regard to crimping causing the fuse to stop burning, I had, of course, in my mind average fuse and decent pliers. If miners will use bad pliers, it is difficult to prevent them, but although I have not tried the experiment, I think that it would be found very difficult indeed to stop any of the present-day brands from burning. With proper pliers it is quite impossible.

Reference has been made to the careless manner in which fuse is stored in the men's boxes underground, but Mr. Barry strikes the nail on the head when he says that a better class of intelligence is wanted. It is impossible to guard against everything, and if manufacturers only knew what their fuse had to stand before fulfilling its normal and proper function, they would hold up their hands in horror. The making of fuse is a most delicate operation, and extraordinary care is exercised at every stage to detect defects. The excessive coiling of fuse is to be deprecated, as Mr. Marquard points out, but until our whole system of mining is altered, coiling must be done. This, even for the best fuse, is a very severe test, and fuse which has been once coiled should never be uncoiled again before being used. Pliability, a property which cannot be defined, is one of the most important things

about a fuse, and yet the pliability must be of a certain kind, for all fuse is pliable.

Let me again advise all users of fuse to go a little bit beyond the Government requirements, and test their fuse as they use it in practice. There is nothing more difficult to describe by specification than the quality of fuse and firing sticks.

Detonators were referred to by Messrs. Price, Pam, and Marquard. Let me say at once that I question very much whether a little saliva would do any harm. One can take a detonator and after removing the sawdust, dip it into water for a minute or two, take it out, and after draining the water off, explode it by fuse as if nothing had happened. It is not a practice which I would recommend any more than I would the addition of a little saliva to the detonator before use. If there is, however, too much water present in any form, this will extinguish the spit of the fuse, which is the real danger. This leads naturally to the influence of a few grains of sawdust either damp or dry. If anyone cares to try the experiment of putting a few grains of sawdust on top of the fulminate of a cleaned out detonator, he will be surprised to find how little it really takes to render the spit of the fuse ineffective. I strongly recommend everyone to try this perfectly safe experiment, using, of course, common-sense precautions. Then with regard to the relative value of No. 6 and No. 8 detonators, I have a strong personal predilection for the larger size for most of the mines of the Rand. I shall not, however, attempt to dogmatize about those mines which use only one brand of explosive. Undoubtedly, No. 8 detonators do increase the factor of safety, as the stronger impulse helps to minimize the influence of bad charging, air gaps, insensitive explosive, and other less important factors. Nowadays, no one uses smaller detonators than No. 6 with the class of gelatinous explosives found on the Rand, but in America, where gelatine is have quite a different meaning, much smaller detonators are used. Within the past year I found a railway contractor using No. 3 detonators for blasting gelatine in Cape Colony with apparently good results, but on my advice he stopped at once. There were several references to the "working ends" of detonators, and certain experiments were quoted, but I must confess to a great distrust of experiments which are carried out under highly artificial conditions. The truth is that we know very little about what takes place in a drill-hole, but a very safe principle to go on is to eliminate everything which would tend to reduce the safety factor. For this reason, if for no other, it is very wrong to put the detonator anywhere else than at the top end of the primer cartridge, the working end pointing directly downwards to all the other cartridges. In this position, it exercises its greatest effect, as is proved by all experiments conducted in the open. Nothing which either Mr. Price or Prof. Cellier said regarding the placing of the primer at any other position than the top of the charge has altered my views in the least. Indeed, I go further to assert that the adoption of the method of placing the primer cartridge with detonator in any other place than at the top of the charge in all probability will reduce the efficiency of mining results as measured by fathomage per unit of explosive consumed. To a limited extent it might prevent "cut off" shots, but after Mr. Meyer's very practical contribution I doubt this very much indeed. The risks which are introduced by this practice far outweigh, in my opinion, any possible advantages, though when water tamping is used, as in the case cited by Prof. Cellier, I can see no risk.

In the discussion, very little was said on the subject of exudation, which, in my opinion, is a matter of the greatest importance. There has been far too much exudation in the past, and I believe this explains many obscure accidents. To a great extent the mines can help. No manufacturer ever wilfully sends out explosives which exude nitro-glycerine, though this may be done inadvertently, because it is not easy to prevent it altogether. The mines can help in various ways which I shall indicate. As in the case of fuse, it is of paramount importance to use the oldest explosive first. Continued storage under Transvaal conditions means exudation with the very best explosives. A well ventilated magazine is much better than a badly ventilated one, and the proper stacking of the cases, which was emphasized by Mr. Marquard, is of the greatest importance—particularly with the larger sizes of cartridges. So far as possible, the cases should be so stacked that the longer axis of the cartridge shall be in a vertical position. Under ordinary conditions, with fresh stocks arriving into the magazines every two or three weeks, the first point, viz., the using of the oldest explosive first, is by far the most important. If I were sure about the entire absence of

exudation, I would fall in with Mr. Marquard's views about copper-sheathed tamping rods: until this takes place, I must oppose their use. Even under normal conditions, I have never been very enthusiastic about copper-sheathed tamping rods. Even Mr. Weston's apparently harmless device of slitting the cartridges, indicates the possibility of explosives getting smeared along the hole, and a thin film of explosive plus grit, plus copper or brass, could easily cause an explosion. There is really no difficulty in keeping a wooden tamping rod in order. It is sheer nonsense to talk otherwise.

Mr. Weston asks my opinion on various points. First, with regard to the slitting of cartridges in order to make them collapse to fit better into the hole or an enlarged section of it, I think it is questionable whether this is permitted by law. If I were an inspector, I would give my ruling against it on legal grounds, but apart from that, I unhesitatingly condemn this practice, as well as all other practices, which expose explosives to grit. The disadvantages and risks far outweigh any possible advantages. *Ipso facto*, I also condemn the practice of tearing off the wrapper at the ends of cartridges. There is no necessity for this, if other precautions are observed, while at the same time it is illegal.

I can well believe that Mr. Weston would find frozen gelatine under the conditions he describes, and it is a fact that gelatines under such conditions freeze very readily.

If I understand Prof. Collier aright, he does not approve of the tying of the fuse and detonator to the primer cartridge, and he stated that the present regulation is not necessary under to-day's conditions. In my paper I have stated the exact opposite, and I know that I am right, and, as a matter of fact, so do most people, including the entire inspectorate, or the regulation would not be where it is. It is just one of those things which by common consent has not been observed, with consequent lamentable results. It is not a new point. I, personally, have been preaching it for years, and both Mr. Waters and myself, as well as all the technical men at the Modderfontein Dynamite Factory, have always observed the regulation. Our record speaks for itself. Natives can do the tying. There is nothing in the mining regulations to prevent them doing so, if under white supervision, and the little extra trouble and expense would be more than offset by the prevention of one accident.

The Nowag clip, to which Mr. Pam referred, is not a perfect device, and I, personally, was very much surprised at its winning encomiums at the Mining Exhibition, but it is something to work on, and I strongly advise the Safety First Committee to see what can be done to improve it. The idea is public property, and anyone can make or sell the clip without fear of "reprisals".

Mr. Waters' point about the reblasting of cuts has been referred to by Mr. Price and Mr. Pam, who point out that the practice is now illegal. That is true, but is the regulation observed? I, personally, do not know, but I have my doubts.

The last point of all, which really has nothing to do with the paper itself, is the firing of holes by natives. Mr. Barry condemns it, and he knows much more about it than I do, but is his wholesale condemnation really justified? It is no secret that natives do practically all the firing on many mines, and I think that an intelligent native is quite as competent as many white men who hold blasting certificates. I have a very high opinion of the native's competence, when he has been carefully taught, and when he can make explosives as he does, I cannot see that he should not do the firing. This, however, raises a *quasi* political point which is probably wisest not to press.

NOTES ON THE TREATMENT OF ANTIMONIAL GOLD ORES FROM THE MURCHISON RANGE.

By H. R. Adam, B.Sc. (Member).

(Printed in Journal, June, 1915.)

DISCUSSION.

Mr. E. M. Weston *(Member)*: The author has chosen a very interesting subject for his paper, and I suggest that the Secretary should try and procure us a description of the MacArthur-Forrest process as installed at the United Jack Mine, Selati, and should also write to the Costerfield G.M. Co., Victoria, and Globe and Phoenix, Rhodesia, for a description of the methods in use there.

The author seems to me to have omitted to furnish some necessary details. Is the gold fine or coarse? Is the gold which is associated with the antimony sulphide *free*,

and can it be amalgamated? What I mean is, if a sample of the ore be finely ground, can the gold be panned out of it, and can it then be amalgamated when quite clean? Or, as in Hillgrove, New South Wales, does some of the gold seem to exist as sulphide or covered with a film of sulphide, so as to be not capable of being amalgamated. The author also gives us no idea whether the ore is payable as a gold ore *per se* or not. I mean that if no antimony were present, or if it could be easily eliminated, would the mine pay as a gold mine? If not, how much of the antimony is it necessary to recover apart from the gold to make the mine pay? All these would require to be known before investigating the subject. The author has apparently overlooked an article on cyaniding antimony ores in Hillgrove, New South Wales, by W. A. Longbottom, which appeared first in the Australian *Mining Review* and was copied in the *Mining and Scientific Press*, June 29th, 1912. This paper omits important details, such as percentage of extraction secured, but it seems to show that the cyaniding of antimonial tailing is practicable at a reasonable cost.

The tailing, much of which was semi-oxidized and acid, contained pyrite with some copper pyrite and stibnite after concentration. It was found that with an oxidized tailing a lower extraction was gained, but consumption of cyanide was less. The treatment took seven days, including filling and emptying. Acid tailing was neutralized to faint alkalinity by lime, great care being taken to avoid any excess of alkalinity that would bring antimony sulphide into solution. Some antimony was, however, dissolved and lead acetate was used in the boxes, while permanganate was used periodically for cleaning foul solutions. The zinc slime carried 1½ oz. to 2 oz. gold per lb. and was fused with excess of lead oxide with iron bars, soda, borax, silica, and flour. The lead was cupelled. Costs were: labour 2s., cyanide 8d., zinc 1½d., fuel 2d., sundries 1½d., royalty 1s., total 4s. 1d. In considering the treatment of gold bearing antimony ores we will suppose first that antimony is at its normal price of about £10 per ton, so that a concentrate of 50 must bear all freight and treatment charges on £20. If it carries with it several dwt. per ton of gold which is not paid for, it is rather a nuisance than anything else. Apparently the Globe and Phœnix does not attempt to save antimony. It is found there also that exposing the tailing and slime to oxidation for some time assists extraction. Let us also suppose that the ore carries enough free gold to pay for that metal alone, and that we wish to recover first as much gold as we can, and, secondly, to recover as much antimony in the form of concentrate as we can economically. To do this successfully we must rid our minds once for all of the idea that seems to obsess so many people who should know better, that gold can be recovered, and should in all cases only be recovered by amalgamation and cyaniding. When they find gold associated with some mineral like antimony sulphide that interferes with this process, they blame Providence for inventing refractory ores. Tin ore and galena are brittle minerals with a specific gravity of about 7, yet in properly devised plants a saving by concentration may attain to 90 . Gold is malleable and has a specific gravity of about 19, but it is apparently bad form to ever rely on concentration. Let me make two heretical statements. On most gold-bearing ores as high an extraction can be obtained by concentration as by amalgamation. Very often, even where inside amalgamation has been practised, with proper concentration appliances it is possible to recover free slime gold and floured amalgam from the tailing. The proof of these statements lies on the theoretical side, in the fact that it is obvious that any particle of gold amalgamated on the plates, and to a large extent in the mortar, must have mass enough to settle in a moderate current of water. If it has such mass it can be retained and separated on various concentrating devices. On the practical side it has been repeatedly done. In Victorian gold mines a simple cast iron table with suitable holes and riffles has been repeatedly used with success. Where inside amalgamation is practised I have repeatedly seen properly arranged canvas strakes on the Australian system recover unamalgamated gold so fine that it remained long in suspension if shaken up in water. If a small test were made on these lines on Rand ore I think quite a surprising extraction could be obtained. I would suggest that if a plate with ¼ in. staggered holes (somewhat conical at the top), and with an arrangement for regulating an upward current in each row, so that only iron particles (such as magnetings) would settle in them, were put in instead of an

amalgamated plate directly below some battery in which no mercury had been introduced, it would be found that an equal saving of gold in the form of a gold-iron concentrate (easily amalgamated later) was being made as with the ordinary amalgamated plate. Because amalgamation costs only about 1d. per ton is no reason why a cheaper process should not be attempted. This would also be done at a lower cost and with great safety against theft. To return to our subject. The treatment plant I would suggest for such a mine as the Weigal would be: Crushing in breakers to 2 in. and sorting out at crusher and on belt after crusher, and also underground, any coarse antimony ore. Second crushing in mill through a suitable mesh to free all gold, the pulp to pass directly to Wilfley table. This would make first concentrate streak of nearly pure gold (containing most of the gold in the ore); 2nd middlings of some fine gold and antimony (also containing some gold embedded in it). This would be re-ground in pan and redressed over a Luhrig vanner very carefully to give a 1st concentrate of nearly pure gold and 2nd tailing to be sold as antimony and gold concentrate. The tailing containing slime gold and slime antimony would be passed over canvas strakes and the concentrate also re-dressed over a Luhrig vanner. The remaining tailing might or might not then pay for cyaniding. There is no doubt, however, that flotation methods offer a most promising line of inquiry, and had the author been able to complete his experiments with oil and agitation, a feasible commercial method might have been evolved.

RECENT INVESTIGATIONS ON DUST IN MINE AIR AND THE CAUSATION OF MINERS' PHTHISIS.

By James Moir, M.A., D.Sc., F.R.S.S.Afr.

(Printed in Journal, July-August, 1915.)

DISCUSSION.

Mr. John Watson *(Member of Council)*: I should like to say, as one who did a good deal of microscopic work at one time, that this paper of Dr. Moir's appeals to me as being a very valuable one and one which would grace the Proceedings of any scientific society in the world. We know the great havoc that miners' phthisis has worked among the underground workers. I would just like to move a hearty vote of thanks to Dr. Moir for his paper, which, I think, will be an epoch-making paper on this subject of miners' phthisis. I have been a member of this Society since 1896, and I do not remember a paper which has more careful microscopic and micro-photographic work than this one of Dr. Moir's.

The vote of thanks was carried by acclamation.

Mr. M. Weinbren *(Member)*: I should like to know if, when one of the finest particles, say a three-micron particle, is thoroughly wetted, it will still be light enough to float, or will it be heavy enough to sink within a short time?

The President: Dr. Moir will no doubt answer that in his reply.

THE USE OF SCOOP DISCHARGES IN TUBE-MILLS.

By W. R. Dowling (Past-President).

(Printed in Journal, March, 1915.)

DISCUSSION.

Mr. K. L. Graham *(Member of Council)*: Mr. Dowling's paper on the above subject synchronizes with a world-wide enquiry amongst tube-mill operators for information as to the merits of the principle involved, namely, peripheral discharge, and will be the more generally welcomed on that account.

Personally, it gives me great pleasure to join in the discussion, although rather late in the day, as experimental work on this subject has occupied the attention of the Rand Mines Metallurgical Staff for the last two or three years, and a considerable amount of useful data has been collected, although I take the opportunity of remarking that I consider this still far from complete. The results in the main entirely corroborate those published by Mr. Dowling, a gain in crushing efficiency, or production of 90, of 20 to 25 , having been shown to be attainable in our more modern plants.

The history of the introduction of scoop elevators on these fields, as told by Mr. Dowling, applies, I think, to all mines originally supplied with "Davidsen" tubes

mills, a device of the kind described having been adopted, as far as the Rand Mines Group is concerned, in 1906.

My attention, however, was first drawn to the increased efficiency due to their use by Dr. Caldecott in the beginning of 1912. Shortly after this Mr. F. L. Bosqui initiated a series of trials at the City Deep, using a scoop with an effective diameter of approximately 58 in., designed and patented by Mr. W. Calder, of the Village Main Reef G.M. Co., Ltd. Latterly, improved modifications of the original type, depicted by Mr. Dowling, have been found equally effective and less costly, and for this reason are in use on some of our mines. The object of the trials in the first instance was to determine the effect of a pulp-lifting device on the crushing efficiency and power consumption of our standard tube-mills. In a later series the diameter of the scoop was varied, the minimum effective diameter experimented with being 21 in.

Before turning to the tabulated figures, it would be well to explain in a few words the conditions under which the trials were run. The greatest care was taken to prevent any possibility of errors in the power readings, which were taken with an indicating wattmeter, the total units consumed being read from an integrating wattmeter. The tube-mill employed in the experimental work was kept specially for this purpose, being brought into circuit for the period of the run only, usually eight hours, a unit of those regularly employed being shut down to compensate, and under these conditions could be examined as often as necessary for load and wear. No samples or readings were taken until normal running conditions were obtained, the work being in sole charge of a member of the departmental staff, in this case Mr. H. S. Ball, whose previous experience on similar work specially fitted him to act in this capacity. In this connection, while in general agreement with Mr. Dowling, in his remarks relative to the necessity of taking average figures only, I should like to point out that this becomes very much more necessary where data has to be collected under routine conditions from units forming an integral part of the plant in regular use. In the trials under review, runs into which abnormal conditions were known to have entered, were discarded, long experience and close observation enabling this to be done with accuracy; the result being that the runs tabulated could be considered as absolutely representative.

In regard to the grading of the pulp feed, this can be considered typical of the practice on all our more modern mills, and in this connection Mr. White's remarks, contributed to the discussion on the paper (*Journal*, April, 1915, p. 235) are of considerable interest, as the higher mechanical efficiency, shown by the City Deep trials, is no doubt due to the lower percentage of 90 in the pulp fed. Owing to local conditions existing at the time of the trials, it was not possible to experiment with feeds of over 100 tons, but the results do not suggest any likelihood of improvement in either crushing or mechanical efficiency over this figure.

By an examination of the respective tabulations interesting comparisons can be made.

In the average of runs Nos. 4 and 5, Simmer and Jack, with a load of 14·46 tons, 118·8 H.P. is consumed, as against 125·3 H.P. in runs Nos. 8, 9 and 10, City Deep, using 10·6 tons of pebbles and producing 11 tons less −90; the mechanical efficiency showing considerably higher in the case of the latter (last column, Table "D"), without making allowance for the extra diameter in the Simmer and Jack mill, the power consumption, apparently, not being affected by variation of pebble load. (See Robinson Deep, Runs 9-15, Cols. 9 and 10.) Close examination of these figures reveals the lower moisture and coarser pulp feed as the differing factors. An analyses of run No. 8, Table 2, Simmer and Jack, and runs Nos. 11 to 14, Table "E", City Deep, show that with the same pebble load, 125 H.P. was required in the case of the former as against 107 H.P. for the latter, the same production of −90 resulting, the principal point of difference apparently being the lower moisture in the City Deep feed pulp.

Comparing runs Nos. 14 and 15, Robinson Deep, with Nos. 8, 9 and 10, City Deep, the latter show a lower power consumption, with about the same production of −90. The factors at variance being the load, feed, moisture, and −90 in tube-mill inlet.

That the crushing efficiency increases with the effective diameter of the lift is indicated by a comparison of runs Nos. 9 to 15, Robinson Deep, with Nos. 2 to 5, City Deep, the power consumption being the same, but the production of −90 considerably more in the case of the latter.

Runs Nos. 1 and 5, City Deep, are particularly interesting, as showing that, within the limits of the percentages tabulated, low moisture does not adversely affect

TABLE "A." TUBE MILL EXPERIMENTS WITH SCOOP DISCHARGE AT THE CITY DEEP.

22 ft. × 5 ft. 6 in. Tube-mill—fitted with ordinary Trunnion Discharge. Pebble load to centre.

Reference No.	Duration of Trial. Hours.	Actual Diameter of Tube Mill in Inches	R.P.M. of Tube Mill.	Current Cost per 24 Hours.	Tonnage Fed per 24 Hours.	Percent Moisture in Feed Pulp.	Height of Pebble Load in Mill.	Tons of Pebble in Mill.	Mesh.	Average Brake H.P.	Grading Analysis. Feed. +60	Feed. +90	Feed. −90	Grading Analysis. Discharge. +60	Discharge. +90	Discharge. −90	Tons −90 Produced per 24 Hours.	Pebble Consumption per 24 Hours. tons	Tons −90 Produced per 24 Hours.
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)		(12)			(13)		(14)	(15)	(16)
1	8	60	32	£5 2 7	300	32	Normal load to centre	10·6	64/100	115	72·1	17·6	10·3	23·3	25·6	51·1	122·4	7	1·46

TABLE "B." 22 ft. × 5 ft. 6 in. Tube-mill—fitted with 58 in. Scoop Discharge. Pebble load to centre 10·6 tons.

Reference No.	Duration of Trial. Hours.	Diameter of Scoop Discharge in Ins.	Actual Diameter of Tube Mill in Inches	R.P.M. of Tube Mill.	Current Cost per 24 Hours.	Tonnage Fed per 24 Hours	Percent Moisture in Feed Pulp	Height of Pebble Load in Mill.	Tons of Pebble in Mill.	Mesh.	Average Brake H.P.	Grading Analysis. Feed. +60	Feed. +90	Feed. −90	Grading Analysis. Discharge. +60	Discharge. +90	Discharge. −90	Tons −90 Produced per H.P.	Pebble Consumption per 24 Hours. tons	Tons −90 Produced per H.P.
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)		(13)			(14)		(15)	(16)	(17)
2	8	58	60	32	£6 16 3	320	37·5	centre	10·6	64/100	148	83·2	9·4	7·4	19·6	22·4	58·0	161·9	30	1·09
3	8	58	60	32	6 11 9	335	31·0	,,	10·6	,,	148	72·6	18·0	9·4	20·0	24·0	56·0	156·1	30	1·05
4	8	58	60	32	6 11 3	346	30·0	,,	10·6	,,	148	80·6	12·2	7·2	21·8	25·0	53·2	159·1	30	1·07
5	8	58	60	32	6 11 3	400	31·0	,,	10·6	,,	148	80·6	12·2	7·2	26·6	24·8	48·6	165·6	30	1·12
Average:						350	32·3				148							160·7	30	1·08

TABLE "C." 22 ft. × 5 ft. 6 in. Tube-mill—fitted with 58 in. Scoop Discharge. Pebble load 7in. below centre 8·5 tons.

6	8	58	60	32	—	270	33·5	8 in. under	8·2	64/100	130	88·0	7·6	4·4	14·6	23·4	62·0	155·0	30	1·19
7	8	58	60	32	5 18 4	290	32·0	centre	8·2	,,	136	77·4	14·0	8·6	13·8	25·0	61·2	152·5	30	1·12
						280					133							153·7		1·15

TABLE "D." 22 ft. × 5 ft. 6 in. Tube-mill—fitted with 21 in. Scoop Discharge. Pebble load to centre 10·6 tons.

8	8	24	60	32	—	295	32·0	centre	10·6	64/100	125	72·0	18·0	10·0	14·4	27·4	58·2	142·2	10/12	1·14
9	8	24	60	32	—	320	32·0	,,	10·6	,,	125	73·0	16·8	10·2	16·6	27·4	56·0	148·9	,,	1·19
10	8	24	60	32	—	350	32·0	,,	10·6	,,	126	78·8	14·4	6·8	24·8	26·8	48·4	145·6	,,	1·16
						322	32·0				125							145·6		1·16

TABLE "E." 22 ft. × 5 ft. 6 in. Tube-mill—fitted with 21 in. Scoop Discharge. Pebble load 7 in. below centre 8·5 tons.

11	8	24	60	32	—	285	32/33	8 in.	8·2	61/100	102	70·2	20·0	9·8	18·2	27·6	56·2	132·2	10/12	1·30
12	8	24	60	32	—	320	32	under	8·2	,,	110	75·0	15·8	9·2	26·2	26·6	47·2	121·6	,,	1·11
13	8	24	60	32	—	360	32	centre	8·2	,,	108	79·6	14·2	6·2	29·4	29·4	41·2	126·0	,,	1·17
14	8	24	60	32	—	400	32/33	,,	8·2	,,	109	79·6	14·2	6·2	33·8	27·4	38·8	128·8	,,	1·18
						341					107							127·1		1·19

crushing efficiency, as when working under these conditions the figures for −90 production reached a maximum.

The considerable divergence observable in some of the results seems to justify my opening remark, that finality has not been reached in these investigations; in any case the probable results of the application of this principle would have to be worked out over the whole range of stamp-tube-mill combinations in use.

The meeting then terminated.

Review.

TABLES AND DATA FOR RAND REDUCTION WORKS.
Arranged by C. O. Schmitt.

This extremely useful compilation renders easily available a mass of data only to be obtained otherwise by the expenditure of considerable time and trouble in searching through books of reference.

The bulk of the tables are from publications of Dr. Caldecott or Mr. H. A. White, and of course bear directly on the everyday routine of the reduction plant.

It is issued in two forms, one a sheet suitable for posting up on an office wall and the other a smaller sheet for desk use: so that the desired information is obtainable immediately, and that this is of a kind very frequently needed will be apparent from the following index:—

(1) Relative stamp duties; (2) Useful equivalents; (3) Mill pulp, quantity of; (4) Slime pulp table; (5) Decantation slime treatment formulæ; (6) International Atomic Weights; (7) Capacity of circular vats; (8) Assay ton weight system; (8a) Capacity Butter slime filters; (8b) Capacity rotary sand filter tables; (8c) Speeds and capacities of tube-mills; (9) Launder grades; (10) Capacity of centrifugal pumps; (11) Weight conversion tables; (12) Slime pulp formulæ; (13) Bisulphate of soda solution; (14) Tube-mill pebble loads; (15) Sand pulp table; (16) Imp. standard wire gauge; (17) I.M.M. Standard laboratory screens; (18) Battery screening formulæ; (19) Capacity of diaphragm cone classifiers; (20) N.C.F. system; (21) Tube-mill feed formula; (22) Flow solution through extractors; (23) Capacity of sand filter vats; (24) Standard solutions for testing; (25) Formula for cyanide solution; (26) Thermometer conversions; (27) General notes; (28) South African measures; Specific gravities of solids; Solubilities of salts in hot and cold water.

Though it has obviously not been easy to decide just what to include and what to omit, it would be difficult to suggest any notable addition or omission, having regard to reasonable size and legibility. Mr. Schmitt appears to have made a very well-balanced and neatly arranged selection, which should be of considerable value to all reduction works.

The price is 12s. 6d. for the desk sheet, while a few copies only are available in a smaller size which may be folded for pocket-book use, price 5s. each. They can be obtained from Mr. Fred Rowland, P.O. Box 1183, and it may be mentioned that half the profits are to be devoted to the objects of the S.A. Red Cross Society. G. H. S.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

FERROCYANIDE DETERMINATION OF ZINC.—"The method of titration in alkaline solution is the method recommended for use at Broken Hill for the estimation of zinc by titration with potassium ferrocyanide. The first modification for use with acid solutions was formally rejected.

Titration in Acid Solution.—Weigh out 0·5 gm. to 5 gm. of ore in an 8 oz. beaker and dissolve in 10 cc. concentrated nitric acid. When all red fumes have ceased add a few crystals of potassium chlorate and evaporate to hard dryness. [This does not seem superior to the common Western procedure of treating in a casserole with a saturated solution of potassium chlorate in strong nitric acid.] Allow to cool, and then add 50 cc. of hot water, 15 cc. of ammonium hydrate (sp. gr. 0·88) and 7 gm. of ammonium chloride, and boil on a sand bath for at least four minutes.

Filter and wash thoroughly with a 1% NH_4Cl solution until the filtrate measures about 300 cc. Add 2 gm. sodium thiosulphate and one drop of methyl-orange solution. From a burette then add concentrated hydrochloric acid until just acid, then 5 cc. in excess. Heat to 80° C. and titrate with potassium ferrocyanide, using a saturated solution of uranium acetate as an outside indicator. The end point is reached when one drop of solution and one drop of the acetate indicator give a faint chocolate colour on a white spot plate.

Standardization of the $K_4Fe(Cy)_6$ Solution.—Weigh out 0·3 gm. of c.p. zinc oxide, previously ignited to redness and cooled in a desiccator, or 0·25 gm. c.p. metallic zinc, and dissolve in 10 cc. of hydrochloric acid. Dilute to about 200 cc. with hot water, then add 15 cc. ammonia water (sp. gr. 0·88) and 7 gm. of ammonium chloride and proceed as given above for the assay.

Titration in Alkaline Solution.—The potassium-ferrocyanide solution contains 32·3 gm. of the pure salt per litre (1 cc. = 0·01 gm. zinc). A ferric-chloride solution containing in each litre 200 gm. of cream of tartar and 2 cc. of a neutral ferric-chloride solution carrying 10% iron is also used. This emulsion must be well shaken before using and 30 cc. used for each assay, as specified below. The outside indicator is pure glacial (99% to 100%) acetic acid

To standardize the solution, weigh out two lots of pure, freshly ignited zinc oxide, dissolve in about 5 cc. of dilute nitric acid (1 : 4) and evaporate nearly to dryness. Dilute with 100 cc. hot water, add 7 gm. pure ammonium chloride and 15 cc. of 0·88 ammonia, dilute to about 300 cc. with hot 1% ammonium-chloride solution and add 30 cc. of ferric-tartrate emulsion. Titrate at 70° C. with potassium ferro-cyanide solution, until one drop of the solution gives a faint blue colour on a spot plate with the indicator.

The difference in weight of the two quantities of zinc taken, divided by the difference in the cubic centimetres of solution taken, is the absolute standard. The weight of zinc taken, subtracted from the cubic centimetres of ferrocyanide used multiplied by the absolute standard, is the "blank" to be deducted from all determinations.

The Assay.—As no titration should require less than 2 cc. nor over 25 cc. of the ferrocyanide solution, weigh out 0·5, 1 gm. or 2 gm. of sample into an 8 oz. beaker. If sulphide, dissolve in 6 cc. to 10 cc. of strong nitric acid and boil. When red fumes cease, add 1 gm. of chlorate and evaporate to complete dryness. This precipitates the manganese as oxide.

For roasted ores, oxides, etc., boil with 7 cc. to 10 cc. of strong hydrochloric acid, evaporate nearly to dryness and add 10 cc. HNO_3. Boil until most of the acid is decomposed, then add a little powdered potassium chlorate and evaporate to complete dryness.

Cool, add 100 cc. hot water, break up the mass with a rubber-tipped rod, and add 7 gm. of ammonium chloride, 15 cc. of 0·88 sp. gr. ammonia and about 0·5 gm. sodium peroxide. Boil for a few minutes and filter off the precipitated iron, alumina and manganese. Redissolve the precipitate in the least necessary hydrochloric acid and reprecipitate. Wash the precipitate with boiling 1% ammonium-chloride solution until the bulk of the filtrate is about 300 cc.

Add 30 cc. of the ferric-tartrate emulsion and titrate at 70° C. with potassium ferrocyanide. If copper is found to be present, before adding the ferric tartrate emulsion make the solution just acid with hydrochloric acid, add 1 cc. excess, and then about 20 gm. of clean granulated lead (zinc free). Warm on the sand bath with occasional stirring until all the copper is precipitated, decant the liquid, wash several times by decantation, add 30 cc. ferric-tartrate emulsion, 15 cc. of 0·88 ammonia, and titrate, using the acetic-acid indicator on a spot plate.

It is advisable, when dissolving roasted and sintered ores, to rub with a little water first, to prevent adhesion.

With very low-grade ores the amount of ammonia present may prevent the precipitation of the zinc when titrating. In this case acetic acid is added until a precipitate forms, and the titration is then completed.

[In connection with any method of ferrocyanide titration it should be remembered that the zinc ferrocyanide precipitate is of constant composition only in case the ferrocyanide is in excess. To obtain the best results, therefore, the solution to be titrated should be divided into halves fairly closely, and one-half titrated. All except 1 cc. or 2 cc. of the second half of the solution should be added and the end point over-run with the ferrocyanide solution. Allow to stand a few seconds and add the last of the solution, finishing as accurately as possible.

Where many zinc determinations are run, e.p. ammonium chloride becomes a burdensome and apparently unnecessary item of expense, for the ordinary commercial sal ammoniac seems to answer perfectly.—Editor, *E. and M. J.*"—*Engineering and Mining Journal*, Feb. 6, 1915, p. 285. (G. H. S.)

A Rapid Method of Estimating Nitrates.—The oldest method for the quantitative determination of combined nitric acid is that of Pelouze, in which a known weight of the substance to be assayed is boiled with excess of ferrous chloride (or sulphate) and concentrated hydrochloric acid, when the following reaction takes place :—

$$3FeCl_2+HNO_3+3HCl=3FeCl_3+NO+2H_2O$$

The remaining ferrous chloride is then estimated either with standard bichromate or permanganate. An alternative method (Knecht and Hibbert, "New reduction methods in volumetric analysis," p. 67) is to estimate the ferric chloride directly by means of titanous chloride.

The nitrometer of Lunge affords a rapid and accurate means of estimating combined nitric acid, and is perhaps the most universally used, especially in works practice. It is based upon the fact that nitrates, in presence of concentrated sulphuric acid and metallic mercury, yield the whole of their nitrogen as nitric oxide, which is measured directly as such.

A third method is a gravimetric one and is based upon the property of diphenyl-endanilohydrotriazole (Nitron) of forming a sparingly soluble precipitate with nitrates.

A fourth method of estimating nitrates is based upon the quantitative reduction of nitric acid to ammonia in presence of excess of caustic alkali by means of a suitable reducing agent, according to the scheme :—

$$HNO_3+4H_2=NH_3+3H_2O$$

The ammonia formed is collected in standard sulphuric acid and the extent of the neutralisation of this is ascertained by titration with standard alkali, or in case very small quantities only are present the distillate is nesslerized. The reducing agent originally employed in this method was zinc dust, but later it was found that aluminium was more suitable for the purpose. In both cases, however, a considerable amount of time was considered necessary to effect complete reduction.

In a paper published in 1903 (Ber., *36*, 166; this J., 1903, 232) the author drew attention to the fact that titanous hydroxide was capable of effecting the complete reduction of nitrates to ammonia. A further examination of this reaction showed that the reduction is almost instantaneous.

When caustic soda is added to a solution of a titanous salt, black titanous hydroxide is precipitated, but this begins to decompose almost at once, yielding nascent hydrogen and the white titanic hydroxide, probably according to the following equations :—

$$Ti_2(SO_4)_3+6NaOH=Ti_2(OH)_6+3Na_2SO_4$$
$$Ti_2(OH)_6+2H_2O=2Ti(OH)_4+H_2.$$

The new method, or rather the modification of the old method, is based upon this reaction, and when properly carried out has given entirely satisfactory results.

A convenient amount of nitrate for a single estimation is about the equivalent of 0·1 gm. potassium nitrate. Thus, in the assay of a commercial sodium nitrate, about 1 gm. is accurately weighed, dissolved in water, and made up to 100 cc. Of this, 10 cc. are measured into a copper flask, excess

of caustic soda is added, and then about 20 cc. of commercial titanous sulphate or chloride. The distillation can then at once be proceeded with and, after the boiling has been continued for a quarter of an hour, the operation is finished and back titration is effected.

It is essential that the reagents should be added in the sequence as stated, for if the acid solution of the titanous salt is allowed to react with the nitrate, reactions take place which result in the expulsion of a portion of the nitrogen either as nitrous or nitric oxide, and the result is consequently unreliable. It is also obvious that the titanous salt should be free from ammonia. Personally I have not had any trouble in this respect, but I am informed that ammonia has been found in small quantities in some of the older commercial products. In the newer products, however, this is not the case, and the price need not be a deterrent, for two estimations can be carried out at a cost of the reagent of considerably less than one penny.

Two estimations carried out in the manner described on pure potassium nitrate gave respectively 99·80% and 99·83%. The results have been corroborated by different observers and have invariably revealed the reliability of the process.

Nitrites are also quantitatively reduced to ammonia by the action of titanous hydroxide in presence of excess of caustic alkali. This was shown to be the case by a duplicate experiment with pure crystallized silver nitrite, which yielded in both cases 99·47% $AgNO_2$.

The quantitative reduction of nitric acid to ammonia by the lower oxide of a metal as described in the above short note is, I believe, the only instance of its kind, and I am convinced that the method affords the most rapid of all existing processes for the exact estimation of combined nitric acid. Any curtailment of time required for such an estimation is a gain to the analytical chemist and indirectly to the industries. The method has been introduced with success into the practical curriculum of the Chemical Department of the Municipal School of Technology, Manchester."—E. KNECHT. *Journ. Soc. Chem. Ind.*, Feb. 15, 1915, p. 126. (J. A. W.)

METALLURGY

SHELL BALLS *v.* FLINT PEBBLES.—"The star of the flint pebble in certain kinds of tube-mill work seems to be about to wane, even if it be not going to set, unless we are in dreaming too sweeping conclusions from reports of mill men that reach us. The tube-mill was developed from the ball mill, in which the crushing media were a steel lining and a set of steel balls. The consumption of steel per ton of material crushed was always high, although both balls and lining were made of manganese steel, in some cases at least, from an early date. For this reason chiefly flint pebbles were early substituted in the tube-mills of metallurgical practice, and various expedients were adopted to reduce the wear of steel on the linings.

Lately, however, some enterprising mill men have reverted to the use of balls of manganese steel or chrome steel, which may be of superior hardness to what were used in the early days of the ball mills; as to this we do not know. Any way, it is found that with a given mill, the capacity is greatly increased by the use of steel balls, so much so in certain cases that two mills are able to do the work for which three were previously required.

In the previous paragraphs we have had reference especially to the use of the tube-mills in connection with processes of mechanical concentration. In connection with hydrometallurgical processes there may be other considerations. While cost and service may be paramount in many cases, there still remain to be considered those instances where metallurgical effect must be regarded. It is possible that by adding finely ground iron to a pulp destined for cyanidation, an excessive cyanide consumption would result. Should this happen, the steel balls could not be used in cyanide plants. If they are hard enough to avert that difficulty, however, there is no apparent reason why they should not be used."—*Engineering and Mining Journal*, Dec. 26, 1914, p. 1154. (G. H. S.)

THE ARSENATES OF LEAD.—Pure $PbHAsO_4$ was prepared, as a white amorphous powder, by treating lead nitrate solution with disodium arsenate, washing the precipitate, dissolving in the minimum quantity of nitric acid, slowly adding dilute ammonia, with constant stirring, until about three-fourths of the acid were neutralized, allowing to stand overnight, filtering, washing the precipitate with water slightly acidified from nitrates, and drying at 110° C. When fused at a dull red heat, it gave the pyroarsenate, and, by heating with ammonia, a basic salt, $2PbO,(AsO_4)_2,2Pb(OH)_2,10H_2O$. The precipitates formed by treating lead acetate or nitrate with disodium arsenate were, in each case, mixtures of the acid and basic salt, both of which are insoluble in water. The sp. gr. of $PbHAsO_4$ at 20°/4° C. was 5·786, and that of the basic salt, 7·105. The acid arsenate is determined in commercial arsenate of lead by digesting 3—10 gm. (dried at 100° C.) for a few hours, at the ordinary temperature, with 200 cc. of a 5% solution of ammonia (free from carbonate), filtering by means of a Buchner funnel, prepared with a pad of two sheets of filter paper, with a layer of asbestos between (the upper paper being a hardened filter), washing with boiled water until the washings give no turbidity with lead nitrate, and determining arsenic in an aliquot part of the clear filtrate, after boiling off ammonia, by the modified method of Gooch and Browning (Bull. 107 (Revised), Bur. Chem., U.S. Dept. Agric.). The amount of arsenic in the total filtrate, calculated as As_2O_5 and multiplied by the factor 7·6034, gives the weight of $PbHAsO_4$ present in the original mixture.—H. V. TARTAR AND R. H. ROBINSON.—*Journ. Soc. Chem. Ind.*, Oct. 31, 1914, p. 1008. (J. A. W.)

PRODUCTION OF ZINC SHAVINGS.—"Mr. C. C. Freeman, Metallurgist, has kindly furnished the following notes on the "Production of Zinc Shavings," which are particularly interesting at the present time:

"Most of the English zinc smelters aim at producing a spelter suitable for use in making a good quality brass for which purpose very little lead is permissible, and such a spelter cannot be conveniently rolled. On the other hand, many of the Continental works treat zinc ores and concentrates containing appreciable quantities of lead, and the resulting spelter, after refining, usually contains from 1% to 1·5% of lead, which renders their product amenable to rolling; most of the European sheet zinc is, therefore, produced at the Continental works.

Influence of Impurities.—The rolling qualities of spelter are affected to a marked degree by the impurities present, of which the most important are:

1. *Lead.*—Moderate amounts of lead render zinc malleable and ductile, and up to 1½% permits good rolling. With more than 1½% of lead, zinc becomes tender, and although zinc containing 3% of lead can be successfully rolled, the sheets are very tender and liable to crack.

2. *Iron.*—Iron reduces the malleability of zinc and makes it harder and more brittle. Up to 0·125% of iron is not injurious, but above this the bad effects become apparent, and 0·2% of iron seriously interferes with rolling.

3. *Oxygen.*—This may occur as zinc oxide, due to imperfections in distilling; if much of this is present the spelter will become brittle and difficult to work.

Other impurities may occasionally be present in spelter, and of these cadmium (up to 15%) does not interfere with rolling, whilst copper, tin and arsenic render zinc hard and brittle.

The spelter produced by the Broken Hill Proprietary Company contains about 1·2% lead and 0·06% iron, and, therefore, presents no difficulty in rolling into sheets.

Physical Properties.—At ordinary temperatures zinc is quite brittle, and ingots can be easily broken; between 100° C. and 150° C. it becomes malleable and ductile, but above the latter temperature it again becomes brittle, and at 200° C. can be reduced to a powder by pounding. The relatively coarse crystalline structure of cast zinc is destroyed by rolling or hammering, and the specific gravity of the wrought metal is about 7·25 compared with 7·05 when cast. Wrought zinc heated to between 100° C. and 150° C. retains its malleability and ductility after cooling.

Lithgow Test.—In the spelter rolling test recently carried out by Messrs. Hoskins & Co., at the Lithgow Works, their ordinary sheet iron rolling practice was followed. Each ingot contained sufficient metal to form two eight foot sheets of the requisite gauge and two of these ingots, heated to about 125° C., were first rolled into sheets about 5 ft. long and 0·07 in. thick; each of these sheets was then folded in halves and the double thickness further rolled into lengths of about 4 ft.; one doubled sheet was then placed over the other, and the four thicknesses rolled until the full length of 8 ft. ×3 ft. was obtained. After dressing the edges in a plate shearing machine, the individual sheets were easily parted from one another.

Shavings.—In recent years most of the shavings produced are turned from 8 ft. ×3 ft. sheets, about twelve of which are joined end to end and tightly coiled on a wooden mandril, forming a cylinder 3 ft. long and about 6 in. in diameter. This is placed in a lathe, and a tool with a lateral travel cuts the zinc from the end of the cylinder. The average price of sheet zinc, f.o.b., Antwerp, during the year 1913, was £3 14s. 6d. above that for spelter, so that the former may, under normal conditions, be landed in Australia at about £6 per ton above the ruling market price for spelter.

From time to time experiments have been made to try and produce zinc shavings by methods whereby the preliminary cost of rolling into sheet form can be eliminated.

By use of a special turning tool, it has been proved that it is possible to produce a shaving direct from cast spelter, but the process is so slow that the cost of such turning is greater than that of shavings produced in the ordinary way. Shavings produced by this method are more brittle than those produced from the wrought metal and would probably yield an excessive quantity of short zinc when used in the precipitation boxes.

A zinc ribbon has recently been placed on the market as a substitute for the ordinary zinc shavings. This is produced by fine streams of molten zinc falling on to a rotating water-cooled drum which has a peripheral speed of about 950 feet per minute. The ribbon, although of about the same thickness, is much wider and more ductile than the ordinary shaving, and after being in contact with cyanide solution for a time it would probably have a tendency to pack in the boxes.

I understand that several of the mines in Western Australia have experimented with the ribbon, and diverse results were obtained; this is probably due to the difference in the alkalinity of the solutions used. The cost of producing ribbon from spelter by this method is about £5 per ton less than the cost of producing shavings in the ordinary way from sheet zinc."—C. C. Fulleman. —*Monthly Journal of Chamber of Mines of W. Australia*, Feb., 1915, p. 11. (H. A. W.)

MISCELLANEOUS.

Ink for Labelling Reagents.—"For writing on labels of reagents and other bottles exposed to laboratory fumes, the following ink is recommended by the *Chemical Trade Journal*: 20 gm. shellac are dissolved in a hot borax solution (30 gm. in 400 cc. water) and filtered while hot. To the filtered liquid the following solution is added: Aniline black, 8 gm.; tannin, 0·3 gm.; picric acid, 0·1 gm.; ammonia, 15 gm., water, 10 gm."—*Engineering and Mining Journal*, Oct. 31, 1914, p. 789. (G. H. S.)

The Sewage Disposal Commission.—"The conclusion of the labours of this Commission, which have extended over a period of seventeen years, and the issue of its final report, enables us to consider what has been the gain to engineers generally as a result of its work. So far the technical Press does not seem to be enthusiastic. It must be admitted that when the Commission first sat the whole question of the disposal of sewage was in a most involved condition. Land treatment appeared at that time to be the only system acceptable to the authorities. The evidence obtained in the interval, and the various conclusions based thereon show that artificial treatment is perfectly reliable, and land treatment alone is now looked on with suspicion, or, at all events, is not welcome. The Commission has certainly amassed a considerable amount of information which will be most useful to the sanitary expert of to-day in evolving schemes, and it is not perhaps for the next ten or twenty years that the full results of the seventeen years' enquiry will be properly appreciated. There is one point, however, upon which there is immediate and cordial agreement amongst the contemporary engineering Press. We refer to portions of the report which state that it is unnecessary to purify all sewage effluents to the same high standard. Allowance is made for local conditions, and the commonsense view is taken that when the volume of water is considerably in excess of that of the effluent discharged into it, and when the stream below the point of discharge is not used as a source of domestic supply, some latitude may be allowed in the standard to be attained. The burdens which have hitherto been laid upon certain communities may thus in future be considerably lightened."—*Ind. Engineering*, June 12, 1915, p. 347. (J. A. W.)

Abstracts of Patent Applications.

(C.) 277 14. William Fyfe. An improved method of fastening pulleys to cam shafts in stamp batteries. 23.6.14.

This application relates to a modification of the well-known Blanton wedge.

The modification consists in making the wedge double, the tapers of the two parts being in opposite directions, thus making a folding wedge, both parts being secured to the shaft when in action.

It is claimed that a wedge of this construction is more readily released than the single wedge.

(C.) 298 14. James West (1), John Edward Jones (2). Improvements in valves used in jigging and other similar machines.

This application refers to a valve for jigging and other similar machines, more particularly to a flap or door moveably suspended by means of links from a shaft, the point of suspension being out side of the water.

(C.) 133 14. William Frederick Smeeth. Improvements in or relating to clutches for winding engines or other machines.

This application is for an arrangement of the clutch for the drums of winding engines, and has for its object a finer adjustment of rope length when changing levels from which hoisting is being done.

With the usual multiple jaw clutch the limit of adjustment is the pitch of the clutch teeth produced to the radius of the coil of rope on the drum which may be two or three times the actual clutch pitch.

The applicant proposes to make the sliding clutch capable of being pushed back clear of the feathers or flats on which it slides and then rotated to assume a fresh angular position when pushed back into clutching position.

By making the number of teeth in the clutch, one tooth more or less than any multiple of the number of positions the sliding clutch can be turned to, he obtains a differential effect on the position of the drum and a correspondingly small effect on the rope length and thus on the position of the skip or cage at a level.

(C.) 30 15. Charles William Dowsett. Improved means for controlling the underflow of concentrators, classifiers and the like.

This application relates to an improved means for automatically controlling the underflow of classifiers, etc., and consists in providing a valve operated by a yielding diaphragm acted upon by pressure transmitted from the pulp container.

(C.) 102 15. Martin Christensen. Improvements in and relating to the wheels and axles of mine trucks and the like.

This application relates to an arrangement of cone bearing for wheels of mine trucks.

The boss of the wheel is bored taper, the larger diameter being on the outside, a bush turned to a similar taper is secured to the axle and arrangements are provided for setting it further in on the axle to take up wear in the two conical surfaces.

Outside the face of the wheel boss a plate is fixed with a conical end bearing on the end of the axle; a modified form has a spring interposed between the plate and axle.

A recess for carrying lubricants is provided in the boss of the wheel.

It is claimed that repairs and renewals will be greatly facilitated and cheapened by having easily replaceable wearing parts.

(C.) 77 15. John Emery Bucher. A process for forming cyanides and the like. 13.4.15.

This application refers to a process for forming cyanogen compounds from nitrogen, carbon and an alkali or alkaline earth metal or compound thereof under the influence of a molten metallic catalyser, also an apparatus for carrying on the necessary work, in which apparatus the catalytic material is maintained in a fused condition by means of electric current. Means are also provided for causing the vapour of a base and free nitrogen to pass through the bath of fused catalytic material.

(C.) 143 15. Anders Jordahl. Improvements in or relating to reinforced concrete structures. 27.6.15.

This application has reference to the use in reinforcement for concrete structures, of bars placed one above the other and connected by a slotted web, thus enabling the diagonal shear members to be formed by connecting one of the bars on the connecting web at intervals, and thus bending the severed bar to suitable shape. The second claim refers to a bar of similar type in which three bars placed one above the other are connected by slotted webs, such bars being used in continuous beams where negative bending moments occur.

(C.) 126 15. John Van Nostrand Door. Improvements in continuously separating solids from liquids.

This application relates to apparatus for settling or separating finely divided solids from a portion of the liquid in which they are suspended, or for separating quickly settling solids from those suspended in the same liquid which settle more slowly. The applicant proposes to employ a sensitive hydrometer which, in responding to the varying densities, as in a classifier or de-watering machine, operates regulating mechanism or, alternatively, closes an electric circuit which sounds an alarm and brings into operation a solenoid or other suitable electro-magnetic device arranged to operate a lever for regulating purposes.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

BALL, H. S., *l/o* Johannesburg; *c/o* Standard Bank of S.A., Ltd., 10, Clements Lane, London, E.C.

HONNOLD, W. L., *to* Hyde Park Hotel, Knightsbridge, London, S.W.

MACKAY, A. N., *l/o* England; *c/o* Frontino and Bolivia Mines, La Salada, *c/o* Empresa Hanseatica, Barranquilla, Republic of Columbia, S. America.

MCLENNAN, J., *l/o* Johannesburg; Prestea Block A, via Sekondi, Gold Coast Colony, W. Africa.

SORENSEN, S. S., *l/o* Nevada; Sewell, Rancagua, Chile, S. America.

TIFFARD, M. O., *to c/o* Miss Kuhlmann, Empire Road, Parktown.

WHITE, S. B., *to* Corinthian North G.M. Co., Corinthian, W. Australia.

WILLIAMSON, R., *to* Corinthian North G.M. Co., Corinthian, W. Australia.

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Proceedings
AT
Ordinary General Meeting, October 16, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, October 16th, 1915, Mr. J. E. Thomas (President) in the chair. There were also present:—

25 Members: Messrs. H. A. White, Prof. J. A. Wilkinson, Prof. J. S. Cellier, E. H. Croghan, F. W. Watson, John Watson, Dr. W. A. Caldecott, W. R. Dowling, Prof. G. H. Stanley (Members of Council), W. J. Abel, H. R. Adam, R. W. Bennett, H. C. Boydell, J. M. Dixon, T. Halbert, B. J. Hastings, W. H. Jane, A. King, H. M. Leslie, G. Melvill, S. Nettleton, E. A. Osterloh, W. A. C. Tayler, A. Thomas, and M. Weinbren.

8 Associates and Students: Messrs. J. Davidson, O. A. Gerber, F. J. Pooler, R. A. Porter, D. W. Pugh, H. Rusden, T. L. Thorne, and H. Ward.

6 Visitors, and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBERS.

Dr. W. A. Caldecott and Mr. W. R. Dowling were elected scrutineers and after their scrutiny of the ballot papers, it was declared that the following candidates for membership had been elected:—

BLAIKIE, ROBERT FRANCIS. Rand Mines, Ltd., P.O. Box 1056, Johannesburg. Mining Engineer.

WILLIAMS, LESLIE BALLESAT. Queen of the Hills G.M. Co., Meekatharra, West Australia. Mining Engineer.

GENERAL BUSINESS.

DR. HAHN.

The President: Before proceeding further, I might mention that, at our last meeting, a question was raised by Mr. John R. Williams in regard to the action of the Council in electing Dr. Hahn as Honorary Member of the Society. Mr. Williams made a very vigorous protest about the proceeding, and the whole matter was carefully reconsidered by your Council. In view of the fact that Dr. Hahn was born on the 5th January, 1849, at Bethany, in Namaqualand, which was then a British Colony, and is therefore a British subject, your Council did not see why they should alter their previous decision, and Dr. Hahn remains an honorary member.

A DESCRIPTION OF AN IMPROVED METHOD OF REMOVING THE BROKEN ORE FROM STOPES.

Mr. H. C. Boydell, B.Sc. (*Member*): Mr. Weinbren's remarks on the general inefficiency of shovelling in stopes are certainly to the point. It would be hard to imagine anything much more crude in the way of hand labour than the efforts of a gang of, say, half a dozen natives in getting ore from the top of a stope with 250 ft. backs to the boxhole at the bottom. The natives arrange themselves more or less in a row down the stope, scratch the ore from one to the other, and the distance the material is moved at each stroke of the shovel usually does not exceed 3 ft., if it even attains that. In its travel downwards, the material spreads out and assumes, roughly, the shape of a fan with a base of ever increasing length, so that when it eventually reaches the bottom it has to be collected in the same painful manner in order to pass into the boxhole from which it is to be trammed. Whilst this has been going on the tramming natives will probably have been waiting with trucks at the empty box, enjoying their share of inaction, and

being quite content with yelling "pezula" up the boxhole when anybody in authority has come along and noticed what has been going on. The pity of it is that in the majority of instances this takes place under the very eyes of white trommie or shovelling gangers, who, knowing no better, see nothing wrong with work of this kind. Probably not 5 of the white men employed in control of native shovellers on the Rand have ever done ten continuous minutes of shovelling in their lives, and are quite incapable of showing the natives under them how to use their shovels to the best advantage.

This inefficient handling of the broken ore is, as Mr. Weinbren has mentioned, greatly reduced by a properly arranged and maintained system of stope tracks and chutes. With these the distance that the ore has to be shovelled is never greater than that between two adjacent tracks, and allowing for the distance that blasting throws it down the stope, is on the average considerably less. The stulls for the tracks should be carried right up to the face, and the use of a full length (18 ft. to 20 ft.) pump set is preferable to the half-section 9 ft. of rails mentioned by Mr. Weinbren. Apart from the saving of labour in the handling of the ore in its passage to the boxhole, a system of fixed chutes fed by stope tracks has the further great advantage of keeping the working benches on the stope face reasonably free from broken ore and ready for drilling on. Where stope tracks are not used the broken ore collects on the benches further down the face, and this tendency increases from the top to the bottom of the stope. The result is that in many cases the stope gets out of shape, the point or bottom moves forward slower than the upper part, the stope gets too flat, and the disadvantages just mentioned become intensified. Under these conditions the holing of a fresh boxhole is a slow operation.

The use of stope tracks obviates these difficulties, as none of the broken ore travels further down than the distance between two tracks, and the benches at the bottom of the stope have no greater tendency to collect broken ore than those at the top.

Any one who has had experience in the working of stope stopes with "Lancaster boys" on the underhand system knows the difficulties attendant on keeping the point of the stope free from broken ore and the starting of drilling work there each shift in time to prevent that part of the face getting further and further behind. This laggard of underhand stoping can be minimised to a considerable extent by the use of one or more stope tracks running on the stulls, which are essential, in any case, for the safe working of such stopes. Of course, in steep stopes, chutes are not required, their place being taken by lagged passes which have as their bottom the footwall of the stope.

Another advantage connected with the use of stope tracks and the necessary stulls is that the latter keep the stope safe from rolling stones, tend to collect material on the footwall, and generally make the stope safer and easier for getting about in.

Still another advantage of stope tracks is that with them but little time is lost in starting tramming on the level at the beginning of each shift, the box or boxes there being fed by the stope tracks from different parts of the stope. Where shovelling alone is relied on and the broken ore is well up the stope to start with, it is some time before any of it reaches the level. At the Village Deep, where the systematic use of stope tracks and chutes has been practised for a number of years, a side-tipping truck of 10 cub. ft. capacity is in use, the top standing 2 ft. 5 in. above the rails. A hinged short length of rail runs over the chute and enables the truck to be tipped over the latter. This is swung back out of the way when the tipping is finished in order that the chute at this point shall not be obstructed for the use of the track higher up the stope.

A little extra time and labour spent in properly laying and grading the chutes, laying the stope tracks and increasing the head room for tipping is amply repaid by the greater efficiency obtained in the subsequent working.

Mr. Weinbren's claim that the method reduces the number of boxholes required is not a sound one in my opinion. With an interval of 60 ft. to 70 ft., as he states, between adjacent boxes, the point of the stope gets too flat, shovelling work is considerably increased, and owing to the broken ore lying on the benches, difficulty is experienced in getting down to cut or hole the next boxhole.

Generally speaking, the method, taking into consideration the dip of the reef in most of the mines, is capable of much wider application on the Rand than it appears to have at present.

NOTE ON COVERING OF SAND DUMPS WITH SLIME RESIDUE AND BUILDING SLIME RESIDUE DAMS.

Dr. W. A. Caldecott *(Past-President):* In the discussion which took place some time ago* upon the covering of sand residue dumps by spraying with thick slime residue pulp, the manner in which the 3·5 % common salt used on the weight of the pulp assisted in maintaining a coherent coating of dried slime was not made clear. On consideration it appeared to the writer that this effect might be largely due to the formation of hygroscopic calcium chloride through the inter-action of the salt with the dissolved calcium hydrate in the residue pulp. Obviously on this assumption the amount of salt used was largely in excess of the calcium hydrate present, and after a series of experiments the use of equal parts of salt and fine slaked lime was adopted at the Knights Deep, with satisfactory results in regular practice. The amount added to the slime pulp is 0·25 % each of salt and lime on the dry weight of the slime residue. The use of the above reagents does not of course prevent the cracking of the slime layer due to its contraction on drying, for which the only remedy appears to be repeated coatings which fill the cracks. For abandoned portions of a sand residue dump a coating of slime residue several feet thick may be used, and is formed by building up a long narrow slime residue dam upwards from the base of the dump, until the whole face of the dump is covered and such portions of the top as are desired. This latter method was successfully employed at the Robinson Deep G.M. Co. several years ago.†

In reference to the ordinary slime residue dam, it was pointed out to the writer some years ago by the late Mr. H. Weldon, Inspector of Mines, that the usual wall of dried slime residue is only prevented from falling inwards by the viscous mass of pulp inside the dam. When the dam attains a certain height the fluid pressure on the toe of the wall is liable to thrust it horizontally outwards, with the resulting collapse of the wall and escape of portions of the contents of the dam. Where space permits it might be desirable after the main dam has attained a very moderate height, to start a narrow subsidiary dam round the base to strengthen the latter. As the main dam rises in height the subsidiary is raised likewise, so as to maintain the same level below the crest of the former; in time a second subsidiary is started, and so on. By this means the outer face of the dam wall, instead of having a fairly steep slope, would be buttressed by means of subsidiary stepped dams, and thus the tendency of the toe of the main dam wall to movement outwards be checked. The same effect might be produced by using a long narrow subsidiary sand residue dump outside the slime residue dam wall, but this method might in many cases prove somewhat inconvenient and costly.

The President: I think these notes on the covering of sand dumps are most appropriate, considering the weather we have had lately. Most of us have suffered considerably from them, and the fact that slaked lime can be used to replace some of the salt will cheapen things and probably commend itself to the attention of General Managers of mines and induce them to take a more active interest in the matter. In connection with slime dams, it would appear, in the system which Dr. Caldecott mentions, one might lose area, but, as Mr. Dowling pointed out to me the other day, these dams growing up with the other dams would practically give the same area as if one were building up an outside wall all the time.

CLEAN-UP ROOM PRACTICE AT THE SIMMER DEEP.

By W. H. Jane (Member) and F. Davey.

The black sand swept from the plates during dressing is accumulated for 24 hours. At 6 a.m. it is put into a ball mill with 200 oz. mercury and ½ gallon of water and ground for 4½ hours. The amalgam scraped from the plates each morning, plus 1½ times its weight of mercury, and 1 gallon of water is then added to the contents of the mill. The mill is run for a further two hours, stopped, and the contents carefully discharged into an enamelled bucket, any overflow going into a tray placed underneath the mill. The bucket with its contents is then taken to an amalgamated copper table, and the contents washed into an enamelled dish. The amal-

* See this *Journal*, March, 1914, *et seq.*

† See the Johannesburg *Star*, July 3, 1914.

gam is washed clean with a good stream of water, which carries all the rich slime into a vat placed at the foot of the table. These washings are accumulated for one month and receive further treatment in the barrel and batea. The washed amalgam is then cleaned of particles of steel by means of a horseshoe magnet, and carefully skimmed. After stirring and transferring from one dish to another several times, each time being carefully skimmed, the amalgam is taken to the amalgam press. The press, fully opened, is filled with as much clean water as it will hold. The amalgam is then added, the press closed, and the amalgam pressed into a cake. It has been our experience that this method of filling the press with cold water, whilst not giving so high a percentage of fine gold as can be obtained by using hot water, gives a higher percentage than where no water is added to the press, and yields, here, pressed amalgam containing consistently 37 % fine gold from amalgam scraped from plates. This is obtained by making the maximum size of the cake 500 oz. The skimmings from the amalgam, consisting of particles of unerushed amalgam, amalgamated base metals, and minute quantities of osmiridium, are collected daily, and locked in the strong-room with amalgam from the plates. It is treated monthly, being placed in the batea with half its weight of mercury, and sufficient water to form a thin slime, but not enough for the batea to throw out any of its contents.

The batea is run for four hours. After taking out the balls, the whole of its contents are taken to the clean-up table and cleaned in the same way as the amalgam from the plates, but require more stirring and skimming, and also the addition of more mercury. The skimmings yield 60 % of its weight of amalgam containing 32 % fine gold. The skimming from this cleaning is panned for osmiridium, and are put into barrels with material from the general clean up. The amalgam recovered by steaming mill plates is placed in ball mills with 1½ times its weight of mercury, water added, and the mills run for one hour. This amalgam requires less cleaning and yields about 30 % fine gold. The skimming is placed with that from the daily scrapes. From the tube-mill liners a considerable amount of gold is recovered in the following manner: All material retained by the liner after the working load has been run out is brought to the clean-up room. There it is treated in two lots, termed "No 1" and "Cement". "No. 1" consists of all the loose material in the tube-mill after the bars are taken out, and "Cement" is the cement adhering to the shell and also that cleaned from bars. "No. 1" is washed and sorted on a ¾ in. mesh screen, the wood, pebbles, bolts, etc., sorted out, and the pebbles returned to the tube-mill. The product passing the ¾ in. mesh screen consists of particles of steel or battery chips, rich pyrite, and coarse free gold not amalgamated, but amalgamating readily in barrels. This is placed in a barrel and treated as described later. The fine and cement are treated separately. The amalgam recovered from these sources yields an average of about 46 % fine gold. From the monthly general clean up of launders, etc., the pieces of wood, waste, etc., are sorted out on clean-up tables, and cleaned separately, and large pieces of amalgam roughly broken, and the whole treated in the barrel and batea. The amalgam from this source averages about 36 % fine gold.

The floor of the clean-up room is granolithic and slopes to a channel leading to a sump. All floor washings, etc., flow into this sump, the overflow from which runs into a channel leading to another sump, from which, after settling, the clear water is pumped to the mill tailing pit. These sumps are cleaned out monthly, and with the contents of the small vats at the foot of the clean-up tables and trays under the ball mills, are treated in barrels and bateas, and yield an amalgam of 36 % fine gold. When treating material in a barrel and batea, a rough estimate is formed of the amalgam likely to be recovered, and 2½ times that weight of mercury added, with about 3 lb. of lime and sufficient water to form an easy flowing pulp. The barrel is run for 15 hours. Immediately after stopping the door is taken off and the barrel turned until the contents discharge into an iron tray with a chute leading to a batea. Close examination of the pulp is necessary, as, if the richness of the contents is under-estimated, the amalgam will be too thick for treatment in the batea. This can be remedied by adding mercury in the batea. The pulp from the barrel flows into the batea, and this in turn throws out into a launder leading to a large settling vat sunk into the floor. At the foot of the launder is placed a 9 mesh sieve to prevent wood, waste, etc., getting into the vat. As soon as the batea starts throwing out steel a 100-mesh sieve is put in place of the coarser one, and the steel remaining

on the sieve taken to No. 1 vat for steel, which will be described later. The batea is stopped when it ceases to throw out steel. After the balls are removed the steel is carefully removed from the main body of amalgam and taken to No. 1 vat. The amalgam is then dipped out of the batea, cleaned and pressed. The skimming, mostly steel, is also placed in No. 1 vat. Four vats are required for the treatment of the steel caught in and outside the batea, and are known as Nos. 1, 2, 3 and 4 steel vats. The contents of No. 1 vat are screened in water on 100-mesh sieves, the plus being put into No. 2 vat, and the minus into No. 3 vat. The contents of No. 2 vat are cleaned on 100-mesh sieves in water. Pebbles and amalgamated copper chips are sorted out. The pebbles are returned to the mill and the amalgamated copper chips, from 50 oz. to 100 oz. per month, containing 1½ fine gold, retorted at intervals and the residue sent to the pan furnace. During the present year, at different times, four diamonds have been found in this material. About 10 tons of +100-mesh steel are collected in six months and dumped on a convenient site outside. There it is allowed to rust for some months and then washed again on a 100-mesh sieve. From a lot of 10 tons treated in this way after three months' exposure, 400 lb. of fine was obtained, assaying 25 oz. fine gold per ton. The same lot after 18 months' exposure yielded 20 −100-mesh, assaying 4·3 oz. fine gold per ton.

The contents of No. 3 steel vat, containing fine steel and pyrite, averaging about 75 steel, with some free amalgam, are treated in the batea. About 60 lb. are put into the batea with 400 oz. mercury. The batea is started and a steady stream of water is kept flowing into it. As the batea keeps throwing out steel it is caught and taken to No. 1 steel vat. The remainder of the steel is gradually fed into the batea, which is run until it ceases to throw out steel. It is then stopped and cleaned out as described before. The fine steel collected with the amalgam in the batea is returned to No. 3 steel vat and treated with the next charge. If great care is not taken when opening the barrel, and too little mercury has been used, the steel in No. 1 vat will be too mixed with thick amalgam to clean, and the method described for treating fine steel will be found satisfactory. The amalgam is cleaned as usual. The skimming, being the final skimming of the whole treatment, is carefully panned free of steel, which is taken to No. 4 steel vat.

This leaves the base amalgam, and any osmiridium which has been concentrated down to this point, in the pan. The amalgam is panned off and the osmiridium in the pan collected. The base amalgam is put with the amalgamated copper chips. The fine steel is taken from No. 1 vat and washed over the clean-up table, which is cleaned after every 80 lb. are treated. This reduces it value by about 25 oz. fine gold per ton. It is caught in a tray placed at the foot of the table and bagged whilst wet. When dry it is run through a ball-mill, re-bagged and sold to the smelting works. From 3 to 4 tons, assaying from 15 oz. to 20 oz. fine gold per ton, are collected every six months.

The amalgam collected on the clean-up tables is very base, and is treated in the ball-mill with twice its weight of mercury for an hour, cleaned as usual and the skimming put with amalgamated copper chips after panning for osmiridium.

The large vat in the floor into which the pulp from the batea flows, and also the overflow from the vats at the foot of the clean-up tables, collects all solids. Its contents are always covered with water. It contains, when full, just over 10 tons. This at intervals is treated in the black sand cyanide plant, consisting of a small tube-mill, circulating pumps, and two treatment vats with conical bottoms. The ordinary decantation process being the system installed. The pulp from the vat is carried in buckets to a screen above the circulating pump, and washed into the tube-mill circuit. It is then pumped to a small cone placed above the tube-mill inlet; the fine overflow passing into the settling vat, and the underflow through the tube-mill. On leaving the tube-mill it passes over an amalgamated copper plate back to the circuit; at the head of the plate is placed a ¼ in. mesh sieve. This, when full of pebble chips and steel, is quickly changed and the contents washed through a 100-mesh sieve into No. 1 steel vat, the coarse remaining on the sieve being sorted later on. When the settling vat is three-quarters full, 100-mesh sieves are used at the head of the plate and the feed from pulp vat stopped. The tube is kept running until the settling vat is seven-eighths full, the box sieves being kept clean and the 100-mesh sieves changed and contents screened into No. 4 steel vat. By this time the pulp leaving the plate is nearly all steel, −100-mesh, and is collected and put into No. 4

steel vat. This fine steel is not so rich in amalgam as the fine steel from the latter. A recovery of about 5 oz. fine gold per ton is obtained by washing it over the clean-up table.

After cleaning up all fine steel, the tube-mill is stopped for the day. After settling, the water is pumped from the decanters to the mill sump. About two tons of pulp are treated per day, the tube-mill running for four hours. The plate is well dressed before starting each day and scraped as soon as the tube-mill is stopped. When 10 tons of pulp have been collected in the settling vat, the tube-mill is opened and any amalgam found put with the scrapes. This amalgam is given an hour's run in the ball-mill with twice its weight of mercury, and cleaned in the usual way. The skimming being very base, is put with the amalgamated copper chips, after panning for osmiridium. The amalgam gives 30% fine gold. The clean water in settling vat, which now becomes No. 1 treatment vat, is decanted to half way, pulp circulated for one hour, and then transferred to No. 2 vat, in which 2·0 lb. of cyanide has been dissolved. The original sample is taken during this transfer. The charge is circulated and pumped back to No. 1 vat, settled for 3 hours, and the solution, about 10 tons per day, decanted to the zinc boxes. The charge has this double transfer each day, using 10 tons of 0·1% KCN solution. Solution samples for assay are taken each day, and when assaying under 1 dwt. fine gold per ton of solution, a sample of the pulp is taken for undissolved gold. If the result is satisfactory, two water washes are given and the charge is pumped to a small dam outside, and when sufficiently dry to be easily handled is put with the current tailing.

From an original value averaging 177 dwt. fine gold per ton, an extraction of 97·8% is obtained. The average grading of the pulp after tube-milling is 95% -200-mesh.

The retorting and smelting of amalgam and gold takes place at intervals. Two retorts are used, each holding about 5,000 oz. of amalgam, in eight trays. They are loaded at 7 a.m., fires being started after the retorts are closed. Retorting generally is finished three hours after the mercury starts dripping from the condenser. The water given off, *i.e.*, moisture in amalgam which comes off first, is weighed with the mercury. If this is not weighed, the supposed loss of mercury is increased by 1 to 1½%. The loss of mercury is very little. The retorted gold is melted in crucibles of suitable size in the usual bullion furnaces. A handful of borax is added with the last bit of gold put into the crucible. When the gold is molten the slag is skimmed off, half a teaspoonful of charcoal added, and as soon as the molten gold is poured, a little borax is sifted on to its surface. The loss of weight in smelting averages 0·5%. The bars are easily cleaned, and the average fineness is 892·5 gold and 103 silver.

The clean-up room is large, giving plenty of room for all operations. No pulp leaves the clean-up room until discharged as residue from the black sand treatment. All wood chips, blocks from tube-mill liners, waste, old launder belting and any other combustible stuff are cleaned in clean-up room and periodically sent to the pan furnace. Large pieces of wood are cleaned and used up in lighting the bullion furnaces. The ashes from these furnaces, after being crushed and washed, are sold to the smelting works. All worn-out amalgam brooms are burnt in the clean-up room. A 9-mesh screen is placed over the fire-bars of the cleaned-out bullion furnace and the brooms burnt on this. The ashes are collected and stirred into a bucket of water. The floating particles and wire are sent to pan furnace. The settled product put into a ball-mill with an excess of mercury and the mill run for one hour. Over 1 oz. of amalgam per broom is recovered.

In writing the above, the authors wish to enter a plea for what they believe to be a neglected side-line of reduction works. Recognising that 0·01 dwt. fine gold per ton is an almost negligible factor in assaying, yet with the tonnages crushed on the Reef today it amounts to a very considerable item, they ask the members of the Society to let as much light as possible on the subject under discussion.

Since writing the above there has been occasion to send the retort trays to the fitting shop, to have them planed down. In order to get them thoroughly clean, a system of scaling them was adopted. They were first painted with hydrochloric acid and allowed to stand 18 hours. They were then heated to a bright red heat, and immersed in water. A scale about 1/16 in. thick came off easily. This scale, with the addition of water only, was placed in the ball mill, and the mill ran for 12 hours. The mill was then stopped and mercury added to the contents. After a run of 1 hour the mill was stopped and the contents cleaned in the usual way. About 4 oz. fine gold was recovered from each tray.

This we consider very interesting in view of the fact that the trays were apparently clean before sealing.

The authors' thanks are due to Mr. H. S. McGregor, General Manager, and to Mr. J. E. Thomas, Reduction Works Manager, for permission to publish this paper.

The President: In proposing a hearty vote of thanks to Messrs. Jane and Davey for their interesting paper, I wish to say that I hope all our mill men will avail themselves of the opportunity of entering into this discussion. There is not the slightest doubt that there is a great divergence of methods in clean-up room practice on the Rand. As we are all after the same object, that is, to save as much gold as possible, care should be taken, after treating all the huge tonnages we do, that gold should not be allowed to escape in small quantities. We get the gold as far as the clean-up room, and it happens occasionally that gold may escape inadvertently. Any discussion on this subject which will tend towards the saving of a grain of gold will tend to bring matters towards that state of perfection which we all desire. I have here the four diamonds found on the Simmer Deep over a period of eight months. They are diamonds. Dr. Corstophine has examined them and says they are, and told me they were absolutely typical of the diamonds found in banket. I believe some have also been found at Modder B., and I believe it is quite a common occurrence to find them in the mortar boxes in the Klerksdorp district. I have also some black material, containing 80% of osmiridium and 3% platinum, which I will hand round for the inspection of members.

THE USE OF SCOOP DISCHARGES IN TUBE-MILLS.

By W. R. Dowling (Past-President).

(Printed in Journal, March, 1915.)

REPLY TO DISCUSSION.

Mr. W. R. Dowling *(Past-President)*: In thanking those members who have added to our knowledge of the subject by contributing discussion to my paper, I have to acknowledge disappointment that there was not more discussion from many members who could have taken part.

The point Mr. White makes that pulp should not flow back into the tube-mill, either after reaching the outflow trunnion or through leaks in the scoop is of great importance. Should these defects be present the level of the pulp in the mill will be higher and approach ordinary conditions now existing.

The size of the scoop recommended at 24 in. diameter and the sand feed at 600 tons per day were arrived at by actual experiment as being the best figures for practical work under existing conditions. Improvement of liners and pebble feeders on the lines he indicates, as well as consideration of elevating arrangements, and possibly classifiers, may lead to changes of views on these points.

With reference to Mr. White's point as to difference in level of pulp between inlet and outlet ends where the scoop is used, the writer has seen indications of the sand of a mill at rest being lower at the discharge end, and believes that there is a continuous slope of sand from inlet to outlet ends, which is more apparent in a mill provided with a large diameter scoop. If one considers a peripheral discharge mill with ample and free discharge so that the discharge capacity is greater than the feed, the pulp would certainly slope from inlet to outlet. In a scoop discharge mill, provided the capacity of the scoop were sufficient, the same conditions would probably obtain.

Mr. Herald's contribution is a useful compilation of the various liners tried and used in tube-mills, and for the purpose of reference is most valuable. Further, it is an excellent illustration of how members of the Society can place their knowledge and experience on record for the benefit of others, when a suitable peg is provided on which to hang their matter.

Regarding the shoe shank liner described, Mr. Herald has given me the following note:—

"The two-thirds inlet section of tube-mill lining made from shoe shanks gave a nett running life of 297·6 days. After running 162·7 days the lining was patched, 11 shanks all at one place having come loose. The wear on the shanks appears to be from one to one and one half inches, and a considerable number of the shanks will be quite good enough to be made up again.

"Two barrels of concrete from the shanks were treated by the usual clean-up room method, and only a trace of amalgam found in each barrel.

"The weakness in the lining was as anticipated, the last two inches of concrete was

either too weak to support the heavy shrunk, or the cement was washed out too fast by the pulp. I think the former. To overcome, or at least minimise this weakness, another section of shoe shank lining has been installed with a mild steel bar five inches deep and three-quarters of an inch thick between each row of shank blocks. I think there are possibilities in the use of this class of mine scrap, but there are weaknesses to be overcome before the full benefit can be obtained, and I will give the results and costs of this section of lining later."

Dr. Calderott's contribution does not lend itself to discussion in my reply, as I find myself in complete agreement with him on the points he discusses. His remarks on the short tube mill for use on the Rand afforded an excellent opportunity for its advocates to put their case before members, since this has not yet been done.

Coming now to Mr. K. L. Graham's contribution, we have to thank him for placing his valuable data before the Society. In general, the results of experiments conducted at the City Deep confirm those on the Gold Fields mines, and they emphasize the value of the scoop discharge for improved crushing capacity and mechanical efficiency. In the case of the City Deep trials the pulp fed to the spread tube-mill to serve for the purpose was coarser and contained appreciably less -200 product than did that of the mines mentioned in the paper. Small amount of fine (-200) product in the feed is desirable and makes for good crushing result, but the grading analysis of the final pulp entering the cyanide works is the important test, and this must be constantly kept in mind. Thus at the Robinson Deep, although the screen in use at the stamp-mill is coarse, resulting in a coarse screen pulp, it is found clear in obtaining a final pulp of 85% to 87% of -200 the feed to tube-mill becomes debat d with fine material from the tube mill return pulp, which must be accepted. In other words, in striving for a coarse underflow of cones to feed the tube-mills there is a great liability to coarse overflow to the cyanide works.

According to Mr. Graham's tabulated results of trials at the City Deep, the crushing capacity is appreciably increased with increase of diameter of the scoop, but the mechanical efficiency, *i.e.*, tons of -200 product per horse power is best with the 21 in. diameter scoop.

The President: The discussion on Mr. Bowling's paper has not been very great. In his reply he has brought out a few more points which add to the value of the original paper read before this Society.

NOTES ON THE PRACTICAL TESTING OF WORKING CYANIDE SOLUTIONS.

By Edward H. Croghan (Member).

(*Printed in Journal, April, 1915.*)

REPLY TO DISCUSSION.

Mr. E. H. Croghan (*Member of Council*): I thank Messrs. H. A. White, J. Watson, J. Gray, and Professor Stanley for their remarks, but I regret that of the many others interested in cyanide work no one has contributed to the discussion.

I am glad to note that Messrs. White and Watson are to some extent in agreement with myself that there is room for improvement in the matter of recording results, and perhaps at some future date the adoption of such terms as CN^1 and OH^1 may become a practical proposition.

Mr. John Watson refers to the strength of silver nitrate I employed, which is just half the strength of that used in probably 90% of the Rand extractor houses. This is chiefly a matter of personal convenience, since titrating 50 cc. cyanide solution (instead of 100 cc.) with a silver nitrate solution of half the usual strength, obviously gives the same result.

Regarding Mr. Gray's criticism, no one appreciates more than myself the sterling value of Mr. G. W. Williams' paper, to which I made reference, and I would refer Mr. Gray to the opening paragraph of my notes, from which it is obvious that my paper was intended to illustrate, firstly, the great diversity of results that may be obtained in testing working cyanide solutions, and, secondly, to promote a discussion as to the most suitable procedure which should be adopted on this field."

In the first respect I believe I have succeeded, but in the second I must admit disappointment.

With regard to the correction by Mr. Gray of the page number of my reference to Mr. Williams' paper, this is a matter for our Secretary to look into, as it is evident his volume and mine have different indexing.

NOTES ON THE TREATMENT OF ANTIMONIAL GOLD ORES FROM THE MURCHISON RANGE.

By H. R. Adam, B.Sc. (Member).

(Printed in Journal, June, 1915.)

DISCUSSION.

Prof. G. H. Stanley *(Past-President)*: In connection with this paper it may be of interest to have some figures showing results of tests in my laboratory on some of the ores mentioned by Mr. Adam, the primary object being, however, gold and not antimony extraction.

Ore of Class II.—A sample assaying 26 dwt. was crushed through 40 linear mesh screen, pulped and passed over an amalgamated plate. Extraction was only 10 %. After making slightly alkaline with lime and grinding with mercury in a mortar extraction improved to 35 %, while if slightly acidified the extraction was 38·9 %.

Another portion of the same sample was concentrated on a Wilfley table (after passing the amalgamated plate). The concentrate assayed 200 dwt. and tailing 6·2 dwt. The tailing was classified and the sand submitted to cyanide treatment. The original value was 6·6 dwt., and after four days' treatment with 0·24 % solution, the residue assayed 1·4 dwt. and the cyanide consumption was over 10 lb. per ton.

In another experiment on the same lines, amalgamation was omitted. The concentrate from the Wilfley amounted to 6·85 % and assayed 225 dwt., the sand amounted to 76·5% and assayed 8·2 dwt., and the slime to 16·55%, assaying 10·75 dwt. The sand was leached as before for four days, after which the residue assayed 3·1 dwt. with cyanide consumption again over 10 lb. per ton. The slime was agitated with 0·1 % KCN solution for eight hours: residue assayed 6 dwt. and cyanide consumption was nearly 6 lb. per ton.

In another cyanide test, using the amalgamated plate, but omitting concentration previously, the sand residue assayed 14·5 dwt. with cyanide consumption of nearly 12 lb. per ton, while the slime residue assayed 5·9 dwt. with cyanide consumption about 4 lb.

Still another test was made by passing the 40 mesh pulp over the plate, then passing the tailing through a 60 linear mesh screen and amalgamating by grinding in a mortar. Sixty per cent. extraction resulted: the tailing classified and cyanided as before gave sand residue assaying 1·5 dwt. and slime residue 3·9 dwt., cyanide consumption being 12·35 lb. and 5·45 lb. Another portion of the 40 mesh pulp, after the amalgamated plate, was ground to −200 and agitated with cyanide as slime. The residue assayed 17·25 dwt. and cyanide consumption was 5·3 lb. per ton.

Another sample after amalgamation and concentration was treated with dilute sulphuric acid to remove oxidized copper compounds, thoroughly washed and oxidized with dilute permanganate solution and cyanided. Residue, however, assayed 6·4 dwt. and cyanide consumption was 7·3 lb.

Another sample crushed to 60 mesh, was amalgamated and concentrated and the tailing ground with mercury in a mortar, then classified into sand and slime. The sand assayed 6 dwt., and after leaching for five days, showed a residue of 1·9 dwt. with cyanide consumption of 5·7 lb. Slime residue after eight hours' agitation with 0·1% assayed 5·6 dwt. (The concentrate amounted to 9·9 % assaying 200 dwt.)

Obviously, then, the ore calls for some method of treatment by concentration, about 10 % of concentrate carrying about 77 % of the gold, while after close concentration, cyaniding the sand at least is apparently feasible, though the slime appears rather hopeless, and consequently fine grinding to obtain a high amalgamation extration appears out of the question. On the other hand, if the crushing is not fine, say 60 mesh, clean concentration is impossible, and it was the knowledge that in some cases flotation methods were obtaining good results on mixtures of fine sand and slime which led me to suggest to Mr. Adam that an investigation of that method of treatment would be useful.

Ore of Class IV.—Similar tests were made on ore of this class assaying 13 dwt. It is very difficult to amalgamate, even grinding the slightly alkaline pulp with mercury only extracting 8 % and badly flouring the mercury. After amalgamation and classification, the sand (76%) assayed 12 dwt. and after cyaniding as before gave a residue of 8·25 dwt. with cyanide consumption of 5·8 lb., while the slime residue assayed 2·4 dwt. with cyanide consumption 2·8 lb.

Another portion was concentrated on the Wilfley, 4·85 % of concentrate being obtained of assay value 179 dwt., also 79% sand, assaying 4·1 dwt., and slime assaying 8· dwt.

By cyaniding as before, the sand residue assayed 1·2 dwt., and slime residue 1·05 dwt., with cyanide consumption of 1·8 lb. and 3·7 lb.

On this ore, then, concentration followed by cyaniding extracts about 90 %.

I need hardly point out that the cyanide consumption would almost certainly be much less in practice, but would certainly be high, and here again better concentration and removal of interfering minerals would doubtless improve the cyaniding results. The concentrate obtained from both classes of ore was tested by amalgamation, cyaniding and chlorination with very poor results, and would have to be treated by a smelter.

These results serve to illustrate the difficult nature of the ores, and while in some places payable tonnages exist of ore rich enough to yield profits in spite of the somewhat expensive treatment necessary, a cheaper and more perfect method would have a great effect on the development of the district, and I trust that Mr. Adams's work will contribute to that end.

Mr. H. A. White (*Vice-President*): I should just like to ask Prof. Stanley, with reference to these larger consumptions of cyanide he refers to, if he has made any effort to determine in what form that consumption takes place? Whether it was in such a form that some cheap regenerative process might reproduce the cyanide, or whether in such a form that cyanide might be available for further extraction of gold?

Prof. G. H. Stanley (*Past President*): The cyanide solution was found to contain a little copper, but, apart from that, it had a good deal of sulphide and ferrocyanide. So that the solutions became very foul indeed, and were not suitable for further use. We have even found that in 24 hours' treatment we obtained a better extraction than after treatment for four days.

Mr. H. C. Boydell, B.Sc. (*Member*): The author is to be congratulated on his work in connection with this subject, as such ores have always been the *bête noir* of the metallurgist. As has been mentioned by Mr. E. M. Weston, on the Hillgrove mining field, near Armidale, New South Wales, gold ores with a varying antimonial content have been mined for many years past. An interesting feature about this field is the unusual association of the auriferous stibnite and senarmontite, both of which, in the past, were mined at the surface by small co-operative parties, the ore in most cases being roughly concentrated in hand jigs. At the time of my connection with the field (1898-9) the Sunlight Company mined and treated an ore with an average stibnite content of 3 to 4 % in the following manner:—The ore after passing through a rock-breaker went to 40 head of stamps with the old-fashioned punched screens and the usual low discharge. Inside amalgamation was carried on and the pulp from each mortar box passed over amalgamating tables 20 ft. in length. The whole of the pulp after amalgamation was concentrated on eight Frue vanners. The extraction obtained was from 75 to 80%, and the loss of mercury was not much higher than that usually met with in stamp batteries. The tailing, averaging 1½ dwt. gold per ton, were not further treated (the cyanide process being then not extensively used in Australia) and owing to the situation of the mill they could not be saved for future treatment. In the mining work occasional patches of almost pure stibnite were met with, and such increases in the antimonial content of the ore were dealt with by hand-picking both underground and at the battery.

The concentrates from the Frue vanners averaged 2 oz. of gold per ton, and after drying were sold on assay value to the smelting company then operating at Dapto. There was a small penalty of a few shillings per unit of antimony deducted from the gold values of the ore, in addition to smelting charges.

At the West Sunlight Company an ore of similar character was treated in practically the same way. Owing to the better situation of the battery, the tailings were saved in this case, and have since been successfully cyanided.

Both these companies have ceased work for some time owing to the ore becoming too poor in depth.

The Eleanora G.M. Co., operating on the opposite side of the Baker's Creek Gorge, mined an ore containing much more antimony. Near the surface it was pure stibnite, and was worked purely as such. As depth was attained the stibnite decreased, and at the time mentioned averaged 10 %. The method of treatment was the following. The ore after reduction in rock-breakers passed to 60 head of stamps with punched screens and low discharge. No inside amalgamation was practised, and the

place of the usual amalgamated copper plates was taken by blanket "strakes". The pulp leaving the latter was concentrated on Frue vanners. The "blanketings" were amalgamated in Berdan pans, and the overflow from these concentrated on another set of vanners. The extraction averaged 70 to 75 of the gold content of the ore. The tailings were stacked and have since been cyanided. It is most probably the treatment of these that was described by W. A. Longbottom in the paper referred to by Mr. E. M. Weston.

This company has also stopped working for the same reason as the others, and at the present time, so far as I am aware, the only mine operating on any scale at Hillgrove is the Bakers' Creek Company, which, at the time I speak of, was treating an ore similar in antimonial content to that of the Sunlight Company. At the risk of being considered digressive, I may mention that the Baker's Creek Mine, in one part of its workings, has always been subject to what, on the Rand, are called "air blasts", a local phenomenon that has been dealt with in one of the papers of the Geological Survey of New South Wales.

Costerfield, near Bendigo, in Victoria, has had a very similar mining history to Hillgrove. At the surface the ore was purely antimonial and the stibnite decreased as depth was reached. The only operations being carried on in 1910 (and quite possibly also at the present time) were by an English company. The treatment consisted of crushing and concentration without amalgamation, the concentrates being shipped to Great Britain for treatment by the Macarthur-Forest process. Attempts have been made from time to time to cyanide the large accumulation of old tailing at Costerfield. One of the most recent was that of the Costerfield Tailings Co. The procedure followed was mixing the tailing with salt, roasting in Merton furnaces for the purpose of volatilizing the antimony, and then leaching with cyanide solution in the ordinary way. Owing to the poor extraction obtained the work was abandoned. The operations in this plant, owing to lack of proper precautions in the treatment of the zinc slime with acid, resulted in one of the very few known cases (if not the only case) of poisoning by antimoniuretted hydrogen.

Some reference has been made to the difficulty of disposing of stibnite and antimonial concentrates, and more particularly in obtaining any payment for the gold content of these materials. Antimony smelting was carried on in Melbourne, and antimonial concentrates were purchased by the chlorination works in Bendigo and Ballarat. Possibly enquiry in these quarters would open up a market for the South African antimonial products.

The author refers to difficulties experienced in the assaying of the Murchison antimonial ores, but does not say whether these are due to the stibnite or to oxidized antimony minerals. If the presence of stibnite has caused him trouble in his gold estimations, I cannot understand it, as some years ago, when assaying such ores, I had no difficulty in eliminating the stibnite either with hydrochloric acid prior to fusion or with a proper admixture of nitre to the charge in the pot fire assay itself. The results obtained were sufficiently good to form the basis for the disposal of considerable quantities of concentrate.

It would be interesting if Mr. Adam would mention whether the oxidized minerals, such as senarmontite, valentinite, cervantite and kermesite are met with in the Murchison Range, and also what percentage of arsenic, if any, is associated with the antimony.

From experience obtained elsewhere in the treatment of such ores, it would seem to me that not much difficulty should be met with in the treatment of the Murchison ore by crushing, concentration and cyaniding of the tailings, together with amalgamation where the latter operation is possible. The chief difficulty would appear to be in the disposal of the concentrate and the obtaining of payment for the gold content. South Africa, at the present time, with its lack of custom smelting and chlorination works is a poor field for a mining company carrying out concentration work.

RECENT INVESTIGATIONS ON DUST IN MINE AIR AND THE CAUSATION OF MINERS' PHTHISIS.

By James Moir, M.A., D.Sc., F.R.S.S.Afr.

(Printed in Journal, July-August, 1915.)

DISCUSSION.

Mr. S. Nettleton *(Member)*: In one of the most valuable papers presented to this Society for a considerable time, Dr. Moir has contributed information worthy of

analysis by every mining engineer on the Rand. The outstanding features of the investigations appear to be that when proper appliances are used for laying the dust, rock drills need not be regarded as a source of danger, and that it is only particles of silica of less than 3 or 4 microns diameter which may be regarded as dangerous to the human lungs. Information on the character of air found in stopes a few hours after blasting has taken place would be of considerable interest to the mining community. In the table on p. 7 of the Society's *Journal* the air samples marked D. are described as the result of "dry blasting of a cut", and presumably these are somewhat akin to what is obtained by breaking rock from benches by machine or hammer stoping. From the formula quoted in Dr. Moir's paper on rate of falling of quartz particles in a still atmosphere, the time taken for settlement of dust through a distance of 5 ft. or 150 cm. is apparently as follows:

Particles of 1 micron diameter—51 hours.
Particles of 2 micron diameter—1 hour 23 minutes.
Particles of 3 micron diameter—37 minutes.
Particles of 4 micron diameter 20 minutes.

According to tables on page 7, five minutes after blasting a dry cut the number of particles present in each cc. of air was 86,000, 15 minutes after blasting the number was 19,000. Assuming that the larger and heavier particles are the first to settle, there would apparently be in the residue a large increase in the percentage of the fine dust. Yet according to column 6 the percentage by weight of the very fine dust decreases from 93·5 to 80 , and the percentage by number is practically the same. Do the figures given in columns 6 and 7 refer to percentages of very fine in total amount of dust present in each instance?

As an efficient water-blast will reduce the amount of fine dust from blasting of a cut to what may be encountered on the surface near a mine dump in Johannesburg, the problem of dealing with dust in development drives has apparently been solved.

Attention should be given to the condition of the air in stopes where dry blasting is general, where there are no water-blasts, and where the air currents are probably just sufficient to prevent the dust settling for many hours. On some properties the amount of shovelling and tramming done on night shift assumes considerable proportions. Is there any information available regarding the condition of air in stopes within 12 hours of the principal blasting period of the mine? The bulk of the rock drilling by machines is carried out between 8 a.m. and 3 p.m. This introduces large quantities of pure air into the mine, and it does not appear to be desirable that the air in stopes and other workings should be usually examined under the most favourable conditions.

In a recently issued report of the Government Mining Engineer it was pointed out that "there is no single measure that can be taken which is more conducive to healthy conditions underground than the abolition of the double shift, and it is to be hoped that those mines which are still on this system will see their way clear to dropping it."

Information as to amount of labour engaged on night shift relative to total number of underground workers on the Rand would be of great interest, but for obvious reasons there would be difficulty in obtaining details on this point.

Dr. Aymard, an Associate of this Society, in referring to the phthisis question 15 months ago, said that there was no evidence that the efforts made during the past few years have been successful in reducing the steady progress of the disease. Can Dr. Aymard's remarks be controverted? There should be available to members of this Society and the general public detailed information upon this point.

It should not be necessary for workers to go underground until risk of inhalation of particles of silica has been reduced to a minimum, and as the major portion of the dust cannot be dealt with by water-blasts and similar appliances, it should be drawn out of the mines. In a district where most of the mining is at depths exceeding 1,000 ft. from the surface, natural ventilation cannot be regarded as satisfactory, and means for circulating large quantities of air throughout the workings should be regarded as necessary to the equipment of a mine as a winding engine. Small fans scattered here and there throughout mine workings rarely give satisfactory results, and there should be no tinkering with a question which has so important a bearing on the future of Rand mining as ventilation. Every mine official should be compelled to take an intelligent interest in the subject, and in a few years it will not be suggested on a public platform without contradiction that in many cases the ventilation of the mines and

the underground arrangements were a disgrace to those who controlled the principal industry of South Africa.

Mr. F. J. Pooler (*Associate*). This paper is one of exceptional value, not only to us here, but to the Empire as a whole in these days. But some of Dr. Moir's results are very disquieting. Leaving aside the dust question for the moment, such phrases as: "A water-blast was supposed to be in use, but sometimes was out of action or too small to be of any use"; "The maximum CO found was 0·11 %"; "A completely unventilated area of a certain mine gave CO_2 1·046 and CO 0·12 on the average"; "The worst gas encountered in this set gave CO_2 1·40, CO 0·20, O_2 – 17·80 %": give one furiously to think. It is fairly obvious that despite the efforts made, and we are all agreed that progress has been made, things are still very bad in some of the mines. Why should the CO be so high in these odd cases? How is it that the water-blast "was sometimes out of action or too small to be of use"? Someone is to blame—the Government Inspectors cannot be everywhere at once, and surely now that the peril of the thing is a by-word here, we of the public who are outside the mining industry ought to feel assured that the Government is insisting on all possible precautions being taken, and seeing to it that these are carried out in the mines. It seems that the fact that miners' phthisis has become a plaything in the baneful cockpit of party politics has a lot to do with the fact that researches such as these, necessary, valuable and imperative as they are, are only now being undertaken, and are revealing dangers hitherto only hinted at, dangers that should have been abolished years ago.

One looks for rays of hope in this paper. They are there, but are almost smothered by Dr. Moir stating with regard to the laying of the dust as far as the "very fine" particles are concerned that "the result was satisfactory, and showed *when the laying devices were in order and conscientiously used*, there was no predominance . . . . the percentage being no higher than in the original untreated air." I do not quite know what the author means by original untreated air, but it does not look very promising that he has to make this confession. Following this, we read that "When the point of the drill is wetted by having a jet of water directed into a 'down'-hole, the amount of dust getting into the air is only 2 or 3 % of what happens during dry drilling," and then "it is surprising to find how little variation there is in the percentage by weight of 'very fine' particles, they constitute about two-thirds of the total weight of dust in all sorts of mine samples." I must apologize for my obtuseness if I am wrong, but I cannot connect these statements in any other way than to mean that though the actual quantity of dust has been very considerably decreased, the improvement in getting rid of a bigger percentage of the "very fine", which really matters, as compared with the coarse, which apparently is not of such importance, is still a matter for the future. Presumably the question Mr. Weinbren asked last month supplies the answer to this.

When this subject was being discussed by us some three years ago, a medical member of the Society made the statement that men in the mines' crusher-houses and on the Municipal road-stone crusher-plants were also contracting the disease. It is comforting to read that Dr. Moir states that the former, at any rate, are comparatively immune.

The research work of this paper shows that a considerable and notable advance has been made in experimental determination of dust, but apparently much has to be done in the mines yet before the dust evil is overcome. Suggestions as to further advance in this, the all-important direction, did not fall within the scope of this paper, but those of us who are among the author's admirers look confidently to him and the engineers, among whom are some of the most active members of this Society, for a speedy and permanent improvement of the conditions.

Might I ask why the author divides Haldane's figure for air breathed per minute by four? Seven thousand cc. per minute seems rather low even for light work. Also how it is though "dry-drilling is, of course, illegal," that the author is able to give us details of dust during this illegal process? Presumably illegalities are legalized in the interests of scientific research! May I also ask why "rounded tetrahedra" are assumed as the shapes of the smallest particles when "polarization of particles smaller than 15 microns is too faint to be of any use"? If the particles of this size and less are more nearly spherical—which seems reasonable to suppose—there will be less of them per mgm. than the table shows. It is a

tration to the author's desire not to obscure the horror of the results that he has chosen too small a size and so increased their number.

Mr. H. R. Adam (*Member*): The first part of Dr. Moir's valuable paper leaves little room for criticism, and it is good to know that as a result of the work of Dr. Moir and his colleagues, future progress will be measured not by guesswork but by accurate analysis. The difficult and laborious analytical work, along with the very fine micro-photographs, gave abundant evidence of the nature of the problem, especially when the author's remarks towards the end of the paper that "the underground man gets in addition an occasional dose of CO and of tubercle germs" is also kept in view. It seems a pity, therefore, that the results obtained in investigating the efficacy of the methods in use of "laying" the dust have been presented in such a very synoptic form.

As far as drilling is concerned, Dr. Moir's results show that "wetting" is effective when actually performed in the hole, *i.e.*, that the "very fine" dust can then only emerge in the form of slime. It is, however, well known that the dust caused by drilling is insignificant compared with that caused by blasting, and the only results given on samples taken after blasting deal with the air after blasting the "cut" with and without "best water blast". On the matter of the air in stopes, say, three or four hours after blasting when tramming commences, not a single result is given. In fact, admitting that the investigation shows that the dust from drilling and development can be dealt with, it does not touch the much bigger and more difficult question of the air in stopes, and in these circumstances the statement made in italics that "when the laying devices were in order and conscientiously used, the result was satisfactory" is extremely misleading.

Since no details of the effects of different types of "laying devices" are given, discussion on this point does not apply to the paper. However, since the "laying" of the "very fine" dust depends on the "wetting" of the particles, the question may be asked whether the committee has any data on the effect of additions such as sugar (in the form of molasses), soap, alkalies, etc., to the water in order to increase its "wetting" power. In this connection an abstract on the beneficial results of humidifying mine air by means of steam, contained in that number of the *Journal* in which Dr. Moir's paper is published, is of great interest.

Under the paragraph referring to the rate of falling of quartz particles, the statement is made that the smallest particles formed in dry-drilling and by blasting cannot possibly settle in the eddying atmosphere of the mines. The trouble is that most mines have many spaces, such as back stopes, old stopes, dead ends, etc., where the only eddying currents that are formed are due to blasting and then "very fine" dust which had settled is disturbed again.

The President: I understand that Mr. Hilditch Smith is making some experiments in connection with respirators, and we will probably have the results at the next meeting.

CYANIDE CONSUMPTION ON THE WITWATERSRAND.

By H. A. White (Vice-President).

THE PREVENTION OF HYDROLYSIS IN CYANIDE SOLUTIONS.

By H. M. Leslie (Member).

(Printed in Journal, September, 1915.)

DISCUSSION.

Dr. W. A. Caldecott (*Past-President*): Referring to Mr. White's paper, the thanks of members are due both to the author for his masterly handling of the mass of figures resulting from Mr. Meiklejohn's able laboratory investigation into this subject, and also to the Transvaal Chamber of Mines, who provided the funds required to carry out the research.

In the second paragraph of the paper a clerical error occurs in the statement that a million pounds worth of cyanide are used annually on the Rand. This is about double the actual figure, according to the Ninth Annual Statement of the Trade and Shipping of the Union of South Africa, issued by the Department of Customs and Excise, Capetown. In this Bluebook it is stated that the import into the Union of South

Africa for the year ending 31st December, 1914, was as follows:—

	Quantity. lb.	Value (in country of origin). £
Cyanide of Potash	50,734	1,887
Cyanide of Sodium	10,084,868	372,430
Total	10,135,602	374,317

Assuming the present cost of sodium cyanide delivered on the mines to be 1s. per lb., and that the imports for the current year are approximately the same as for 1914, the total yearly cost of cyanide used in South Africa is now about half a million sterling.

With reference to the decomposition of an aqueous alkaline cyanide solution, the hydrolysis* referred to by the author consists in the inter-action between dissolved cyanide and water, resulting in the formation of hydro-cyanic acid. This in turn readily undergoes change with formation of brown humus-like and other products. Assuming the production of insoluble humus-like substances through the decomposition of HCy, these would be discharged as a solid in the residue and not in solution. This would constitute a loss of cyanogen, or rather of nitrogen derived from alkaline cyanide, which is difficult to estimate owing to such compounds forming a very minute percentage of the total weight of the solid residue. Decomposition by atmospheric carbonic acid likewise results in the liberation of HCy, and hence the presence of adequate free alkali in solution should lessen both these decompositions of the alkaline cyanide by causing the Cy radicle to remain combined with an alkaline base, instead of forming the readily volatile and decomposable HCy compound. In view of the important function of protective alkali in working cyanide solution, further knowledge of the relative protective efficiency of NaOH and $Ca(OH)_2$ (calcium cyanide being rather an unstable body) would be valuable, and also whether the free alkaline strength required is a constant, or varies with the temperature and the amount of free cyanide present. It may be noted that since sunlight affects the rate at which gold dissolves in a cyanide solution,† it may also influence the rate of decomposition of cyanide.

In reference to the reaction between 0·10 KCyS solution and lead-coated zinc shavings, I might suggest that the author should repeat the experiment, but with a solution containing also 0·10 KCy, as the formation of ZnS is possibly dependent upon the reduction of KCyS by nascent hydrogen set free in the presence of cyanide, zinc and water.

It might also be of assistance to members if the author in his reply indicated what in his opinion is approximately the relative importance, expressed in percentages of the total loss, of the various causes of cyanide consumption under ordinary working conditions in a Rand cyanide plant.

Mr. H. M. Leslie (*Member*): The only point I desire to touch on at this stage is to correct the impression which Mr. White seems to have gathered from my remarks in connection with the protection afforded by alkali in my Modder B. tests. All I have said is that the protection afforded by this means is not complete, and that from an economic point of view, taking into consideration the cost of the extra alkali required to keep the liquor up to this high standard, together with the loss of cyanide which is still shown to go on, that so far as £ s. d. is concerned there is nothing to be gained. Further, in this connection in the Model Plant Brown vat test to which he refers, the cyanide regenerated does not seem to have been taken into account in the tests given as showing the "Loss in Hours, Pounds per ton Slime" (Table O.), which seems to be amply confirmed by the figures following, which state that in tests referred to 53·8 of the total cyanide lost was recovered in the apparatus used.

PERSISTENCE OF ORE IN DEPTH.

By T. A. Rickard, M.Inst.M.M.

(*Printed in Journal, Dec., 1914—Jan., 1915.*)

REPLY TO DISCUSSION.

Mr. T. A. Rickard: In acknowledgment of the compliment of the further discussion before the Chemical, Metallurgical and Mining Society of South Africa, I beg to reply to those who, at Johannesburg, have commented on my paper.

Mr. E. M. Weston is disinclined to accept the logical consequence of my argument against indefinite persistence of ore in depth. He imputes to me an undue

* J. E. Clennell, "The Cyanide Handbook," p. 105.

† See this *Journal*, Vol. V., p. 215, 1905.

tendularity with the younger beds of Western America, and charges me with "neglecting almost entirely a very large and important class occurring in the older rocks". This is wholly incorrect. In my paper I discussed particularly the distribution of ore in the pre-Cambrian of Nova Scotia and of South Dakota, as well as the Archean of the Mysore and the Minas Geraes goldfields. The records of economic geology will show that I may fairly claim to be excused from local, or even regional, prejudice, for I have examined and described ore deposits in the pre-Cambrian of Western Australia, Broken Hill, and Nova Scotia, the Cambrian of New Zealand, the Lower Silurian of Ballarat and Bendigo, as well as of Tasmania, not to mention numerous districts in the United States that afford examples of ore deposits in rocks older than those of the Witwatersrand.

As to my alleged neglect of "the effects of denudation in truncating orebodies", it would seem as if Mr. Weston had not read the original paper. Indeed, I am compelled to conclude that my critic had only heard the introduction read by the Secretary and had omitted to read the paper itself, which had been published in the preceding *Journal* of the Society. I recognise, of course, that it is not unusual to offer comment on a technical paper that one has not read. It adds to the possibilities of discussion. In my paper, of course, I gave particular attention to the denudation idea, as illustrated by the Nova Scotian and Californian gold veins. In fact, I hope to be forgiven for saying that the whole tenor of Mr Weston's argument is that of one who has not listened to the evidence on the other side, contenting himself with a repetition of theories and ideas that are now obsolete. To call them "old-fashioned" may be disarming, but it does not make them convincing. Clunes, Ballarat, and Bendigo, of course, are well known to me. To describe the orebodies of those districts as "the waste products of the rearrangements of the elements contained in the minerals making up the rocks themselves," is mere abracadabra, a mumbling of words that have no meaning. That "these changes" are apparent at the greatest depths attained at Ballarat or Bendigo may be true, for, as we are not told what those changes are, Mr. Weston can defy anyone to prove the contrary. But if he means to say that the deepest mine-workings at Ballarat and Bendigo do not show decided impoverishment—a general and unmistakable impoverishment in depth, then he will have told the members of a scientific society what is contrary to known facts. In its issue of June 10, 1912, *The Bendigo Advertiser*, the capable exponent of the local mining industry, acknowledged that the "shallow ground" theory had been vindicated. The editor said: "In almost every, if not in every, gold-mining area in the world the experience is that as considerable depths are attained, the gold is scarcer. The depositions of quartz may be more or less or practically the same as in the upper levels. Opinions vary on that point, but most authorities agree that as the earth is penetrated to great depths gold is less plentiful. Bendigo investors and speculators know from bitter experience the absolute truth of the latter point, as far as this field is concerned."

Undoubtedly the term "ore-shoot" is occasionally misleading in suggesting a definiteness of shape and direction that rarely obtains in nature, but it is a term that is used sparingly in my paper, therefore the implied criticism is beside the mark. As to Rhodesia, it has suffered not so much from denudation as from over-capitalization—and irresponsible talk of persistence in depth. "The younger orebodies of the Sierras and Cordilleras" may have been exposed to excessive denudation, but the exploitation of them has suffered as much from blind disregard of facts. As to the denudation, I have shown, in the case of Nova Scotia, for example, that the erosion of the country-rock does not prove the equal erosion of the gold-bearing ore, not until it is demonstrated that the gold-bearing ore extended formerly to the surface of the eroded terrain. Mr. Weston and others take it for granted that if an ore-bearing vein is found, for example, in a series of quartzites and slates that has been cut down by erosion to a depth of four or five miles, that therefore the vein in this country-rock and the ore in that vein must have been denuded to a vertical extent measurable by 20,000 ft. or 25,000 ft., and inferentially that the ore must persist to an equal depth. The assumption is unwarranted unless it can be shown that the ore in the vein is as old as the country-rock. This they cannot prove. If, on the other hand, I find that mine-workings only a few hundred feet deep suffice to penetrate below the ore, and if I see evidence indicating that the richness of the ore diminishes at a pitifully short dis-

tance below the present surface, then I say that their talk of former persistence of the ore for 20,000 ft. to 25,000 ft. upward, or of probable persistence 20,000 ft. to 25,000 ft. downward, is non-science, vulgarly called nonsense.

In regard to the impoverishment, or otherwise, of the Main Reef series, it may seem improper for me to speak positively to the members of a technical society at Johannesburg. In my paper I have quoted engineers not without honour in their own bailiwick. Mr. Horwood's theories may, or may not, be convincing to Mr. Weston, but unsupported generalizations have little weight as against the painstaking research of a real student. Mr. Horwood has contributed to our knowledge; Mr. Weston to our conversation. A piece of dike, impregnated with gold, from the Ferreira mine, for example, is of more importance to this discussion than the sweeping statement that "it is most plain that the dikes have utterly failed to bring about any general or extensive mineralization of the enclosing rocks". I have held that piece of dike from the Ferreira in my hand; it contains vein-quartz in which native gold is visible, and it came from the dike where the latter was above the banket, in the way illustrated by S. J. Truscott on page 111 of "The Witwatersrand Goldfields", although there the dike is shown as seen in the Worcester mine. This specimen, however, came from the Ferreira and from the dike illustrated on page 112 of the same book. The dike is between 4 ft. and 5 ft. thick, it crosses the banket, with effects similar to those noted by Professor Truscott as having been observed in the Worcester mine, namely, where the dike was above the banket the latter was richest along its upper portion or hanging wall; and where the dike was under the banket, the latter was similarly enriched along its lower or foot-wall side. Dikes accompanied by veins of quartz are numerous on the Rand; also they influence the richness of the banket. In other parts of the world it has been proved that the deposition of gold in lodes has been effected by thermal waters in the wake of dike-intrusions and other forms of eruptive activity. On the Rand the familiar signs of such agencies are not wanting. The assertion that the Rand is unique is unconvincing to men of wide experience.

The question of the vertical distribution of gold in the Main Reef series is not likely to be solved so long as the prosperity of Johannesburg is in danger of being injured by the facing of a possibly unpleasant fact. "Let sleeping dogs lie" is the not unnatural motto of the big companies that have no wish to see their shares depreciated. Mr. B. Meyer's mention of a commission to make an investigation is enough to cause a panic on Throgmorton Street. Mr. Marriott came to the meeting at which my paper was discussed and exhibited a stope-section of an important group of mines, on this section the grade of the ore was indicated by various colours, but the upward limit recorded was far below the average of past production or even of present production in two or three of the most notable properties. Mr. Marriott regarded this information as belonging to his own clients, and he presented it to the meeting only as an unusual courtesy. As a matter of fact, it was information to which every shareholder in the companies controlled by his clients is entitled.

Mr. Meyer speaks with a moderation that is disarming. I wish only to make the point that the vertical depth of the ore exposed in the Far East Rand is diminished relatively by the flat dip of the lode. As far as my observation goes, the persistence of ore is affected not by distance from surface along the dip, but by vertical depth.

I desire to congratulate Mr. L. W. Mac on a constructive contribution to this discussion. Undoubtedly the analogy between Pilgrims Rest and the Rand is most suggestive. However, even if rich ore has been found in the Clewer lode at a distance of six miles from the outcrop, it must be remarked that the difference in vertical distance is relatively trifling, in other words, no proof of persistence in depth is afforded thereby.

Professor Cellier's comment is enriched with humour befitting the discussion of a subject that has been obscured by solemn asservations having little support in the vicissitudes of the miner. Yes, indeed, the sampler appreciated facts long before the chairman and his academic friend, the field geologist, learned to look the truth in the face. I desire to thank Dr. Caldecott for his friendly invitation, so much in accord with my wishes.

Mr. Hay agrees that it is "universally recognized and admitted" that "pay shoots do not continue down indefinitely." He has overlooked Mr. Weston. I agree that "no amount of theorizing will ever tell us much." In my little paper I tried to

proves that a particular theoretical generalization was wrong, and I quoted facts in support of my contention. But Mr. Jenkins thinks it "rather a question of geology than of mining profits", although the definition of "ore" with which I started is definitely tied to the economics of mining rather than the imaginations of geology. Mr. Heathcote wants us to start our comparisons of richness from "a focus of deposition" and from this assumption he passes swiftly to the conclusion that "it may be taken as certain that some deposits extend to over 12,000 ft. in vertical depth and values [whatever that may be] will be as good as any yet found." All that I care to say is that a large body of ascertainable facts points to the contrary.

Mr. Egerton Wood says that the original paper "is rendered more or less futile by the confusion in the use of the term ore". He complains that I used the term in a sense different from that used "by perhaps 90 of the members of the Institution", and presumably of the Society also. What may be the definition that is acceptable to the 90 I do not know, but I feel assured that it would be utterly useless for the purpose of a scientific discussion. Try a debate on persistence of "values" and "unpayable ore" and see where you land—in the limbo of absurdity and the morass of pseudo-scientific verbiage. In all fairness, I ask Mr. Wood and the others to recognise that no discussion is profitable until the principal terms of the question to be discussed are carefully defined. I defined "depth" and "ore" so that all of us might know what was the particular question before the Institution and the Society. If "ore" is to be defined as vein-matter or lode-filling, then I admit at once that I know but little about the persistence of it in depth, and I care less. Such an enquiry is not related to the economics of mining; it is a purely geological question as to which only a theoretical answer is possible, because the miner will never sink so deeply as to obtain the necessary evidence.

Post Scriptum.—Since my reply to the discussion before the Institution was transmitted, I have read the later contributions by Messrs. James Park and P. V. Robbins. Before closing finally, I trust my rejoinder to the comments of members in London and in Johannesburg, I desire to express my keen appreciation of the criticisms and suggestions of the two members just mentioned. Professor Park is known as a keen geologist and a clear writer on the economic phase of geology. His criticism is helpful. Indeed it is difficult "to find a generalization broad enough to satisfy all the facts". In controverting a generalization that my own observation has led me to consider fallacious—namely, that of indefinite persistence of ore—I could not avoid seeming to set up an opposite generalization. My point of view is that of the miner technically trained, namely, the mining engineer. That is why ruminations concerning the continuity of vein-filling having no economic value did not come within the purview of my paper. Professor Park writes concerning "the probable depth to which profitable ore will descend". I thank him for bringing the discussion back to its text. But he also dwells upon the truncation of veins by denudation, and says that in certain areas "it is almost certain that the veins now existing in these areas are only the rootlets of what may have once been great veins". It is curious how the geologists insist upon this idea; even the best of them. Yet the pretty theory has been killed by the brutal facts of mining. The Nova Scotian example is the clearest. There erosion has been tremendous; a great thickness of rock has been removed; the geologist says that the small veins of gold-bearing quartz formerly extended through this great thickness of rock; that the gold veins were eroded; that their gold was carried by glaciers and other agents of erosion into the sea; that, in short, the veins to be seen at the present surface are only the roots of those that reached to the ancient surface, many miles higher. From this exercise of the constructive imagination it is not far to a vision of ore-bearing veins that persist for miles below the present surface. Why should the present surface cut the veins at a particular horizon where they are exceptional in richness or poverty? All of which would be splendid if the facts could be wiped out. They refuse to be deleted. Again I say that the talk of "roots" is misleading if we have no proof either of a high-reaching trunk or of deep penetration downward. The gold veins of Otago, like those of Nova Scotia, are ore-bearing to a shallow depth. If a number of quartz seams in thin beds of slate—as in Nova Scotia—contain enough gold to be profitably exploited for only 500 ft., are they likely to have persisted 20,000 ft. upward or are they likely to persist to 20,000 ft. downward? Until the geologist

can answer that question, his ruminations will remain unprofitable.

Long ago, in 1891, after an examination of several gold-mining districts in Otago, I came to the conclusion that the gold of the rich alluvial deposits (mentioned by Professor Park) was derived "not from the few comparatively insignificant quartz-lodes, but from the great mass of the quartzose schists. The quartz folia which form the characteristic feature of the schists of Otago are known in places to carry gold far outside the limits of any of the particular lode-channels that also traverse them."[a] If that be so, it would not require any great depth of erosion to yield the quantity of gold that has been discovered in the detritus, which has undergone natural concentration.

Mr. Robbins does well to quote from the "Structural Relations of Ore Deposits," by S. F. Emmons. No paper ever did more to advance the application of geology to mining. Many papers have been written since then, and many of greater length, but at the time that paper was given to the mining profession it was the best bit of advice that the miner had received from the geologist. With the quotation made by Mr. Robbins, I agree heartily. I thank him also for emphasizing the limitations within which the subject was presented for discussion. As to his suggestion of a grand inquest on the corpses of dead mines, I fear that he will get no favourable response. Most of those whom he addressed will agree with Mr. Farish, whose remark on the subject of such a compilation of knowledge I quoted in my paper. The notion still prevails that unpleasant facts should not be paraded before the public. As to that, my opinion is fairly well known. Science is not built on make-believe; an industry cannot thrive on pseudo-science; an ostrich does not escape danger by poking his head into the sand. For the moment, my interest is not in the general public, but in the profession. Whatever the public may think, however foolish the notions it may cherish, it is incumbent on the mining engineering profession, in appraising mines and in equipping them for exploitation, to have a clear idea concerning the relative persistence or non-persistence of ore in depth.

The meeting then terminated.

[a] "The Gold-Fields of Otago." By T. A. Rickard. Trans. A.I.M.E., Vol. XXI, (1892), p. 442.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

OPENINGS FOR THE PRODUCTION OF CYANIDE.—F. W. Traphagen and J. C. Bailar, of the Colorado School of Mines, have digested a great deal of chemical information concerning cyanide manufacture in the January issue of the School of Mines *Quarterly*, and we are reprinting that portion of the article which goes directly to the question of commercial production. The authors make no claim to original research in the preparation of the work; they have drawn upon the standard authorities.

Production from ferrocyanide.—The old process, which was the source of potassium cyanide for many years, consisted in the decomposition of ferrocyanide by heat. The ferrocyanide was made by fusing together animal matter containing nitrogen, such as hair, hides, hoofs and dried blood, finely divided iron and potassium carbonate. The resulting mass was leached, the solution evaporated to the crystallization point and the potassium ferrocyanide allowed to crystallize out. The dried crystals were fused and the temperature gradually raised to dull red, the fused mass is stirred from time to time and care is taken not to allow the temperature to rise above the prescribed point, otherwise further decomposition would take place.

The reaction is as follows:

$$K_4Fe(CN)_6 = 4KCN + FeC + N_2$$

This shows that one-third of the cyanogen is lost. Rogus, and later Liebig, proposed the addition of potassium carbonate to the fusion by which the loss of cyanogen was somewhat lessened:

$$K_4Fe(CN)_6 + K_2CO_3 = 6KCN + FeO + CO_2$$

A further reaction, however, takes place, and instead of obtaining the final result as shown above we get:

$$K_4Fe(CN)_6 + K_2CO_3 = 5KCN + KCNO + Fe + CO_2$$

or possibly

$$2K_4Fe(CN)_6 + 2K_2CO_3 = 11KCN + KCNO + Fe_2O_3 + 2CO_2$$

Wagner proposed the addition of powdered charcoal and the substitution of sodium carbonate for potassium carbonate:

$$K_4Fe(CN)_6 + Na_2CO_3 + C = 4KCN + 2NaCN + Fe + CO_2 + CO$$

This method, it will be noted, produces a mixed potassium sodium cyanide.

Before using the Castner process, which will be described later, the Roessler, Hasslacher Co., the principal manufacturers in the United States, used a process proposed by Erdenmeyer. This involves the use of metallic sodium and also gives a mixed cyanide:

$$K_4Fe(CN)_6 + 2Na = 4KCN + 2NaCN + Fe$$

Fully 98% of the cyanogen is recovered.

Wichman and Vantin proposed the use of potassium or sodium alloy as being cheaper than the metals alone. These alloys are made by electrolysis of the chloride, with a fused cathode, usually of lead, and the substitution of the lead alloy for the sodium suggested by Erlenmeyer.

In Dalimot's process the sodium is produced during the reaction with ferrocyanide, *i.e.*, it is the more active nascent sodium which effects the change. The sodium is produced by the reducing influence of calcium carbide.

Etard's Process.—Dry ferrocyanide is fused with dry sulphocyanide:

$$K_4Fe(CN)_6 + KCNS = 5KCN + CN_2 + FeS$$

The iron sulphide settles out during fusion, the cyanide is decanted while still fused, and the cyanogen gas is collected in an alkaline solution.

The addition of potassium carbonate prevents the loss of cyanogen gas during the operation, and results in the production of seven, instead of five, molecules from the same amount of cyanogen bearing material. A small amount of carbon is also added.

$$K_4Fe(CN)_6 + KCNS + K_2CO_3 + 2C = 7KCN + FeS + 3CO$$

Production from Sulphocyanide. The principal sources of sulphocyanide are the illuminating gas industry and by-product gas producers.

Two general methods are made use of, first by removal of sulphur through formation of a metallic sulphide, and second by oxidation of the sulphur.

Erlenmeyer showed that potassium permanganate oxidized the sulphur in acid solution:

$$5KCNS + 6KMnO_4 + 4H_2SO_4 = 5KCN + 6MnSO_4 + 3K_2SO_4 + 4H_2O$$

This method being too expensive to find commercial application, he showed that the same effect could be produced with nitric acid and barium chloride.

Rasebon uses nitric acid as the oxidizing agent under fixed conditions. This method is the basis of the process used by the United Alkali Co. The solution of the sulphocyanide and nitric acid in slight excess is added to hot water, or preferably mother liquor heated in a closed boiler to 96° C. This solution is constantly stirred during the addition of the sulphocyanide, the sulphur is oxidized to sulphuric acid, while a mixture of nitrous acid, water vapour, nitrogen oxide, carbon dioxide and hydrocyanic acid is set free. These gases are separated by use of suitable absorbents, and the cyanide recovered in the final form of potassium cyanide.

Beringer uses barium salts and claims a reduction in the loss of cyanogen. The reaction in his process is as follows:

$$Ba(CNS)_2 + 2Ba(NO_3)_2 + H_2SO_4 = 2HCN + 3BaSO_4 + 4NO$$

Playfair attempted to reduce sulphocyanide with hydrogen in accordance with the reaction:

$$4KCNS + 6H = 2KCN + K_2S + 3H_2S + 2C + 2N$$

Even if this reaction were complete, which is far from being the case, only 50% of the cyanogen would be recovered. The use of hydrocarbon vapours for reduction proved a complete failure. Next he used metals as desulphurizers with considerably better results, and was able, ultimately, to secure a 70% saving.

$$KCNS + Zn = KCN + ZnS$$

Luttke uses zinc powder, fusing the mixture at 360° to 400° C. and securing 90% of the theoretical yield.

The British Cyanide Co. used a process in which copper, with the aid of hydrogen, in place of zinc, is used as the desulphurizing agent. A sulphocyanide of copper is made by fusing a sulphocyanide with excess of finely divided copper. The sulphocyanide of copper so formed, is decomposed with hydrogen, hydrocyanic acid being liberated, which is absorbed in an alkaline solution.

$$2CuCNS + 2Cu + 2H = 2HCN + 2Cu_2S$$

The copper sulphide may be converted into copper salts or into copper for use again and hydrogen may be replaced by coal gas or by water gas.

The results obtained by Conroy, who substituted iron for copper, were very satisfactory.

In Hetherington and Muspratt's process iron filings or turnings are heated with tar to reduce the oxide coating, and then heated with alkaline sulphocyanide and coal tar to about 350° C. As a result of this treatment alkaline ferrocyanide, a tarry residue and sulphide of iron are produced. The ferrocyanide may be leached from the mass and crystallized by evaporation.

Several other processes differ from the preceding only in some unimportant details.

In Finlay's process a mixture of alkali, or an alkaline earth, with charcoal is heated to 1,000° C. in an atmosphere free from oxygen gas, consisting of nitrogen and sulphuric anhydride. Through a solution of the product of the above treatment is passed a current of nitrogen and carbon dioxide in presence of an oxidizing agent. Hydrocyanic acid is liberated and collected in an alkaline solution.

Synthetic Processes. Bunsen, in 1845, attempted to produce potassium cyanide in a specially constructed blast furnace. It had been known for some time that one of the products of iron blast furnaces was potassium cyanide, and Bunsen's aim was to increase the amount of this material by special conditions. The furnace was filled with alternate layers of charcoal and potash, and a powerful current of air blown through the ignited mass. While potassium cyanide was thus formed the losses were very great and the product greatly contaminated.

Possoz and Boissiere used a process similar to Bunsen, soaking the charcoal with a saturated solution of potash. A plant was operated without commercial success, producing, however, ferrocyanide instead of cyanide, carbonate of iron being a part of the charge.

Another process was based upon the substitution of baryta for potash, as this base fixes nitrogen readily.

With reducing agents in presence of nitrogen, the following reaction takes place:

$$Ba + 2C + 2N = Ba(CN)_2$$

This is converted into potassium cyanide by reaction with potassium carbonate:

$$Ba(CN)_2 + K_2CO_3 = 2KCN + BaCO_3$$

Mond modified this process, making briquettes composed of barium carbonate, wood charcoal and tar. These were treated in a reducing atmosphere, then treated with a gas rich in nitrogen and previously heated to 1,400° C. The yield of barium cyanide was about 40%, of the theoretical.

Lambilly decomposes illuminating gas in presence of nitrogen and an alkali. This process has very important features, and some modifications of it may be expected to produce very important commercial results.

A number of processes depend upon the action of nitrogen or a mixture of alkali and charcoal, and differ in no very great particulars from Bunsen's method. These processes are too numerous to be cited in the present paper.

Chipman uses sulphuric dioxide and nitrogen, and produces both cyanide and sulphocyanide.

Moise heats a mixture of boron nitride, potassium carbonate and charcoal and secures the following reaction:

$$4BN + 3K_2CO_3 + 2C = 4KCN + K_2B_4O_7 + CO_2$$

The boron nitride is produced by heating to bright redness a mixture of finely divided borax and ammonium chloride.

An early process of Castner consisted in passing nitrogen through a column of heated charcoal

down which a stream of molten sodium was passing.

$$Na + C + X = NaCN$$

In Mehner's process molten barium carbide is electrolyzed in an atmosphere of nitrogen, a carbon cathode being used.

Frank and Caro use carbides, that of barium appearing to be the most suitable. Nitrogen, free from oxygen, or nearly so, is charged with moisture and passed through the heated carbide.

Ammonia was substituted for nitrogen in this process, and it has also been found that cyanamide can be thus made. A plant using this process was erected at Frankfort, but was closed down because the results were not sufficiently profitable to merit its continuance.

A modification of this process uses a mixture of calcium carbide and potassium carbonate upon which, while at a red heat, nitrogen gas acts.

The reaction are :

$$CaC_2 + K_2CO_3 = K_2C_2 + CaO + CO_2$$
$$K_2C_2 + 2N = 2KCN$$

With ammonia the reaction is even better :

$$K_2C_2 + 2NH_3 = 2KCN + 6H$$

A number of other processes, which will not be cited here, make use of carbides. Some of these bid fair to have a marked commercial success. Ammonia reacts very much more readily than nitrogen and hydrogen uncombined, and its use is the basis of many processes.

In one process a mixture of nitrogen, hydrogen and ammonia are passed over heated carbon, ammonium cyanide being produced.

Mactear uses a mixture of carbon monoxide and ammonia, which he passes over heated charcoal, ammonium cyanide being the product.

At the Stassfurt Chemical Works ammonia gas is passed through charcoal at red heat, the charcoal having been previously charged with potash. Much cyanate is formed, but this may be reduced to cyanide by heating the charge to full redness after the first reaction is completed.

Many modifications of these processes have been patented, but they are too numerous to be considered in a short paper.

Hood and Salamon made use of zinc in conjunction with ammonia and sodium carbonate :

$$NH_3 + Na_2CO_3 + Zn = NaCN + ZnO + NaOH + H_2O$$

The zinc oxide was reduced later with carbon and the sodium hydroxide converted into carbonate by use of carbon dioxide.

Schneider uses a sodium-lead alloy which reacts on a mixture of acetylene and ammonia, which are forced through the fused alloy.

Castner's process makes use of the reaction taking place between ammonia, heated charcoal, and metallic sodium :

$$NH_3 + Na + C = NaCN + 3H$$

Castner finds it better to proceed in two stages, first making an amide, $NaNH_2$, by passing anhydrous ammonia gas over sodium heated to 300° C. or 400° C :

$$NH_3 + Na = NaNH_2 + H$$

This amide is converted into cyanide by contact with charcoal :

$$NaNH_2 + C = NaCN + 2H$$

This process is in commercial use in Germany and in this country.

A process making use of the production of cyanamide as an intermediate step in the manufacture of cyanides has been worked out by the Deutsche Gold and Silber Scheide Anstalt. Had the Clancy process met with the success its promoters promised there would have been a great demand for cyanamide in metallurgy.

Various other processes have been devised. Some of these used various salts, more or less expensive, and many made use of electrolytic reactions.

Bueb's process making use of the by-product of the beet sugar industry is one of particular interest to us in Colorado. Beet residues contain a large amount of nitrogenous materials, and these may be without very great difficulty converted into cyanide. The residues are subjected to distillation in special apparatus.

In conclusion we are in a position in this country to not only supply our own needs of cyanide, but, if necessary, by developing our unused resources, to supply the world."—*Mining Science*, March, 1915, p. 49. (H. A. W.)

Use of Titanous Chloride in Qualitative Analysis.—"Titanous chloride (0·8% solution) reduces hot neutral or slightly acid solutions of platinum, palladium, or iridium chlorides, yielding the respective metals. It reduces cold solutions of auric chloride, forming a blue solution with red fluorescence and a grey-blue precipitate on standing. This test will detect gold in the presence of platinum or iridium, but free hydrochloric acid must be present to prevent precipitation of the latter. With alkali tungstates acidified with dilute hydrochloric or sulphuric acid, it gives an intense characteristic blue coloration, whilst with molybdates it produces a brown coloration changing to green or blue on standing or on heating. It decolorises acid solutions of alkali vanadates; the liquid becomes pale blue when heated. It reduces solutions of chromates to chromium salts, selenious acid to selenium, tellurous acid to tellurium, and sulphites and thiosulphates to sulphur. *Organic acids*:—Formates and acetates are reduced, with the formation of violet or violet-black basic salts. Alkali oxalates give a yellow coloration, and a yellowish-brown precipitate on boiling; succinates a voluminous violet-grey precipitate; and neutral tartrates a grey precipitate. The intense violet coloration given by solutions of alkali citrates is a sensitive and distinctive test. After a few hours the surface of the liquid is decolorized by oxidation, but at the bottom the colour persists for several days. Alkali lactates give a pale blue to violet-brown coloration, the intensity of which rapidly diminishes. Alkali benzoates give a voluminous brownish-green precipitate; salicylates a yellowish-green precipitate; and tannin (in aqueous solution) a voluminous red-orange precipitate. Citric acid can be detected in the presence of formic, acetic, succinic, lactic, and benzoic acids by the reagent, but oxalic and salicylic acids interfere.—A. Monnier. *Ann. Chim. Analyt.*, 1915, *20*, 1-4.—*Journ. Soc. Chem. Ind.*, Feb. 15, 1915, p. 153. (J. A. W.)

Petrol From Synthetic Crude Oil.—"When any hydrocarbon, such as kerosene oil, melted paraffin, vaseline, fuel oil, or any such material, is subjected to the treatment described it comes out after treatment a fluid having the characteristics of crude oil. Even water-white kerosene, after five minutes' treatment, comes out a dark green liquid by reflected light (dark red-brown by transmitted light), and which resembles ordinary Pennsylvania crude oil so perfectly that even an oil expert cannot pick out bottles of the manufactured

crude when placed among bottles of natural crude.

This strong resemblance in appearance also extends to marked similarity in composition. When this "synthetic" crude is distilled, it yields gasoline, kerosene, gas oil, and lubricating oils, much as any ordinary crude oil would do.

By this process "synthetic" crude oils have now been made from such dissimilar materials as paraffin wax, kerosene, fuel oil, rod wax, bottom settlings, gas oil and many like substances. In every case the resulting "synthetic" crude oil has resembled natural crude oil and on distillation has been found to yield practically the same products.

Apparently this process discloses certain fundamental principles governing the transformation of hydrocarbon bodies such as has produced crude oil in nature. Apparently it also points out and discloses in a most interesting way the lines of work which must be followed to transform any hydrocarbon into any other hydrocarbon.

Much work has been done by others recently, in attempts to make gasoline from heavier oils. In general, all these processes have yielded light products in some ways suitable for use as gasoline, but having the disadvantage of a marked yellow colour and a bad odour, which could only be removed by treatment with acid, resulting in the loss of a large amount of material.

By the present process, we change any hydrocarbon into crude oil, and this crude oil gives about 15% of absolutely colourless gasoline, without objectionable odour. The residue, after removing the gasoline, can be again put through the same process, when it will give more crude oil, and from this an additional 15% of gasoline can be obtained.

This process can be repeated over and over, and in our experiments up to now we have obtained even from still residue (the very heavy residue remaining after petroleum refining) up to 55% of colourless gasoline of excellent quality. From gas oil we have obtained 60% of high grade gasoline, and from paraffin, 65% of gasoline.

It should be emphasized that in every case the gasoline so produced has not been inferior, yellow, or bad smelling gasoline, but has been gasoline of the very highest grade, free from heavy ends (that is to say, free from products not of the proper gravity) and having an odour not one particle more objectionable than that of ordinary, carefully refined gasoline made in the usual way.

Apparatus and Process. The present invention rests upon the discovery of the fact that, when any hydrocarbon is placed in a vessel of such size that the hydrocarbon occupies about three elevenths of the total volume, and is under these conditions heated to such temperature as brings about within the vessel a vapour pressure of 500 lb. per sq. in., the hydrocarbon introduced into the vessel becomes changed into "synthetic" crude oil. For the best results a very small amount of a suitable catalyst (ordinary graphite, finely powdered nickel, copper, etc.) is added to the oil before the treatment described.

From the experiments made (representing some 2,000 tests) it appears evident that some very fundamental change is brought about in oil when treated in the manner described, and that this change is of so deep seated an origin as to lead to the general rearrangement and regrouping of the molecules, and leading to the production of new chemical bodies very similar, if not absolutely identical, with the material present in natural crude oil.

It is not yet at all certain that this method is capable of practical application in oil refining, in view of the fact that the pressure employed is quite high for commercial work. On the other hand, the remarkable results obtained are of such nature as to lead without doubt to efforts being made to build apparatus of large size for such work, but the success of these experiments cannot now be foretold.

This research work has been conducted under the financial direction and assistance of Mr. John T. Milliken, of St. Louis, Mo., president of the Milliken Refining Co." W. O. SNELLING.—*Metallurgical and Chemical Engineering*, March, 1915, p. 180. (G. H. S.)

NATURE OF COMPOUNDS OF NITROGEN IN COAL AND COKE.—"Glycocoll, asparagine, albumin, glue, pyridine, azobenzene, hydrazobenzene, acetonitrile, phenylisocyanate, and nitrobenzene were subjected to destructive distillation under conditions similar to those obtaining in coal distillation: only those substances containing an amino group or substituted amino group yielded ammonia, hence the substance in coal which yields ammonia on dry distillation is probably of an albuminous nature. Extracts of coal with organic solvents contained more nitrogen than the insoluble residue, but the presence of amino groups in the extracts could not be proved. Coke when treated with steam at high temperatures yielded over 89% of its nitrogen as ammonia when precautions were taken to prevent decomposition of the latter, agreeing with the assumption of Rau and Christie that the nitrogen of coke is present as a nitride of carbon."—E. TERRES.—*Chem. Zeit.*, 1915, *39*, 73.—*Journ. Soc. Chem. Ind.*, March 31, 1915, p. 265. (J. A. W.)

WATER SOFTENING BY ELECTRO CHEMICAL METHODS.—"Among the papers prepared for the meeting of the American Society of Refrigerating Engineers held recently in New York City was one by Mr. C. P. Landreth, of Philadelphia, calling attention to the fact that electricity will hasten chemical reaction and thereby improve the results obtained with boiler feed-water softening compounds. To soften water it is analysed in the usual way and the chemical compound determined which will precipitate the scale-forming compounds present in the water. For instance, calcium hydroxide is used to remove calcium bicarbonate, sodium carbonate is employed to remove calcium sulphate, etc. After the softening compound has been added to the boiler feed-water, it is circulated past parallel electrodes which are placed close together in order that as much of the water as possible may be brought in contact with the surface of the plates. Because of its ionising property electricity will separate the compounds into their components, thereby hastening the recombination to form precipitates, which coagulate and are removed without difficulty. Unaided by the addition of any energy, the reaction between calcium sulphate and sodium carbonate to form insoluble calcium carbonate and soluble sodium sulphate is ordinarily very slow, but if electricity be allowed to ionise the water containing these compounds the precipitate can be quickly removed. In a plant mentioned by the author of this paper, the calcium carbonate in solution was reduced to 3·91 per gallon, which is within 1/160,000 of the

theoretical amount which would be held in solution, the electrolysis producing sufficient coagulation, so that the precipitate could be readily removed in filters. Ten million gallons of water per day was treated in this manner with a continuous power requirement of only 180 watts per 1,000,000 gallons."—C. P. Landreth. *Indian Engineering*, March 6, 1915, p. 157. (J. A. W.)

METALLURGY.

Tube-Milling Efficiency. "In the December number of *Metallurgical and Chemical Engineering* there is an interesting article by Mr. W. J. Pentland, entitled "Notes on Tube-Milling in All Slime Cyanide Practice." Under heading, "Duty of Tube Mill," Mr. Pentland questions the use of the tube-mill in closed circuit as being the most efficient method of using that machine. In regard to this phase of tube-mill practice, one hears the statement so frequently nowadays by engineers and millmen generally, that "*the most efficient machine is the one which has the greatest quantity of pulp in closed circuit*," that I had nearly ceased to consider it as a debatable question. However, this statement may be no more axiomatic than the one that "the proper percentage of moisture for tube-mill feed is 38

During the summer and early fall of 1914, I spent three months daily watching the operation of a 72 in.×20 ft. tube-mill. This tube mill was fitted with the Chalmers & Williams patented adjustable quick discharge, and was running on split feed with conical tube-mills crushing copper ores for the flotation or oil-concentration process.

It is not the purpose of this article to make comparisons of the work of the two types of machines, but to use the results of some of the observations made during this time to throw some light on the question of the desirability of operating a tube-mill in closed circuit with a classifier, in order to obtain the highest efficiency from the tube-mill; and, incidentally, to show that with this type of tube-mill at least, there is no law which makes it necessary for the tube-mill feed to contain 38% of moisture, for the percentage of moisture in the feed carried from so dilute a pulp as two parts of water to one part of ore, to only 29% or 30% moisture.

The tube-mill under observation was used to prepare ore for treatment by the oil-flotation process for recovering the mineral; but the same conditions which would make the mill most efficient for this work would also make it most efficient for preparing ore for treatment by cyanide or, in fact, any subsequent process. It was decided, in order to get the best results from the oil-flotation process, to have the pulp as it left the classifier contain about 2½ parts of water to one part of ore, as it was desired to send the pulp directly to the oil-flotation machines without the intermediate operation of dewatering it. Naturally the work of the classifier was very materially affected by this thick pulp, but in reducing the capacity of the classifier to separate the finished product from the oversize, it brought out very clearly the effect of the classifier on tube-mill efficiency; for, if the classifier had been of ample capacity to do efficient work, the results observed on account of its lack of capacity would not have been noticed.

Effect of Classifier on Capacity.—It was observed that when feeding the tube-mill with material 6-mesh and finer, at the rate of 144 tons dry ore per 24 hours, about 8 of the product remained on 100-mesh screen, with about 70% of −200. Practically no material was returned from the classifier, so that in effect the tube-mill was producing a finished product as above. In order to maintain the desired density of the pulp flowing from the classifier, it was not practical to send the feed to the classifier, but directly to the tube-mill, using the classifier only to separate and return to the tube-mill the oversize from its product. When the feed was increased substantially to more than 144 tons, oversize began to be returned by the classifier. About 240 tons per 24 hours was the limit of capacity of the classifier, when making a clean separation.

When the tube-mill was fed at the rate of 240 tons per 24 hours, a screen analysis of the classifier overflow showed 15%, remaining on 100-mesh screen, with about 56 − 200; and in the return to the tube-mill from the classifier there was 22% + 100 and 16 − 200. Thus showed that it the classifier had had capacity to separate the oversize from finished product, the 16 of 200 which was returned with the oversize to the tube-mill would have passed out with the discharge from the classifier, making a total of 72 200; and if in making this seperation 7 of the material under 60-mesh and on 100-mesh had passed out from the classifier with the oversize where it belonged, there would have been left the 8 on 100-mesh in the overflow as in the first case.

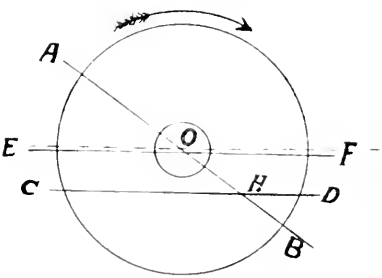
Under the first condition, with the classifier practically inoperative and returning no oversize, the tube-mill was making 92 of 144 tons, or 132·48 tons, of −100 mesh. In the latter case the tube-mill in connection with the classifier returning oversize, was grinding 92 of 240 tons, or 220·8 tons, of −100-mesh product; but the classifier did not do its work properly, as 30 of the material returned to the tube-mill was −100-mesh and considerable oversize went off in the overflow from the classifier. Later when the classifier was changed it was found that there was no more −100-mesh product made when the tube-mill was grinding at the rate of 240 tons per 24 hours, with a classifier returning the oversize, than there was when it was grinding at the rate of 144 tons per 24 hours with no oversize returned from the classifier. These results would indicate that the efficiency of the tube-mill was decidedly greater when worked in closed circuit, than when it was attempted to deliver a finished product direct from the tube-mill in open circuit.

Results at a later period, when a still larger classifier had been installed, showed that the tube-mill worked even more efficiently when fed at the rate of 288 tons per 24 hours. In fact, every time during this period of three months when we thought we were beginning to reach the capacity of the tube-mill on a certain feed and to produce a certain mesh product we would find that what we had really determined was the capacity of the classifier used in connection with the tube-mill and not the capacity of the tube-mill itself or of an efficient combination of tube-mill and classifier.

Effect of the Pulp-load on Efficiency.—A recording watt-meter was used in connection with the motor driving the tube-mill, and the observations taken from this record showed that the load of pulp which the tube-mill was carrying had a most decided effect upon the ultimate duty or efficiency of the tube-mill by its effect upon the power necessary to drive it. With a feed at the rate of 144

tons per day containing about 40% moisture, the motor took 75 kw. With a feed at the rate of 240 tons per day containing about 38% moisture, the motor took 62 kw. At another time with a feed of 180 tons per day with 38% moisture, the motor took 70 kw. By reducing the moisture in the tube-mill discharge to 30%, by increasing the amount of material returned from the classifier, thus making a heavier floating load in the tube-mill, the power was reduced to 65 kw. When the tube-mill was fed at the rate of 350 tons per 24 hours, feed containing about 40% moisture, only 55 kw. was required to drive it. It would seem clear, therefore, that in making a comparison of the first two conditions, namely, (1) grinding 144 tons of dry ore all from 6-mesh feed to 100-mesh product, without any classifying; and (2) grinding 240 tons of dry ore all from 6-mesh feed to 100-mesh product with a classifier returning the oversize; it is evident that, not only was the capacity increased by using the classifier in closer circuit with the tube-mill, but the efficiency of the tube-mill was also increased by having a greater floating load, due not only to its ability to handle a larger amount of floating load, but also to a greater floating load from the amount of oversize in the closed circuit. The reduction in power necessary to drive it was from 75 kw. to 62 kw.

To emphasize this point: If we assume that power costs 1 cent per kilowatt hour, then the cost for power per ton of ore ground in the first case would be twenty-four hours times 75 cents, amounting to $18.00 for 144 tons of ore, or 12½



CROSS SECTION THROUGH TUBE MILL

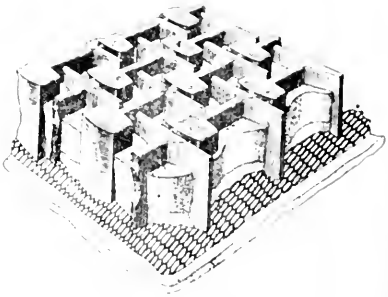
cents per ton. In the second case, the cost for power per ton of ore ground would be 24 hours times 62 cents, amounting to $14.88 for 240 tons of ore, or 6 2/10 cents per ton. This certainly is one answer to the question as to whether the tube-mill is more efficient when operated with or without a classifier in closed circuit.

The results of my observations over this period of three months on this particular tube-mill, showed plainly that, with a given load, the power decreased very rapidly as the density of the pulp was increased, starting with a density of two parts of water to one part of ore and reducing the amount of water to 30%. This was the greatest density obtained at any time, but there was no indication of congestion or clogging; and with a load at the rate of 240 tons per day, I would have liked to reduce the moisture still further if it had been practical. Of course the adjustable quick discharge opening below the feed opening and increasing the flow of pulp through the tube-mill.

The pebble load is an unbalanced load. The pulp, or the floating load, is a balanced load. Any increase in the weight of the latter, due either to increasing its density or increasing the amount by carrying a greater quantity of material in the circuit, is bound to decrease the power necessary to drive the tube-mill. Referring to Fig. 1, let AB represent approximately the pebble load in action; let CD represent a certain pulp-load level; let EF represent another pulp-load level; let O represent the centre of discharge opening. Then the more closely the triangle BHD approaches the triangle BOF in quantity or the triangle AOE in weight, or both, due to increased density, the less will be the power required to drive the mill. The instantaneous effect of changing this factor in the operation of the tube-mill, as shown by the recording watt-meter, was most interesting, and both the mill crew and visitors were much surprised at the results shown." W. B. EASTON.—*Metallurgical and Chemical Engineering*, Feb., 1915, p. 89. (G. H. S.)

BRITANNIA TUBE MILL LINING.—A novel use of rails, so often useless adjuncts to a mine and mill scrap heap, has been suggested in a form of tube-mill lining, invested by Henry Berquist, of Britannia Beach, B.C. The Hardinge Conical Mill Co. is bringing out this lining, crediting the form to the inventor mentioned.

The accompanying drawing shows the method of embedding the rail sections in the mill, the directions being furnished by the Hardinge Conical Mill Co. They are as follows:



BRITANNIA TUBE MILL LINING.

The interior of the mill must be thoroughly cleansed, especially from all traces of grease and oil. The mill shell should be absolutely clean, not merely wiped. A small quantity of cement mortar is mixed up, using one part of good portland cement and two parts of clean sharp sand, worked up to the consistency of thick paste. The cement is spread over a portion of the mill shell to a depth of about 1 in. The mill should be lined in sections running from feed to discharge openings. On top of the cement, place some type of wire screen, cut up in small sections, not to exceed 1 ft. 2 ft. Any flexible wire screen between 8 and 4 mesh will answer. That shown in the illustration is expanded metal lath, No. 8, gauge

27, unpainted. Press this wire cushion as deeply into the cement layer as possible. It is not necessary that the pieces of wire overlap, but the joints should be fairly close. The screen is desirable to distribute internal forces.

Over this a second layer of cement, approximately 1 in. thick, is spread on top of the wire. Then the rail ends are forced down into the cement as far as possible, using even a light blow if necessary, thus forcing the metal cushion still further into the cement bed. The rail ends should be arranged as nearly as possible in the order shown in the illustration, using 3½ in. or 2½ in. lengths. The 3½ in. length is better in a cylindrical mill. When half the mill is finished, another layer of cement is put in up to about 1 in. of the top of the rails. This is allowed to set and the lining of the other half of the mill finished in the same way.

The principal points in setting this lining are the care that the mill shell is clean, that the cement does not take any set before being used, that the rails once embedded are not disturbed in any manner, that the rails are forced into the cement bed as deeply as possible, that all crevices between the rails are well filled within 1 in. of the top, and that the lining is well set before the mill is put into operation. Mills equipped with this lining should be operated for several hours with half a charge of lump ore before any balls or pebbles are put in, in order to fill the remaining crevices, thus forming a cushion." *Eng. and Mining Journal*, Jan. 30, 1915, p. 239. (H. A. W.)

STAMP MILL TEST.—"Stamp milling has been receiving considerable attention during recent years, and several improvements and patent stamps have been placed on the market with some success. In view of the claims made for various types of mill, I decided to make a test between two of them. The plant was erected at Tincroft, which has the distinction of possessing one of the oldest reduction works in Cornwall. Accordingly, two Nissen single-stamp mills were obtained, also a Sandycroft (English make) five-head ordinary mill. These two units started crushing in June, 1914, and the figures of the comparative test are now available. Each Nissen stamp weighed 2,000 lb., while the five others each weighed 1,250 lb. The two batteries were driven by independent motors, the consumption of power being carefully checked and tabulated by the Cornwall Power Co. Only the coarse material that passed over the grizzly was sent to the breakers feeding the mills, the fine being diverted to the old Cornish battery for treatment. Both mills, of course, would have shown higher capacities had the fine been included in each case, the screen was of 10-mesh standard.

Result of test from June 18 to July 16, 1914, showed the following:

	Ordinary.	Nissen.
Ore crushed, tons	854·44	562·03
Total run, days ...	23·89	24·61
Duty per day, tons ...	35·765	22·837
Stamp duty each, tons	7·153	11·418
Total horse-power consumed	16·241	12·176
Horse-power per ton crushed per hour ...	10·959	12·796
Tons per horse-power per day	2·19	1·875
Cost of power, cents per ton	10·638	12·422

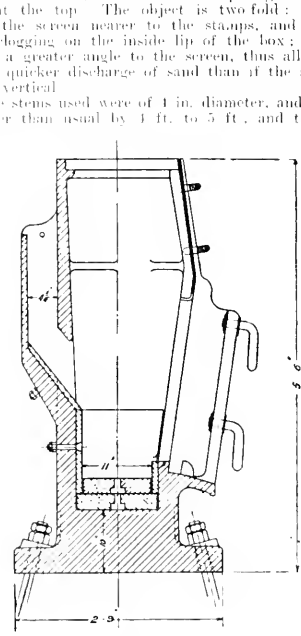
The average grading analysis of the product was as follows:—

Mesh.	Ordinary. %	Nissen. %
+ 20	13·37	15·63
+ 30	15·61	15·97
+ 40	9·21	8·30
+ 60	12·08	11·88
+ 100	16·55	9·91
+ 150	9·13	8·13
+ 200	3·03	3·20
200	26·10	26·63
	99·68	99·68

These figures do not show any great difference in the grade of the pulp from the two mills. The duration of the trial being under a month, the cost of maintenance could not be estimated accurately.

The five-head mortar-box was specially designed by myself in accordance with previous experience in Western Australia. The box was of the straight-backed type with detachable front, rounded corners (to strengthen), cast-steel liners standing 12 in. above the dies and bolted in position with countersunk headed bolts. The high crushing capacity is mainly due to the small area between the shoes and the box, this being only 1 in., and ensures a quick return of the uncrushed pulp to the dies. The screen was made to fit so that the bottom was flush with the inside liner. This was done by putting a filling piece at the bottom of the screen frame, with the sides tapering to nothing at the top. The object is two-fold: (1) it puts the screen nearer to the stamps, and stops any clogging on the inside lip of the box; (2) it gives a greater angle to the screen, thus allowing for a quicker discharge of sand than if the screen were vertical.

The stems used were of 4 in. diameter, and were shorter than usual by 1 ft. to 5 ft., and this, I



SECTION OF MORTAR-BOX DESCRIBED

would point out, reduced the vibration and friction to a minimum. The order of the drop was 1, 3, 5, 2, 4.

The success of an ordinary stamp-battery, I contend, is due principally to having a properly designed mortar box, the stamps falling exactly on the dies well under the cam-shaft. This cannot be secured, unless there are [illegible] iron guides same as [illegible] used in this [illegible] and dies made to fit the [illegible].

Many millmen make [illegible] having the base of the dies too [illegible] putting pieces of iron between them to [illegible] position. This is bad practice, for in the event of one piece of iron getting dislodged [illegible] throws the whole of the dies out of place, thereby at once reducing the crushing surface and efficiency. Hitherto, the necessary space for inside gold and silver amalgamation had generally somewhat influenced the standard design of the usual mortar box; but in this case, in view of obtaining the greatest capacity, it was entirely disregarded.

In Western Australia, the use of cyanide in place of amalgamation brought about this welcome improvement, and the old boxes packed with hardwood and steel plates gave the following capacities, equal to an increase of over 50% on previous results:

Screen, mesh.	Weight of stamps, lb.	Duty, tons
30	1050	4.76
20	1050	6.27
10	1050	8.90

These results were obtained on ore containing 50% to 60% granite, therefore on clean quartz, it can be assumed, a much better duty would have been obtained. E. S. King, *The Mining Magazine, Mining Press*, Jan. 16, 1915, p. 109. (G. H. S.)

MISCELLANEOUS

A Novel Theory of Explosion. "A remarkable and interesting theory of a 'human spark' being the cause of an explosion was put forward by Major Cooper Key, H.M. Chief Inspector of Explosives, at an enquiry into the origin of a fatal explosion in a cordite factory. It was pointed out that before placing the bags of cordite in the bogies, they were first wiped down by hand. This was usually done by the bogie runners, but on the day in question was apparently done by the nitroglycerine men. It was a dry cold morning, and the nitro men had on stout rubber overshoes which would completely insulate them from contact with 'earth.' The bags were of rubber-treated canvas, a marked dielectric, and rubbing them down would undoubtedly generate a static charge of electricity in the person of the rubber, which might well be sufficient to cause a spark to pass if he should happen to touch an uninsulated article. This theory at first sight seems somewhat fantastic, but as corroborative evidence of the probability of the theory being correct, Mr. Duff Grant, Managing Director of the new Explosives Co., tells the story that once when on a visit to Canada, where the air is practically free from moisture when the temperature is below zero, he proved by personal trial that by rubbing the feet on a thick pile carpet the gas could be lit by a spark from the end of the finger, and this not occasionally, but every time. This theory will very possibly throw new light on the origin of certain mine explosions, whose cause has never been clearly determined. It is well known that the floor of many dry and dusty mines is in itself so good an insulating medium that great difficulty is found in 'earthing' the armouring of electric cables. It is conceivable that a man might in such a mine induce in his person a static electric charge, and by touching an uninsulated article, such as a water-pipe, or a haulage-rope, cause a spark to pass which could begin a disastrous explosion." Anon. *Indian Engineering*, July 24, 1915, p. 55. (J. A. W.)

The Spelter Industry in Great Britain. "That Great Britain has been neglectful of her opportunities in relation to the spelter industry is evidenced by L. G. Chiozza Money, M.P., who discusses the situation at length. He places the world's production as follows:

(In metric tons of 2204 lb.)

	Total Output.	Output from Native Ores.
Britain	58,000	6,000
Germany	280,000	240,000
United States	320,000	294,000
Belgium	195,000	500
Holland	24,000	——
France	65,000	15,000
Austria	20,000	7,000
Russia	8,000	8,000
Spain	5,000	5,000
Above and all other countries	1,000,000	1,000,000

According to the statement it is apparent that Great Britain plays an insignificant part in the world's zinc production. Continuing, he says: "It is true that we have not big supplies of native zinc ore, but that is not a sufficient explanation of the humiliating figures which I have quoted. The smelting of zinc, like most other industrial operations, is based, not on native zinc, but on native power plus native enterprise. Take Germany, for example, which cuts such a very big figure in the total table. Germany has very fine native supplies of zinc, but that is not the reason why she smelts so much of the metal.

"As a matter of fact, Germany husbands her native zinc very carefully. Knowing that her native supplies, if worked to their full extent, would be rapidly exhausted, she has done all that she possibly could to work upon foreign zinc, and a part of her production is based upon zinc mined in other countries than Germany. The Belgian production is almost entirely from foreign ore.

"There is, indeed, no reason in the world why the United Kingdom should not possess a gigantic spelter industry, and I have heard it said on trade authority that, by employing the best known plant and methods, spelter could be produced here from imported ores more economically than it can be produced in the German Empire. At least there seems not to be the slightest doubt that we could easily put ourselves upon level terms with our greatest commercial rival in this respect.

"It has been shown that the United Kingdom has an output of only 58,000 tons of spelter. We have to supplement this production by enormous imports. For recent years our imports of zinc have been as follows:

	1909. Tons.	1911. Tons.	1913. Tons.
Zinc ore	73,800	74,400	64,700
Zinc crude	102,600	115,200	145,000
Zinc manufactures	19,000	19,100	18,800

"The imports of zinc ore in 1913 were worth only £418,000; the imports of crude zinc and zinc manufactures were worth £3,980,124." It will also be noticed that while our imports of ore have risen slightly, our imports of the metal have increased enormously. Between 1907 and 1913 the imports of spelter rose from 80,000 tons to 145,000 tons, while the imports of ore fell."

"Thus the opportunity to expand a most necessary industry has for years been staring us in the face. For some unaccountable reason it has been neglected, and we have been content to buy our spelter from Germany and elsewhere. . . . The facts of the case now are that the market is very short of spelter. We can lay our hands upon plenty of ore, but we have not smelting works capable of tackling it in sufficient quantity. It is the duty of the government to put down money at once to establish one or more big producing units. The sum required runs into hundreds of thousands of pounds, but the thing is an investment and a profitable investment, and not a mere expenditure. It is a great pity that the thing was not done at the beginning of the war. If it had been we should not be short of spelter at this time, and we should be saving money into the bargain."—*Mining and Engineering World*, July 17, 1915, p. 104. (J. C.)

Reviews.

THE UTILISATION OF WASTE PRODUCTS. By Dr. Theodor Koller. Translated from the second revised German edition. Scott, Greenwood and Son, 8, Broadway, Ludgate, London, E.C. 327 pp. 9×5½ in. Price, 7s 6d.

The wide range of waste materials covered by this work is shown in the following list of chapter headings:—I. The waste of towns. II. Blood and slaughter-house refuse. III. Fat from waste. IV. Tannery waste. V. Leather waste. VI. Fur and feather waste. VII. Waste horn. VIII. Fish waste. IX. Mother-of-pearl waste. X. Vegetable ivory waste. XI. Waste wood. XII. Cork waste. XIII. Waste paper and bookbinders' waste. XIV. Paper and pulp works by-products. XV. Waste produced in the manufacture of parchment paper. XVI. Wood waste. XVII. Silk waste. XVIII. Waste waters of cloth factories. XIX. Cotton spinners' waste. XX. Jute waste. XXI Utilisation of rags. XXII. Colouring matters from waste. XXIII. Residues in the manufacture of aniline dyes. XXIV. Dyers' waste waters. XXV Waste produced in butter making. XXVI. Molasses. XXVII. Waste liquids from sugar works. XXVIII Fruit. XXIX Waste products of the manufacture of starch. XXX. Brewers' waste. XXXI. Wine residues. XXXII. Indiarubber and gauttchouc waste. XXXIII. Amber waste. XXXIV. Utilisation of turf or peat. XXXV. Manufactured fuels. XXXVI. Illuminating gas from waste, and by-products of the manufacture of coal gas. XXXVII. By-products in the treatment of coal tar oils. XXXVIII. Ammonia recovery. XXXIX. Petroleum residues. XL. By-products in the manufacture of rosin oil. XLI. Soap makers' waste. XLII. Alkali waste and the recovery of soda. XLIII. Sulphur. XLIV. Salt waste. XLV. Gold and silver waste. XLVI. Platinum residues. XLVII. Iridium from goldsmiths' sweepings. XLVIII. Metal waste. XLIX. Tin plate waste L. Calamine slimes. LI. Waste iron. LII. By products in the manufacture of mineral waters. LIII Infusorial earth. LIV Meerschaum. LV. Mica waste. LVI. Slate waste. LVII. Broken porcelain, earthenware, and glass. LVIII. Utilisation of waste glass. *Journ. Soc. Chem. Ind.*, May 31, 1915, p. 583. (J. W.)

THE CHEMISTS' YEAR BOOK, 1915. Edited by F. W. Mack, M.Sc.Tech., B.Sc., A.I.C. Published by Sherratt & Hughes, 34, Cross Street, Manchester, and London. Over 1,000 pp. Two Vols. Price 10/6 net. Students' Edition, 7/6 net.

Many chemists and other scientific workers have availed themselves in the past of pocket books issued annually in the German language. The supply of these is now cut off and the editor has taken the favourable opportunity, so offered, to produce an annual pocket book in the English language.

It is hoped that this compilation will be of practical value to the English speaking chemists, particularly in view of the rapid developments which are expected in the near future from the more extensive application of chemical research throughout the British Empire in consequence of the cessation of the chemical trade with Germany. (J. W.)

STAMP MILLING AND CYANIDING. By Francis Andrew Thomson. Published by the McGraw-Hill Book Co., Inc., New York, 1915. 285 pp. Price, 8s.

This is an excellent little work on a subject which has been covered many times by other writers. One would think that unless something new had been developed there would be hardly any scope for another book. The concise manner in which it has been arranged gives the volume its chance. Strictly speaking, it is a compilation. The author, who is at the State College at Pullman, Washington, admits this. He has drawn freely from the leading journals in the United States, and one in England, which devote most of their space to ore treatment. Manufacturers of machinery for this purpose helped illustrate the book by supplying photographs of their products. These combinations resulted in a variety of opinions on processes.

The 35 chapters include the following discussions: characteristics of gold and silver; principles of amalgamation; the stamp mill and its accessories; a good review; stamp mill amalgamation; modifications and applications of the stamp mill from Gilpin county, Colorado, to the Rand; modified gravity-stamps, steam stamps, arrastras and pans, centrifugal and edge runner mills, and tube-mills and ball mills; comparisons of various mills, that is, work done; history and chemistry of cyaniding; preparations of ores for treatment, from crushing to agitators; dissolving gold and silver; separating solution and pulp; all filters described; precipitation methods; and clean up and smelting.

Following each chapter is a complete bibliography of the subject, a feature in several recent works, including 'Rand Metallurgical Practice.' A table gives the characteristics of slime-filters in convenient form. The two last chapters deal with the relations and functions of apparatus in gold and silver mills, so here are found flow-sheets of 18 types of mills. *Mining and Scientific Press*, May 15, 1915. (W. A. C.)

Abstracts of Patent Applications.

(C.) 118/15. Hans Foerstering (1), Arthur Lüdwig Halvorsen (2). Process of extracting values from ores. 30.6.15.

This application refers to a process for the recovery of metals, precious or otherwise, from cyanide solutions, and consists in passing the said metal bearing solutions through an apparatus where they come in contact with an alkali metal amalgam, where the said metals are precipitated.

(C.) 159/15. Lilian Clegg Tainton. Improvements in the electrolytic recovery of zinc from ores and other zinc bearing materials.

This application refers to a cyclic process of zinc extraction with electrolytic precipitation, in which the zinc salts is combined with an excess of free acid giving a solution of 5 to 30 acid prior to entering the electrolytic cell. The acid solution from the cell after precipitation of the dissolved metal is divided into two portions, one portion being used as the solvent of zinc in the ore which is carried to complete neutralization of the acid. This neutralized solution is then mixed with the portion containing the excess of free acid, so as to maintain the original condition prior to electrolysis. An organic colloid is used to maintain an adherent deposit at the cathode. In silicate ores the dissolved silica acts in the same way as the organic colloid, and the latter may be dispensed with.

(C.) 160/15. Thomas Henry Bradbury. Improvements in the methods of sharpening rock drills and in the means therefor.

This application relates to a method of sharpening rock drills consisting of submitting the inclined or bevelled faces of the teeth or wings of the drill immediately after the forging and shaping operation and whilst in a hot condition to the action of a rapidly rotated file or files, rasps or the like, having raised or recessed double coned cutting or abrading faces. (Claim 1.)

Claim 2 refers to the construction of the circular files or rasps which are formed by one or more pairs of discs rod rotated on a shaft.

(C.) 171/15. Lion Saint Denis Roylance. Ore reduction process. 28.7.15.

This application relates to a method or process of separating and recovering precious metals from their ores.

The applicant proposes to subject refractory material, after same has been crushed by the usual means to a suitably fine state of division, to a scouring and electrolytic action in a specially constructed agitator containing a solution of a suitable electrolyte and connected with an externally supplied electric current. By this means the applicant states that the gold is freed from base metals, such as antimony, arsenic and zinc, also from sulphur, and is rendered more amenable to amalgamation. The treated material, after neutralization, is then transferred to a combined agitator and amalgamator, where the gold is caught on amalgamated plates, and at the same time, by the passage of an electric current through an amalgamating solution containing suspected chemicals, the mercury is kept clean.

(C.) 172/15. Lion Saint Denis Roylance. Ore disintegrator. 28.7.15.

This application relates to improvements in the apparatus used in the disintegration of rebellious ores and, more particularly, to that class of apparatus in which an electric current is used to effect or aid the separation of the metals or the disintegration of the ores.

The applicant proposes to treat refractory ores, which must first be pulverized, by circulating same in a solution of a suitable electrolyte, in the apparatus under consideration, and at the same time, passing an electric current through the charge being circulated.

The electric current is supplied externally, and is passed through the charge by means of a series of plates suspended inside the disintegrator, the ore being forced through the spaces between the plates. An internal wall of the disintegrator is provided with a scouring surface, and the applicant claims that by the combined abrading and electrolytic action, the ore is disintegrated, and after treatment is ready for amalgamation.

(C.) 173/15. Lion Saint Denis Roylance. Agitating electrical amalgamator. 28.7.15.

This application relates to amalgamators, and more especially to electrical amalgamators of the agitating type.

The applicant proposes to treat ores by amalgamation by suspending same in a suitable medium contained in a vat and setting up agitation by circulating with a pump connected through a central pipe provided with radially extending arms. On said arms are nozzles, through which the solution is expelled, the expulsion causing the arms to revolve. At the same time electric current is passed through the charge, causing electro-chemical action, resulting in deposition of the metallic constituents on an amalgamated plate.

An essential part of the apparatus is a ball-bearing gland, similar to that described in Application No. 174/15. Means are provided by which the applicant claims that the metal composing the body of the machine is protected from electrolytic action, and also that the anti-friction bearings are protected from contact with the solution.

Changes of Address.

ALLEN, A. W., *c/o* London; Mineral de Salado, Rodeo, Provincia de San Juan, Argentina.

BOYDELL, H. C., *c/o* Randfontein; Village Deep, Ltd., P.O. Box 1061, Johannesburg.

BROWN, A. H., *to* Hudson Bay Mines, Ltd., Cobalt, Ontario, Canada.

GARDINER, R. L., *c/o* Pretoria; c/o Bulawayo Club, Bulawayo, Rhodesia.

KRAUSE, H. L., *c/o* Rietfontein; New Primrose G.M. Co., P.O. Box 193, Germiston.

LAWRIE, W. W., *c/o* Florida; Nourse Mines, Ltd., P.O. Box 32, Denver.

RICHARDSON, W. P., *to* Pilgrims Street, Barberton.

RIGGIS BRISL, H. E., *c/o* Johannesburg; Spains Hall, Finchingfield, Essex, England.

RUTGERS, C. A., *c/o* D.E. Indies; Snellinstraat 70, den Haag, Holland.

SCHUMACHER, R. W., *c/o* Johannesburg; c/o Central Mining and Investment Corporation, Ltd., 1, London Wall Buildings, London, E.C.

SMITH, L. HEDDLE, *c/o* Johannesburg; Bantjes Coos Mines, Ltd., P.O. Box 2, Florida.

THOMAS, W., *c/o* England; Champion Reefs G.M. Co., Mysore State, S. India.

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Vol. XVI. NOVEMBER 1915. No. 4

Proceedings
AT
Ordinary General Meeting, November 20, 1915.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, November 20th, 1915, Mr. J. E. Thomas (President) in the chair. There were also present:—

25 Members: Mr. H. A. White, Prof. J. A. Wilkinson, Prof. J. S. Cellier, Messrs. H. Meyer, E. Pam, John Watson, E. M. Weston, A. Whitby (Members of Council), H. R. Adam, H. C. Boydell, J. A. Campbell, L. Colquhoun, J. M. Dixon, H. W. Gill, C. E. Hutton, G. Melvill, H. H. Morrell, S. Nettleton, S. Newton, W. S. V. Price, A. F. Rose, C. B. Simpson, A. Thomas, J. F. Walker, and M. Weinbren.

9 Associates and Students: Messrs. L. D. Hingle, L. T. Leyson, G. F. Mathews, A. Milliken, D. Ordman, F. J. Pooler, H. Rusden, H. J. Spandow, and W. Waters.

8 Visitors and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBER

The Secretary: Since the last meeting the following gentleman has been admitted as an Associate:

ROBOTTOM, ARTHUR EDWIN, South Kalgurlie Gold Mine, Fimiston, Boulder, West Australia. Metallurgist.

OBITUARY.

The President: Before proceeding any further with this meeting, I have, with great regret, to announce the death of one of our Members of Council, Mr. Miguel Torrente, who has been a member of this Society practically since its inception, and was also a Member of Council for some time. He was on the Council again this year. Unfortunately, he was taken ill last Tuesday afternoon at a meeting of the Committee of the Co-operative Smelting Works, and died without regaining consciousness. I personally feel the loss very much. I started my metallurgical career under Mr. Torrente about 20 years ago, and was associated with him up to 1902, when I went to the Consolidated Gold Fields of South Africa. I shall certainly miss his advice and assistance on the Council of this Society, and I feel that I have lost a very true and loyal friend. I would ask you to rise as a token of sympathy with his family, and as a token of respect.

All members rose.

GENERAL BUSINESS.

MUNITIONS AND INDUSTRIES COMMISSION.

The President: There is another item that I wish to bring to your notice. The Government has asked us to take part in the matter of providing munitions, etc. A Conference was held, representing the whole of South Africa, on the 3rd of this month, at the Railway Offices, with Mr. Hoy as Chairman. The particular business was to elect a committee representing the whole of South Africa. Mr. H. A. White and myself attended on your behalf. I was elected a member on this committee, representing the Society, and also, at a later meeting, was elected on the Executive Committee, which consists of six members, with Mr. Hoy as chairman. The great idea is, at present, to try and relieve the engineering shops at Home of as much work as possible, so that they may devote themselves to the manufacture of munitions, and thus help the Government. As Mr. Laschinger pointed out at a meeting some time ago, it would be useless to send material out here, 6,000 miles, to turn out munitions which could very well

be made in Europe. If any member of this Society can help in any way, I shall be only too pleased if he will send suggestions to the Council, when they will be considered and brought to the notice of the proper authorities. I have full power from the Council to get any advice on any subject with which I am not competent to deal myself, so that the whole thing can be brought forward to the Government in the proper manner. Now that we have been recognised as a Society in this matter I hope that we shall be able to do something tangible.

MEMBERS OF COUNCIL.

The President: I have to notify a slight change in the Council. Mr. E. M. Weston has been elected to take the place of Mr. Stuart Martin, who, I regret to say, has found it necessary to resign.

OSBORN BARS FOR MORTAR BOXES.

Mr. H. A. White *(Vice-President):* I should just like to mention that a few meetings ago Mr. J. M. Dixon, of the Robinson Deep, mentioned the idea of using old Osborn bars instead of liners for mortar boxes. I would like to say we have carried out the idea at the Geduld in a modified form. Instead of replacing only the back plate of the box, we have extended the liner to go round the three sides, and we find that, in that way, we can hold it more securely. It makes, in fact, a very secure liner indeed, and we are very grateful to Mr. Dixon for having given us the idea. I will get Mr. J. M. Neill to send in a detailed sketch of the method now in use.

Mr. E. M Weston *(Member of Council):* As a mere miner I should like to know why old tram wheels are never used for lining tubes. In most mines there are generally 3,000l or 4,000l half worn-out sets of truck wheels and axles. It seems to me if the axles were sawn off they could be clamped inside the tube mill and then filled up with cement.

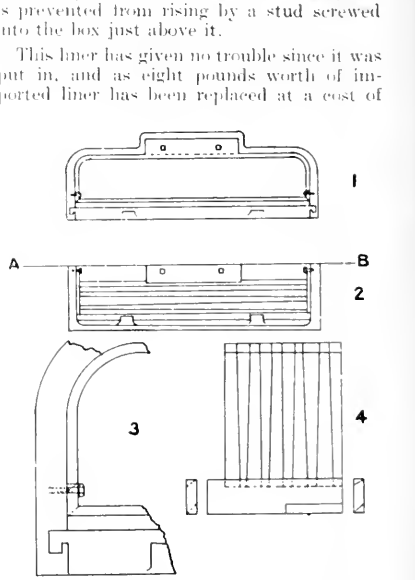
Mr. H. A. White *(Vice-President):* In reply to Mr. Weston, I would say that idea is actually being carried out at the Brakpan Mines. Mr. Dowsett started the notion, and he forms these wheels into blocks with cement. The difficulty is, on the average mine, one cannot find enough wheels to make one liner.

Mr. J. M. Neill *(Member):* The accompanying rough drawing will, I hope, shew the method adopted by me along lines suggested by Mr. Dixon for making use of old tube mill liners as a lining for motar boxes.

It will be observed that the back and side liners are all in one piece, bent to suit the type of box. The sides in the case are flush with the discharge opening, and so a key is necessary to prevent the bar liner coming forward.

The key is bevelled to suit all liners, and is prevented from rising by a stud screwed into the box just above it.

This liner has given no trouble since it was put in, and as eight pounds worth of imported liner has been replaced at a cost of



under 30s., the idea is certainly worth following up.

No. 1 is a plan showing liner in position below line AB of No. 2.

No. 2 is a front elevation, showing the liners at back of box with thick and thin ends of bars alternating.

No. 3 shows in more detail portion of side of box with stud, key and liners (plan).

No. 4 shows an elevation of liners and key from within box as in plan No. 3.

I am of opinion that where a filler is cast on the box down each side of the discharge opening, no key would be necessary and the bar liner could be dropped in from the top of the box. This would not answer where the back of the box is badly worn.

CHIEF SOURCES OF ACCIDENTS IN THE WITWATERSRAND MINES.

By Chas. E. Hutton, M.Inst.M.M., Inspector of Mines (Member).

Under Regulation 268, accidents which incapacitate persons from their ordinary work for 14 days or more have to be reported to the Inspector of Mines, who then holds an inquiry into the why and the wherefore, unless of course it be a mere trivial misadventure.

During last year 1,989 separate accidents were reported, resulting in 608 deaths and 1,715 persons injured. These accidents are classified under the following six distinct heads:—

	Separate Accidents	Persons.	
		Killed	Injured
1. Explosives ...	188	146	167
2. Falls of rock ...	454	204	371
3. Travelling in shafts	61	38	27
4. Machinery... ...	114	19	95
5. Various (surface) ...	193	27	168
6. Various (underground)	982	174	887
	1,989	608	1,715

Whilst it must be admitted that accidents are inseparable from mining, the enquiries into the reported accidents show that 19·8% have been due to contraventions of the regulations or carelessness, and in regard to the deaths no less than 42·3% have been similarly classed, or, in other words, were preventable.

In a series of lectures delivered last June I dealt fairly fully with all classes of accidents, but in this paper I intend only to touch on the chief sources of accidents, it being impossible in a short paper to go into the story of every one.

I will deal first with the accidents due to explosives which are classified as follows: (see Table, next column.)

	Separate Accidents	Persons.	
		Killed	Injured
1. Drilling into misfire	99	97	98
2. Gassing	14	17	6
3. Premature Explosion	4	*7	3
4. Charging up ...	8	9	6
5. Struck by flying stones from blasting	24	9	13
6. Sundry	12	7	11
7. Handling detonators	30	—	30
	188	146	167

From these figures we learn that 66% of the deaths due to the use of explosives were brought about through holes being drilled intersecting old holes, or sockets of holes being deepened, such holes containing explosives that had failed to detonate when fired.

The question is, how can this danger be avoided? In the first place we must ask ourselves how is it possible for a hole once charged and the fuse lit to result in a misfire.

Misfires can be divided into two classes, viz.:—

(1) Holes in which a portion only of the charge has exploded.

(2) Holes in which the charge has utterly failed to explode.

Take first the partial misfires, that is, where several cartridges being in a drill hole, the cartridges at the top will explode, leaving one or two sticking in the bottom unexploded.

We must understand that the blow given by the detonation of the cap in the primer must be sufficiently strong to bring about the perfect detonation of the whole charge. The explosion of a charge does not take place progressively, that is, one cartridge firing the next and so on, but takes place instantaneously.

With a weak cap an explosion may be obtained, but its effect will not extend to the end of the charge, with the consequent result that a cartridge or two remains unexploded in the bottom.

We are told by the makers of explosives that the blasting gelatine and gelignite used in these mines are capable of complete detonation if the miner does his work properly and uses sound detonators. It is therefore evident from the large number of partial misfires that occur on these fields the miner

* Included in the one accident at the City Deep with 5 killed, caused by lightning striking the cable for electric blasting.

fails to do his work properly or the detonators are at fault.

The detonating strength of a cap we know is adversely affected by dampness, and that a weak cap will explode a portion of the charge only.

As the mine air nowadays is to some extent charged with moisture, detonators when stored underground may after a time be affected, and therefore should not be used if stored for a longer period than, say, a fortnight.

In regard to faulty work of the miner bringing about a misfire, I would draw attention to two essential points,—the first being that the cartridges must be inserted one by one and pressed firmly home by means of the wooden charging stick so that the bottom cartridge is in direct contact with the bottom of the hole and each cartridge is in close contact one with another, leaving no air spaces whatever.

Tamping material should never be inserted between cartridges (Regulation 106 (22)) for the simple reason that the detonating force of the cap may not be sufficient to pass through this intervening tamping; secondly, Regulation 106 (19) must be strictly observed. This Regulation insists that the fuse with its attached detonator must be securely fastened to the primer so that it cannot be inadvertently withdrawn. It surely must be obvious that if this is not done there is always the chance of the capped fuse becoming detached from the primer to such a distance that the cap cannot possibly exercise its full detonating influence on the whole charge, with the result that a partial misfire is recorded.

Miners frequently, instead of tying, press the cap deeply into the primer for firmness sake; to bury the detonator thus is at least one of the causes of the production of dangerous fumes, as the fuse sides-spits and sets fire to the cartridge before the cap explodes.

I think it may be said that very few accidents can be attributed to drill holes intersecting holes in which the charge has utterly failed to explode, as such holes are usually easily detected and so avoided, excepting, perhaps, the "cut-off" hole.

The cause of complete misfires may be due to

(a) faulty manufacture of the fuse in the form of an interruption in the continuity of the gunpowder core;

(b) in not tapping out all the sawdust in the detonator before inserting the fuse;

(c) in failing to securely tie the capped fuse to the primer, so allowing it to become completely detached from the primer;

(d) cut-offs caused by the fuse of a charge being destroyed by flying stones from a blast near by, or an adjacent blast tearing away the collar of the hole and in so doing removing completely the fuse and possibly the primer, and perhaps one or two sticks of the charge itself. It should be possible to minimise to some extent the number of cut-offs by tucking the fuse into the hole and so giving it protection.

We will now take another view. Let us suppose the makers of the explosives are not quite correct in their statement that all explosives are always capable of being completely detonated, even if the miner does his work properly and uses sound detonators. This means that any working face after a blast may contain a misfire or misfires.

Regulation 106 (10) which lays down—

> "A ganger shall before pointing to the drillers working under him the position of the holes to be drilled . . . . . carefully examine such face and its immediate vicinity with a view to discovering misfired holes and sockets of holes."

If this Regulation is carried out by all certificated gangers there would, I am positive, be nothing like the number of accidents that have happened in the past. These accidents occur in all working faces, viz., stopes, drives, winzes, shafts, and raises.

Starting with stope faces, 66 persons were killed and 60 persons severely injured during the year 1914 through jumpers coming into contact with misfires. A great fault of gangers is frequently noticeable in that benches are not cleaned down as they should be before a hole is pitched. Many times have I seen machine holes being drilled on benches with a large quantity of rock from the previous blast lying on the faces, the gangers apparently never troubling to clear away more than was necessary to make space for the pitching of the hole to be drilled. Similarly so in hand stopes the benches have not been cleaned down thoroughly before drilling commences.

It may be claimed by the miner that he has not sufficient time to clear his working benches of all dirt, but I would suggest that if each boy, whether on hammers or

machines, be made to understand that no drilling would commence until the bench faces are scraped properly clear, this understanding itself would induce the natives to hasten this work, and drilling would in no way be delayed. It is quite impossible for a miner to effectively examine a face for misfires or sockets unless it has been carefully cleaned down.

It is perhaps not always possible to make sure of the direction of sockets, neither can one be certain of their depth, and as there is always the chance of their containing explosives, the obvious thing to do is to avoid them like poison and start the nearest hole away as far as possible.

I am of the opinion that there is little excuse for the miner to fail in detecting sockets of holes on benches when making his examination, more particularly so if his working place is run on single shift.

Dealing with accidents in development work, we find that no less than 31 deaths and 39 persons were injured through misfires being drilled into during 1914.

Sockets of holes are always present in development faces after a blast, any one of which may be a misfire in the sense that a portion of the charge may have failed to explode and remain sticking in the bottom.

It is sometimes difficult to detect sockets in development ends owing to the fact that the holes, being relatively close one to the other, when blasted are very apt to close the socket so completely that their discovery is troublesome, and for this very reason a new regulation was framed to deal with this difficulty.

This Regulation, viz., 100 (13), demands that before drilling is commenced in any shaft in the course of sinking and faces of winzes and drives being advanced, the ground must be blown over with compressed air or water under adequate pressure, so as to expose all misfires and sockets. This Regulation, although excellent, does not go far enough, for it fails to state that the sockets themselves must be washed out so that their direction may be ascertained; this should always be done.

This Regulation came into force at the end of 1913. Comparing the recorded accidents brought about by misfires in development ends in 1914 with previous years, viz.:

	1912.	1913.	1914.
Deaths	74	56	31
Footage cut	823,100	791,100	718,500

it would appear that the Regulation has been carried out with some success.

We have, however, the evidence of the 31 deaths to show that this blowing over has not been carried out effectively by all miners, and I would sincerely urge that all developers be particularly observant in the strict application of this Regulation.

Gassing Accidents.

Accidents due to gas poisoning are frequently met with in these mines, brought about through the mine air being contaminated by the fumes produced by

1. Incomplete detonation of explosives, and
2. Accidental burning of explosives.

With complete detonation the only harmful gas given off is carbon dioxide, and medical officers state that this gas is never the cause of fatal gassing accidents in these mines. It can be safely said, however, that complete detonation rarely takes place. With incomplete detonation the two dangerous gases, carbon monoxide and nitrous fumes, are produced in various quantities, depending on how close the detonation is to theoretical perfection. On the assumption, then, that carbon dioxide, carbon monoxide and nitrous fumes are ever present in varying amounts, it would appear at first sight that gas poisoning must be expected in a greater or lesser degree, but this is not so if ordinary precautions be taken.

In the first place there is not the slightest tear of gas poisoning happening in workings that have thorough ventilation. There is no record, as far as I know, of any gassing accidents having occurred in such places from incomplete detonation of charges. Fatal gassing accidents invariably occur in or near development ends, and occasionally in back stopes. If the Regulations in respect of the ventilation of winzes and water blasts be observed, and back stopes be furnished with compressed air lines, gassing accidents would be a thing of the past.

During 1914, 17 persons lost their lives through gas poisoning, and in each case the accidents occurred in development ends; in 1913 there were 51 deaths. Gassing accidents happen more frequently in winzes than elsewhere. Let us take the condition of the air in a winze after a blast. After blasting in the bottom, the air in the winze will contain varying percentages of carbonic acid gas and carbon monoxide, and perhaps nitrous fumes. Because of the presence of carbonic acid gas, which is heavier than air, this mixture will remain in the lower part of the winze. To descend into the winze,

therefore, after a blast would be dangerous unless these gases are dispelled. How are these gases to be dispelled? The best answer to this question is to quote Regulations 62, 106 (31). Regulation 62 insists that every winze shall be provided with adequate means of ventilation whereby any foul air at the bottom of such winze may be expelled, and that the air pipe down the winze used for blowing out the foul air shall be fitted with only one cock, and this cock to be at the top of such winze, and, further, no person shall descend until the foul air be expelled. Regulation 106 (31) demands that the ganger shall immediately after lighting up, put into action the water blast, which he shall previously have tested. Carbonic acid gas, and particularly nitrous fumes, are soluble in water, but carbon monoxide can only be got rid of by ventilation. The effect of the water blast is threefold:—

1. Nitrous fumes rendered harmless.
2. Displacement of gases by the induced fresh air.
3. Allaying dust.

There is no doubt that if these three regulations be properly observed no gassing accidents should ever occur in winzes.

A common cause of gassing accidents is due to natives entering development ends and setting about their work before their boss arrives, the fumes of the previous blast not having been expelled. In cases of this kind the natives themselves are certainly at fault, but at the same time the ganger is in all probability also guilty, for it is up to the miner to see that all the natives working under him are acquainted of the danger attending the contravention of Regulation 100 (11), which reads: "No person being the member of a gang under the charge of a ganger shall either at the beginning of a shift or after blasting, enter the working place until he has received definite instructions so to do from such ganger."

I now come to gassing accidents due to accidental burning of explosives. A great number of accidents have occurred on these mines through explosives having been set on fire accidentally. To bring it home to you what can happen when nitro glycerine compounds are burnt, I will give you a short account of the disaster that occurred at the Durbantem Consolidated on the 28th March, 1907. This accident happened at the Caesar Shaft at about 11 p.m., resulting in the death of 6 white men and 53 natives. All these persons were waiting at various stations to ascend at the end of the shift, when some gelatine was accidentally set on fire on the No. 11 level. From the evidence taken at the enquiry, it would appear that a miner whilst locking up his gelatine box, by some means set fire to the contents, which flared up and heat generated to such a degree that the remaining gelatine exploded in a short time. Death was caused in most cases from gas produced by the burning gelatine.

To show how these gases can drift with the ventilation and rope in victims hundreds of feet away from the site of the burning, I give you particulars of a disaster that occurred on the 21st July, 1910, at 8 a.m., in the workings of the Hammond Shaft of the Simmer East, which resulted in the death of one white man and 25 natives. This accident was caused by the carelessness of a miner dropping a lighted candle into his "Explosives" box on the 14th level. This explosives box contained 50 lb. of gelatine. The current of air on the 14th level was fairly strong and carried the fumes through a stope up and along the 13th level, where it split, going east and west. The effects of the fumes were felt on the east side for 750 ft., and on the west side for over 750 ft., and on both sides it found its way to No. 12 level, through stopes where fortunately all persons had been removed. Had the officials not been alert many more deaths might have been recorded.

I am positive that if ordinary care be exercised by the miners and the Mines and Works Regulations be observed, accidents due to explosives would be reduced to a tithe of what they have been in the past.

Falls of Rock.

I come now to accidents brought about by falls of rock, the most prolific of all causes.

As already shown, 151 separate accidents were reported during 1914, resulting in 204 deaths and 371 injured. Mining engineers in the past from all parts of the world invariably looked upon the country rock here in comparison to other fields as so hard that the usual precautions for supporting the workings were not a matter of much moment. We have, however, learnt that this opinion is a wrong one, and vigorous steps are now afoot to counteract the policy of the past. We have instances proving that the more dangerous the underground workings the safer they are to work in; by

this I mean that in mines in which there is almost a continuous settlement of ground going on, managers take such precautions that the workings are made so safe that accidents rarely occur. As a case in point, take the old Jumpers, where the extra safety precautions taken were such that the accident rate decreased as the mining difficulties increased. Falls of rock are brought about by—

1. Non-adhesion to upper strata, owing to stratification planes;
2. Presence of weak planes, such as faults, dykes, and clay partings;
3. Water penetrating joints, loosening the adhesiveness of the rocks;
4. Constant shaking, produced by blasting;
5. Pressure,—so-called "air-blasts";
6. Vibration set up by machine drilling.

We have falls of ground from all classes of excavation faces, as the following table shows:—

	Separate Accidents.	Persons Killed.	Persons Injured.
Hanging walls	234	139	177
Pillars	29	18	19
Stope faces	99	15	98
Drives, etc.	89	32	77
	451	204	371

The Mines Department records show that out of the 451 accidents 10 were due to extensive falls, thus proving small falls are the chief source of the high accident rate, making it plain that this danger requires the careful attention of both the management and men. The majority of accidents caused by small falls are due, I think, to over-confidence or carelessness on the part of the gangers. Miners are too much inclined to take it for granted that because the working place was safe yesterday it will be safe to-day. They have no right to hold that opinion, for the last blast may have so shaken the hanging that a small fall might be imminent, in fact any one of the six mentioned agencies in regard to causes of falls might loosen some ground during the ganger's 16 hours' absence from his working place; for this reason Regulation 106 (5a.) was framed.

Falls From Hanging.

The question of examination of the hanging and taking precautionary measures to prevent falls, is, as one question, the biggest one we have to deal with, for no less than 68 % of the total deaths due to falls of rock were caused by hanging coming away. The hanging must be tested by sounding, for by striking it with a hammer it is often possible to detect whether the ground is firm or not. If there be any sound of drumminess, the hanging should be supported, or the loose or loosened piece brought down. This test only makes evident the existence of a thin slab being loose, for instance, if the sound emitted by hammering would make it appear that the root is solid, it is still possible that it is not firm. It is also very necessary besides sounding to examine the root for any cracks which may or may not be the forerunner of danger. Such cracks should be immediately pointed out to the shift boss, who must satisfy himself that precautionary safety measures are necessary or otherwise. Much useful information can be obtained by inserting "tell-tale" wedges in cracks, as they will indicate whether any movement is continuous or not. The best method to guard against the danger of falls of roof is to erect plenty of supports, even if it does not appear necessary. I have sometimes, when going through a stope, pointed out an apparently dangerous piece of hanging to the ganger in charge, and asked his opinion as to whether he thought it was quite safe to allow natives to work beneath it: I have had various replies to the question. One would say, "Oh, that won't fall this shift," another has said "I tried to get it down, but it would not come," quite cheerfully, forgetting that he was contravening the regulations by placing boys to work under it. Again, an Inspector saw two natives working under bad ground; he withdrew the boys and sent for the ganger. It was pointed out to the ganger that the rock in question looked as if it might fall any moment. He replied that he did not think so. To prove it, a pinch bar was obtained, and at the first touch it fell. Suppose now that that rock had fallen and killed one or both natives; the accident would have been reported, and an enquiry held, and it is quite on the cards the ganger would have stated that he had carefully examined the place before putting the boys to work there, and did not detect danger. It is also possible that evidence might not be forthcoming to disprove the ganger's statement, and the ganger would perhaps have been given the benefit of the doubt, with the result that the Inspector's finding would wrongly read: "Deaths of natives due to danger inherent."

Judging from the foregoing, there is no doubt that a large number of accidents are due to what ought to be classed as criminal negligence on the part of the ganger in charge.

To put the question in a nutshell, the ganger should at the commencement of the shift conscientiously examine his working place for loose or loosened ground, assisted by his boss boy. Should he be doubtful as regards the conditions in respect of safety he must keep his gang away from the place. The shift boss who regularly visits all working places each shift, should be advised of the conditions prevailing, and from him instructions will be received whether or no there is any question of danger being imminent. The shift boss himself may not feel justified at the moment to give a decision without first consulting the mine overseer. I know that waiting for the shift boss will delay work, but a few tons of rock deferred when weighed against danger to life and limb should have no consideration.

I would also draw attention to the provisions of Reg. 106 (6), which states that the ganger shall report to the shift boss of his section of the mine, if any coloured person were in the working place before he arrived. It is possible that it is not generally known that many lives have been lost through natives going into a working place before it has been examined. Natives of a gang should be made to understand that they must wait for their boss at the waiting place, and remain there until their boss orders them to proceed to their work.

The hanging wall, that is, the "shooting wall", should never be drilled into, for by so doing it weakens its natural strength considerably. Should this "shooting wall" run away, and it becomes necessary to break to a new plane so that a minimum amount of waste will be errried in the face, a false hanging will then have to be carried, which hanging will require special attention in regard to its support.

Falls From Stope Faces.

Accidents due to falls of rock from stope faces are much more frequent than is generally supposed. As before mentioned, 99 accidents in this class were reported during 1914, 15 of which were due to "strain burst". As a rule, this type of accident is more dangerous to limb than life, as only 15 lives were lost in the 99 separate accidents, whereas 139 deaths occurred in the 234 accidents due to falls of hanging. Several accidents have occurred in undercutting the face to such an extent that the upper part of the face comes away without warning, and should the stope be in any way steep, this fall may slide down the face, and in transit cause personal injuries. Stope faces should always be sounded, and loose pieces pinched down prior to starting drilling.

Strain Bursts.

In regard to "strain bursts," commonly called "air blasts", I would point out that the term "air blast" is quite meaningless, being non-explanatory. How this name came to be a miners' byeword I do not know; "strain burst" is a much better term. Last year (1914) 32 accidents were attributed to "strain bursts", particulars of which are as follows:—

	Accidents.	Persons Killed.	Injured.
Hanging	3	1	4
Faces	15	1	20
Pillars	8	5	7
Drives	6	3	5
	32	10	36

On the Rand, where stope faces are subjected to strain, the practice in some mines is to drill holes in such faces at right angles. Such holes are drilled and blasted, not with the idea of actually breaking ground, but to release the pressure somewhat, so as to render the possibility of pieces of rock being violently projected from the face less likely to occur. These holes will be placed about 10 ft. apart in a line, and will be drilled to a depth of about 4 ft. This method has been proved to be a success, and should be adopted by all managements where pressure is evident in stope faces.

Pillars.

Turning again to the table of accidents, we see that 29 accidents, resulting in 18 deaths and 19 persons being injured were brought about by falls of rock from pillars during 1914. Of the 29 accidents, eight were due to "strain bursts", causing five deaths.

A number of accidents occurred through falls of rock from the underside of drive pillars, on the south side of the drift. In cases of this sort it is very evident that this danger can be overcome by the erection of closely-lagged stulls on the underside of all pillars showing signs of distress that are situated above workers, or by removing the pillars entirely and substituting packs.

Drives and Cross-Cuts, Etc.

You no doubt will be surprised to hear that no less than 89 accidents were reported under this head, causing 32 deaths in 1914. Nearly all these casualties occurred in completed drives. It must, of course, be admitted that these drives in use over the whole Rand will have a length running into several hundreds of miles; this being so, one might take up the attitude that accidental falls of rock are bound to occur here and there over such great lengths. Undoubtedly one must place a number of such accidents to misadventure, or dangers inherent, but at the same time by a little closer attention to the inspection of the roof and sides, the accident rate might be reduced. By a whole-hearted attempt on the part of managers, mine overseers and shift bosses to observe Reg. 10 (1), and gangers to conscientiously do their best to carry out Regs. 102 and 106 (5.a), there is no doubt the number of accidents due to falls of rock will diminish.

Shift Bosses' Responsibilities.

Shift bosses are most important officials. Given that they are conscientious workers, their duties may appear to them in some cases almost impossible, for not only is the shift boss made responsible in no small degree for handling the broken dirt expeditiously and cheaply, but also to see that a minimum amount of waste is stoped and proper observance of the requirements of the Regulations by the gangers and miners working under him.

For shift bosses to do their allotted daily task in an efficient manner, the district which they supervise should be of such an area that it is possible to allow all the working places to be visited at least once during each shift without undue exertion. This fact must be obvious to all managers, particularly at this juncture, when it has to be admitted that numbers of miners on these fields have not had much experience. Shift bosses in visiting the working should not merely take a cursory glance at the place and pass on, but should spend some little time in the stope with the ganger, asking and answering questions in connection with the work and general conditions.

I would emphasize an important factor, viz., that a large number of men employed underground are not experienced miners. At the moment there are on the Witwatersrand some 10,900 white men working underground, and this number can be split up into three classes, viz.:—

1. Skilled.
2. Unskilled.
3. Learners.

The position is, we have not enough skilled and experienced miners to fill all requirements, with the consequence that there are a number of miners, although certificated, with only a comparatively short tramming, working as overseers of gangs of natives. It is with this class of miners that the shift bosses should do their best to teach now to detect dangerous ground, and what points he should observe to further the "Safety First" movement.

Various (Underground).

The 982 accidents under this head are very varied, from which I will select the most important ones, viz.:—

	Separate Accidents	Persons.	
		Killed	Injured
1. Trucks and tramways	233	23	214
2. Falling of material	442	47	378
3. Falling in shafts or excavations ...	68	41	35
4. Struck by cage or skip	47	20	30
	760	131	657

The majority of the drives in these mines are what is usually termed "single" drives, the tramming track being laid practically central, and the boxes serving the stopes project into the drives. Tramming with, say, 16 cub. ft. trucks, running loaded on a 1" grade, often have a speed of five or six miles per hour, with the consequent result that a collision between a derailed truck and an oncoming one is of a somewhat violent nature, and unless the tramming natives are particularly wide-awake and agile they are apt to meet with severe injuries at times. Derailment of trucks is fairly frequent, owing to spillage from boxes or other causes, and unless the train of trucks is run with a fair distance between one and another accidents are bound to happen.

In the every-day work of filling trucks from boxholes and transporting contents to the grizzlies, there are bound to be many minor injuries inflicted, and this probably will always be so; but the number of fatal

accidents ought to be reduced if the loaded trucks are always kept under proper control, and tracks in good order.

Another source of fatal accidents is the running away of trucks down winzes or incline planes, before they have been attached to the hauling rope.

With the object of preventing this class of accident, Reg. 15 (e) was framed, which reads:—

> "On every inclined shaft, winze, or roadway in which trucks are worked attached to the end of rope, a self-acting turnout switch or other effective contrivance shall be provided, and used to prevent the trucks from running away."

Such a rig should always be installed at the top of every winze, and must be made to work automatically. On angles of low dips, say 20° and under, trucks should be furnished with a drag or trailer, so that in the event of the rope breaking, the truck is brought to a standstill. Finally, the connection between ropes and trucks must be so designed that accidental disconnection be made impossible.

Falls of Material.

Taking the figures for 1914, we see that 412 separate accidents were reported, causing 17 deaths and injuring 378 persons. By far the greatest number of these casualties occurred through stones rolling down stopes, in actual fact, 306 out of the 412 accidents were due to this cause, and 59 were brought about through timber and lagging-poles falling.

Numerous enquiries have been held into accidents brought about by stone rolling down stopes having an angle of less than 40°. This is a danger which is difficult to guard against, it is ever present, for after each blast stones will be thrown many feet away from the face outside the usual shovelling area.

Protecting stulls are usually only erected in stopes that are steeply inclined, that is, 45° or over. I would suggest that managements might accept a lower angle than 45° when considering the question of protecting bulkheads.

Reg. 103 (1) draws the attention of the ganger to the possible danger due to falling material; this Regulation stating that he shall take all reasonable precautions against the injury of persons from the falling or rolling of stones or loose implements or other material. This one short order is the only injunction, in addition to Reg. 8 (3), in the Mines and Works Regulations dealing directly with a class of accident which is responsible for 20% of the total number of reported accidents.

Falls in Shafts: Excavations, Etc.

Accidents due to the above, viz., falling down steep places, are fairly frequent, and usually fatal, for instance the foregoing table shows that in the 68 accidents, 41 persons were killed and 35 injured. These accidents occurred in all manner of ways, most of which in a sense were avoidable, if the injured persons had not become somewhat indifferent and careless in the carrying out of their daily tasks.

Fatalities have happened to natives wandering about a mine without a light, they—feeling their way in the darkness—have fallen down unprotected places in abandoned portions of the mine. Another source is due to a person rushing the cage, despite the protest of the cageman, at a station, after the signal had been given to raise; the cage moves off as he is about to enter, with the result that he falls down the shaft. We have enquired into accidents where natives have lost their lives by being pushed into the shaft through scuffling in attempting to rush the cage.

When a large number of natives are being pulled from one station, double barriers should always be erected, so that the number of natives for one cage load may be passed through the first barrier, and there await the cage. By such an arrangement the natives waiting to ascend will not be subjected to pushing as they wait near the edge of the shaft.

Cases have occurred where natives have been placed on the work of cutting hitches in the footwall of steeply inclined stopes, and whilst so engaged have slipped and fallen, with fatal results. To prevent such accidents the managers of the mines in the Germiston district have framed a special rule, making it compulsory that boys whilst engaged on such work shall be securely fastened to life lines.

Struck by Cages or Skips.

By disregarding Reg. 25, several deaths have resulted. This Regulation does not allow persons to perform work in shafts in which winding operations are being carried out unless the persons so engaged are securely protected from the skip or cage, as well as from rolling stones and falling material. Natives employed in incline

shafts as skip boys have been killed through being struck by skips. It has come out in the evidence taken that it was the practice for skip boys on certain mines to remain on the track after the loaded skip had been rung away, instead of—as they should have done—getting either on the platform or in the ladderway. The skipman has no right to ring away a loaded skip from a box until he has satisfied himself that his spillage or loading boy is clear of the shaft, for in doing so he contravened the aforementioned Regulations.

Inefficient protection of foot travelling ways in incline shafts, which have a dip of less than 45°, has been the cause of several fatal accidents. Reg. 52 lays down that in all shafts the foot-travelling way shall be adequately railed off, so as to prevent any person inadvertently entering the winding compartment. This Reg. in non-steeply inclined shafts is generally complied with by erecting a single hand-rail, but this protection has been proved to be insufficient, for the reason that in many cases the clearance between skip and the aforementioned hand-rail is only a few inches. Should a person travelling on the ladderway allow his body to project over the hand-rail into the shaft, he stands a chance of being struck by a passing skip. More protection should be afforded, by adding two more rails, one above and one below the present existing single hand-rail.

Outside the humanitarian point of view, I would point out that £116,115 was paid last year as compensation in respect of deaths and injuries from accidents, such an amount being a direct charge against the mines. What the indirect cost, through loss of time, tonnage, etc., amounted to, I am not prepared to say. It is clear, both from humanitarian and monetary points of view, that every effort should be made to bring the accident rate down to a minimum.

Mr. E. Pam *(Member of Council):* I have pleasure in proposing a vote of thanks to Mr. Hutton for his very interesting paper. Although I have not had time to read it through carefully in order to criticise it fully, there are one or two points to which I would like to call attention. One point he mentions with regard to misfires, the detachment of the fuse from the primer, has in many cases recently been got over by a supply, at each headgear, of sufficient string which the men use to tie their fuses on to the primer. Originally large balls of string were supplied which rapidly disappeared, and it is now the practice to cut it up into reasonable lengths so that a man, when he takes his candles, also takes a certain number of pieces of string which he uses for this purpose.

One other point, where we go slightly beyond the regulations, is in regard to gassing accidents. There is no question that even to-day a large number of natives do go into the working places before the white men, and this, as Mr. Hutton points out, is a very frequent cause of gassing accidents. Recently, all the mines have been circularised and asked to make a particular point of warning the natives, not only when they come originally into the compound, but again when their second ticket is issued to them. The old practice was that the natives were warned when they started work; but it is fairly obvious that a raw native coming up from his kraal would not understand 99 per cent. of the warning that he receives. On the other hand, when he has been underground for a month, he has probably a reasonable idea of what is happening underground, and the warning may mean something to him. We hope, anyhow, that this second warning may assist us in preventing accidents.

With regard to accidents due to trucks, Mr. Hutton quotes a one per cent. grade as being common. Most of us unfortunately suffer from grades of one per cent. and even more, but the modern practice is to use a 0·7 per cent. grade which is safer and more efficient.

I do not think there are any other points I could bring out to-night, but I think it is clear that the industry as a whole owes quite a lot to these various papers, and to the Safety First Committee, which has been working along these lines. The Government Mining Engineer in his last report showed a very considerable decrease in the accident rate; and I think one is justified in saying that a large proportion of the improvement is due to the publicity recently afforded to the means of preventing accidents.

Mr. W. S. V. Price *(Member):* I should like to second the vote of thanks to Mr. Hutton for his very interesting and instructive paper. I am thanking him from the miner's point of view, the previous speaker having done so from the manager's purview. I will not attempt to criticise the paper this evening; though I hope to do so at a future date. One remark that amused me was that in which he suggests a stricter adherence to the Mining Law, which lays down that the

miner or ganger must instruct the natives in all the dangers of the mine, and all the things he must or may not do. If a miner was such an accomplished linguist as he would need to be to do that, there would be no necessity for him to spend his time underground; he could find much more healthy and comfortable employment on the surface.

The President: I think this is a very valuable paper. Mr. Hutton has delivered lectures in various parts of the Rest on this matter of safety, and he has been kind enough to condense them in a suitable form for publication in our *Journal*. I think we are very fortunate in being able to put this paper on record. At the same time, I hope it will lead to a good deal of discussion, which will bring home to people working underground the necessity for taking the proper precautions. As a mere surface man, I think it is appalling to look at those figures on the blackboard there and see how many people have been injured, more especially as, as Mr. Hutton has pointed out, many of these cases might have been prevented.

There are several mining men present, and, if they are not prepared to speak on this paper this evening, I hope they will go home and talk seriously over the matter and bring up any points they may have at our next meeting, or even before that. We have no meeting now till next February, but we shall publish at least one *Journal* in between, and they could be published in that.

Mr. H. A. White *(Vice-President)*: There is one point in connection with this paper. As it is of general interest, I feel inclined to mention it; that is, the question of the native being directly instructed, which has been brought up by Mr. Pam for the first time in my recollection to-night. The Government is certainly in charge of these natives, and is responsible for their protection, and I really think that a little more might be done in this direction. There are certain regulations which the native has to carry out. They are very simple. There is also the fact that a man can look after himself, if he knows how, better than anybody else can look after him. I think this idea of Mr. Pam's might be extended slightly and the native instructed by a competent native linguist. I refer to the remarks of Mr. Hutton, that the native is usually in charge of someone who is not a competent linguist. The Government might arrange instruction to the natives as far as is possible in those regulations which they can carry out for themselves—such as, for instance, not going into a stope before the ganger. Every "boy" should have such things explained to him, and, if inspectors were to periodically visit the compounds at suitable times and instruct the newer "boys" by means of suitable interpreters, I think something would be gained.

Mr. E. M. Weston *(Member of Council)*: I would like to point out that there is a regulation which instructs, in regard to the dangers of underground work, that notices should be posted up in the compounds in the native languages.

Prof. J. Cellier *(Member of Council)*: If the regulations are carefully read I think it will be found that there is ample provision made for acquainting natives with those portions of the regulations with which they are mainly concerned, and it is unfair to expect the Inspectors of Mines to go round teaching natives.

As regards posting up abstracts in the native languages, those of us who know anything about natives must recognise that natives learn much more readily by word of mouth than by caligraphy.

If the existing regulations are honestly carried out, and Mr. Pam has shown that they are met not only in word but in spirit as well, there seems to be no necessity to add to them.

Mr. F. J. Pooler *(Associate)*: Mr. White, apparently, would throw the responsibility for this on the Government. It does not seem fair. The Government is not employing the native; the responsibility must come from the employer, not from the Government. It is surely the duty of the mines to provide these linguists, not the duty of the Government. The Government is discharging its function in educating the native as far as it is able, and should not be expected to educate him in the Mining Regulations as well.

Mr. E. Pam *(Member of Council)*: It does not seem to be quite clearly understood by a large number of members that it is the usual practice at the present day to warn natives as to the danger of certain practices. The only point I made was, that at present certain regulations are explained to the native when he first comes into the compound by the linguist, who happens usually to be the Compound Manager, and that in the majority of cases the native is not capable of understanding them before coming into practical touch with his work. Our present proposition is

that he should be instructed again when he receives his second ticket, that is to say, when he has been underground for one month.

With regard to the regulations on which he is instructed, these have been arranged by the mines and passed by the Government Mining Engineer's Department, more especially with regard to the rules regarding dust. It so happens that at the present moment a new set of rules is being forwarded to the Government Mining Engineer for his approval, and, in a very short time, a new list will be circulated to all mines about which it will be part of their duty to instruct the natives.

The President: I think the start of this discussion is most promising. I hope it will be kept up, and the subject of the paper thoroughly thrashed out.

NOTES ON THE TREATMENT OF ANTIMONIAL GOLD ORES FROM THE MURCHISON RANGE.

By H. R. Adam, B.Sc. (Member).

(Printed in Journal, June, 1915.)

REPLY TO DISCUSSION.

Mr. H. R. Adam *(Member):* I have to thank those who, in discussing these notes, have supplied much interesting and valuable information.

In reply to Mr. Pooler, roasting with salt has, I believe, been frequently attempted with similar ores, but without practical success. Losses would probably occur by volatilisation of gold as chloride, but possibly the chief difficulty lies in obtaining effective action of the salt on account of the easy fusibility of antimony sulphide and its tendency to form lumps.

As was explained in the notes, no gold extraction tests have so far been made since the best conditions for flotation have yet to be determined. The success attained in dealing with the tailing will depend on the degree of success obtained in floating the antimony sulphide.

The remarks of Mr. Hans Meyer on the results obtained at the Cam and Motor lead one to hope that further information will soon be available of roasting experience there, where the largest roasting plant in South Africa is being operated. The temperature at which arsenical and antimonial gold ores should be roasted is of the utmost importance, the best results being obtained with as low a temperature as is sufficient for oxidation in the time available.

Referring to Mr. Weston's contribution, it is stated that sufficient information as to the state of the gold is not given, also that no idea is given as to whether the ore is payable as a gold ore *per se* or not. As Mr. Weston has drawn out a scheme of treatment suitable for the Weigal mine, the information given must have been fairly clear as to whether the gold is "free" or not, and as regards the payability of the ore, the notes did not refer to any particular mine on the "Antimony Belt." However, quoting Professor Stanley, "In some places payable tonnages exist of ore rich enough to yield profits in spite of the somewhat expensive methods necessary," and to this may be added that in many other places profits would depend on the success of other methods.

In view of the work being done in America on the concentration of gold and silver ores by flotation, Mr. Weston's remarks on concentration are very much to the point. Arguments for concentration require, however, to be backed up by figures as to the cost of obtaining the gold from the concentrate, and Professor Stanley's and other tests on ordinary concentrate from the Murchison Range ores indicate that smelting would be required. Mr. Boydell, in his notes on Australian methods, states that in one case the concentrate assaying 2 oz. gold per ton was sold to smelters, who deducted a few shillings for each unit of antimony present, consequently with any such arrangement very careful re-dressing of first concentrate would be required as indicated in Mr. Weston's proposed schemes for the Weigal Mine.

The results obtained by the Sunlight and West Sunlight Companies, as given by Mr. Boydell, seem very good—75 to 80% extraction by amalgamation and concentration on ore carrying from 3% to 4% stibnite.

The oxidised antimony minerals named by Mr. Boydell are all stated to occur in the Murchison Range (see Hall: "The Geology of the Murchison Range"). The antimony sulphide itself is remarkably free from arsenic, though I have come across patches of arsenical pyrite close to the stibnite veins.

RECENT INVESTIGATIONS ON DUST IN MINE AIR AND THE CAUSATION OF MINERS' PHTHISIS.

By James Moir, M.A., D.Sc., F.R.S.S.Afr.

(Printed in Journal, July-August, 1915.)

DISCUSSION.

Mr. G. Hildick Smith *(Member of Council)* It has always been my intention to write a short note, which I had intended to read before this Society, on the question of respirators, and as Dr. Moir has stated in his paper that "the very small size of the dust particles in mine air is the main reason why respirators are of little use as a protection underground," I feel that I cannot allow such remarks to pass without comment.

The question of the efficiency of respirators has always interested me, and I have very strong views on the matter. The prevention of Miners' Phthisis is the most serious and important subject affecting the mines of the Rand.

As is evidenced by Dr. Moir's paper, much valuable research work has been done to try and elucidate the difficulties attendant on the prevention of Miners' Phthisis, or, in other words, to prevent silicious dust entering the lungs of underground workers. One remedy is to have dust-free air underground. So far, with all the watering that is at present being done underground in all mines, the results obtained are far from perfect, although the conditions in respect to dust in mine air are to-day enormously improved compared with what they were only a few years ago. Perfection, however, has not yet been reached. That there will still be a general improvement I have no doubt, but it will always be necessary for all underground workers to take all the precautions possible to preserve their health. There is one way in which I am convinced that this can be done, and that is by use of an efficient respirator. There is a great difference between an efficient respirator and a commercial type of respirator. Now Dr. Moir says he examined some time ago all the commercial types of respirators, and from the results then obtained by him the Miners' Phthisis Prevention Committee and the general public has apparently been led to believe that respirators are of no use.

On such a grave question as Miners' Phthisis the public is entitled to know exactly what experiments have been carried out, what methods employed, and what were the materials tested, with the results in each case—before Dr. Moir proved finally that a respirator was of so little use. This information I must ask Dr. Moir to give us in his reply to the discussion on his paper. The question is such a serious one that it demands a much more careful study than it has yet received. The study of the use of a respirator is also useless as done by Dr. Moir from the purely theoretical point of view only.

I must ask Dr. Moir if he has ever had practical experience in the use of respirators underground, if so: What is noticeable as to the condition of flannel or other material after such material has been breathed through for any length of time? Is a respirator of any material more impervious to dust after being used for some time in a humid underground atmosphere or not?

In order that all underground workers may know without any doubt the true facts in respect of the advantage of the use of a respirator, I suggest that the Miners' Phthisis Prevention Committee should at once institute thorough, organised research work on the subject. In order to do this tests should be made on lines similar to the following:—

(1.) All the various types of material from which respirators could be made must be tested by aspirating air through them, both when the various materials are dry and also in different states of dampness.

(2.) Mucus from the back of the nose and throat of underground workers to be examined for dust:

(a). When underground work has been performed without the use of a respirator.

(b). When a respirator has been worn.

From the results of such tests an efficient respirator to be designed, made to a standard pattern, so that those underground workers who wish to preserve their health may feel assured that when worn systematically such a device will prevent them from contracting phthisis.

I have worn respirators of many types made by myself—not the commercial types and have tested many kinds in practice, judging by experience which I felt were the most efficient, and I am certain that my lungs would have been in a much worse condition than they are to-day had I not done so. To substantiate some of my remarks I have, in conjunction with Mr. Rusden, of the Ferreira Deep Assay Office, made some

tests, the results of which are to my mind most important and have a great bearing on the respirator question. That further research work is necessary is admitted, but undoubtedly the final results will go to prove the fact that a dust-proof respirator is possible, and one which can be worked in with comfort at the same time. Mr. Rusden will describe the apparatus and the methods used for testing the flannel, French flannel of a type I use in my own respirator. I will only give the results obtained, which were as follows:—

Test No.	Quantity of dust in mgm. (−200) per cm. passed through open funnel.	Quantity of dust in mgm. (−200) per cm. obtained through funnel covered with flannel.	Percentage of dust arrested by the flannel.	Remarks.
1	2·26	1·18	47·8	1 thickness of flannel
2	3·05	0·00	100·0	2 thicknesses of flannel
3	4·58	0·195	97·7	3 thicknesses of flannel
4	3·51	0·84	76·1	4 thicknesses of flannel

It is important to note that all flannel used for testing was perfectly dry, no experiments having been made with flannel of the same dampness as obtains when a respirator has been worn for a short time underground. It is highly probable that in such a case still more satisfactory results would be obtained.

In all the above tests Mr. Jas. Gray did the estimation of the quantity of dust in the sugar.

It is needless to comment on the figures obtained from the experiments, as such small quantities of silica are being dealt with that absolute theoretical results could not be expected.

Experimental errors are unavoidable. What I wish to so strongly emphasize again is the general direction in which the results of our experiments point.

Mr. H. Rusden (*Associate*): The method of taking samples of dust in air is well known to you; I propose, however, to give a brief description of the manner in which the samples were taken, results of which are appended.

These dust samples were collected from the air of the Ferreira Deep sample crushing room.

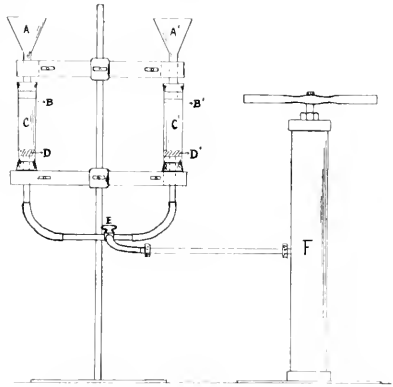
The crushing machines are all boxed in, an aspirating fan is worked all the time samples are being ground to remove the dust when the machines are being cleaned with a blast of compressed air.

The casing of each machine is connected to a common intake pipe of the aspirating fan, so that comparatively little dust escapes into the crushing room.

The apparatus for collecting the dust samples was placed about 2 feet from the casing of three crushing machines (1 Gates small type rock-breaker and 2 Ainsworth's fine grinding machines).

Whilst the dust samples were being taken the rock-breaker was working a portion of the time and the 2 fine-grinders were working all the time.

A glance at the sketch will best make clear the arrangement of the apparatus.



Apparatus for Sampling Dust in Air.
A.A.′ 3″ glass funnels. B.B.′ 200 mesh screening. C.C.′ 40 gm. −10+20 Tate's Coffee-crystal Sugar. D.D.′ cottonwool plugs. E. 3-way cock F. Suction pump.

Two sugar tubes placed side by side are surmounted with funnels, the bottoms of the tubes are connected with a 3-way-cock, which in turn is connected to the suction pump.

By manipulating the cock, alternate complete strokes of the pump are aspirated through first one, then the other sugar tube.

Whilst taking samples, the mouth of one funnel was left open, but the mouth of the other funnel was closely covered with one or more thicknesses of flannel, sampling was therefore conducted in pairs

An endeavour was made to regulate the pump-stokes so as to imitate as near as possible a man breathing.

Sample No.	Cubic metres air aspirated through each tube	Litres per minute aspirated through each tube	Milli grams 200 dust per cubic meter of air.	Thermometer: Dry bulb, Deg. F.	Thermometer: Wet bulb, Deg. F.	Remarks.
1	1·8563	10·5	2·26	61	55	Funnel mouth open
2	1·8563	10·5	1·18	61	55	Funnel mouth covered with 1 layer flannel
3	1·0475	8·73	3·05	66	59	Funnel mouth open
4	1·0475	8·73	nil	66	59	Funnel mouth covered with 2 layers flannel
5	1·0258	8·20	4·58	71	58	Funnel mouth open
6	1·0258	8·20	0·195	71	58	Funnel mouth covered with 3 layers flannel
7	1·1957	8·85	3·51	79	63	Funnel mouth open
8	1·1957	8·85	0·84	79	63	Funnel mouth covered with 4 layers flannel

The thermometer readings are given in the table, as these may be of interest.

Mr. E. M. Weston *(Member of Council)*: I was interested in Mr. Hilditch Smith's contribution on respirators. I do not believe that the smother type of respirator, even if it could be made efficient, would ever come into general use, since only a few people can stand the use of it. I believe that if the use of an efficient respirator is the solution of the phthisis question the possibilities of a battle respirator have not been exhausted. I can only say that I have designed one which Dr. Moir thinks worthy of trial; that it is being made, and that Dr. Moir has promised to test it. If it is on the right lines the Society will hear more about it.

Much has been said and written about the disease. Dr. Moir is, I think, entitled to much honour for fearlessly leading the way in the search for the real facts, whatever they are, and in stating the truth as it appears to him.

Let me state the truth as it appears to one who can claim to be fairly well acquainted with underground conditions.

Conditions are without doubt infinitely better than they were; and all credit is due to those who have pushed things to their present stage; but phthisis has not yet been stamped out on the Rand because:—

(1.) In the past a few mines have been working under conditions that made proper ventilation impossible. The best one can say about this is that no efforts and money have been spared to remedy early mistakes; but the price has had to be paid.

(2.) The great majority of the white miners are utterly careless and indifferent on this subject, and many of them are even yet quite ignorant about it. They do not even take the trouble to complain of conditions that are due simply to the management not being aware of them. I could quote instances to prove this.

(3.) The natives know little about dust dangers, and care even less than the white miners. They will make dust every chance they get. I have seen them drilling dry with a Leyner drill.

(4.) There may be possibly a few mines in which the dust allaying apparatus are not efficient everywhere; but the Mine Inspectors have managed to see that the underground officials do their part of the business with fair efficiency; and for every place where water is not supplied, there are twenty where it is not properly used. I have never known a case where the directors or managers were unwilling to remedy defects when they were pointed out to them; but a miner recently, writing on the subject, was right when he said that "there were often not enough pipe fitters employed underground."

(5.) Experience has proved, I believe, despite the convictions of a few self-satisfied officials, whose underground inspections I am afraid sometimes suffer from stage management, that mining officials cannot adequately enforce the regulations when 90 per cent. of the employees are either apathetic or determined to ignore them. It is not fair to expect them to do Government work and perform the other arduous duties of their position.

(6.) The present staff of inspectors have enough to do to see that the mine officials provide facilities for the use of water and proper ventilation and to prevent or enquire into accidents.

(7.) I am not alone in holding that phthisis can be entirely prevented if additional in-

spectors, mine police, or whatever you like to call them are appointed. They must, however, be men who will use whatever powers they are entrusted with tactfully and helpfully. Warning and instruction must be their primary duties. Officials and miners must not be persecuted unless all other means of remedying evils have failed. They must also work on somewhat different lines from the present staff, and they must on a certain number of days in the month spend full shifts below both night and day. One such official appointed for every, say, 50,000 tons milled per month could save the Government much wasted wealth in human derelicts and fatherless families and in hard cash. He could save the industry the loss of skilled miners, and that 5d. per ton milled, which threatens to expand to 8d. or 9d. It is time surely to face the truth. The regulations are not perfectly enforced, and underground ventilation is not yet nearly perfect, and while this is so, phthisis will shorten our lives and decrease shareholders' profits.

Mr. E. Pam *(Member of Council):* Dr. Moir has written a paper, which is undoubtedly very interesting, on the methods used in determining the size and number of particles of dust in air. It may strike many members of this Society and the general public as being rather strange that there has not been more discussion on this subject among mining men, because, unfortunately, in addition to this paper on the determination of the size and number of dust particles, Dr. Moir has made certain inferences with regard to the conditions underground which, really, do not belong to the paper, and on which he was not in a position really to give an opinion. Most of the members are aware that the Miners' Phthisis Prevention Committee is still sitting, and that they are at present drafting a report which will probably be published within a few months. Before this report is published, it would be very inadvisable for any members of that Committee to make any statement or give any definite opinion, as they are bound to use figures which they have received confidentially. The result is that those men who really have a certain amount of knowledge—anyhow, as much knowledge as there is to be gained on the subject—are not in a position to criticise the points raised by Dr. Moir with regard to dry drilling and the use of methods for the prevention of dust. It is seems to me that, if after the publication of the main report of the Miners' Phthisis Prevention Committee, someone would read a paper before this Society, to give an opportunity for discussion, there would be a very large number of interesting points raised, and, very probably, many valuable suggestions; but meanwhile, I think, it would be wise to discuss only the main points of the paper, which really centres round the determination of the size and number of dust particles.

Prof. J. A. Wilkinson *(Vice-President):* It had not been my intention to speak this evening on this paper, as I was not aware that this was the final discussion. Since the period when this subject was first brought before this Society, over 10 years ago, and the fierce discussion which waged at that time, but little advance has been made in the truly scientific investigation of the actual dust formed in mine air until the appearance of the paper upon which the author and his collaborator, Mr. Gray, are most deservedly to be congratulated. The adoption of the sugar filter to this problem certainly gave a method for determining the total mass of dust per unit volume, but gave little clue to the masses, and none to the numbers, of the particles of which it was composed, a problem, which has been entirely solved by the methods elaborated and worked out so carefully by Dr. Moir. The discussion, which has taken place on this paper, has given the impression that, in spite of the advance which it records, there awaits yet further research in the direction of the provision of a method of that accuracy, which one is accustomed to associate with technical work, and which can be utilised to yield results within a given reasonable time. The utilisation of the hydrolytic dissociation of normal sodium carbonate led some years ago, in the hands of the author, to a simple technical method for the determination of the CO_2 in mine air, which could be readily carried out on the spot by underground workers and the elaboration of an instrument which would achieve a similar object with regard to the dust problem and would solve this problem in a manner which it would be possible for the practical man to comprehend. The methods of optical analysis naturally appeal in this connection, and I would suggest that something might be done in this direction in order to discover, if possible, a method which would comply with the requirements mentioned.

Mr. M. Weinbren *(Member):* Returning to the question of respirators, it is obvious that

a respirator to be of any use should be light, and should fit the face perfectly, otherwise the miners will not use it. Although the flannel shown possesses the first quality, it would not be of much use as a respirator, for if the material is held up to the light, one can see the holes through it, which means, of course, that the material would be powerless to stop the fine dust which causes harm. As a matter of fact, the Miners' Phthisis Prevention Committee tested a respirator made of similar fabric, and it was found one hundred per cent. of the dangerous dust went through it. If the flannel shown were made wet it would stop more dust, but it would be difficult to breathe through. Further, Mr. Rusden did not tell us how the respirator would be made to fit the curves of the face, so that no dust should get through between the respirator and the face.

Dust sampling is carried out by pumping a certain amount of air through sugar crushed to - 10 and - 20 mesh. It therefore occurred to me that if this sugar could be utilised in the right way it would form an efficient respirator. I gave this a trial by filling a Covers' respirator with about 40 gm. of sugar and then inserting the sponge. The respirator seemed to stop the dust very well, but proved to be too heavy.

As regards the results obtained by Messrs. Hildick Smith and Rusden, I think they would have been more valuable if the tests had been carried out underground instead of in a crusher house, where the dust is a good deal coarser than that from underground, and the probabilities are that the percentage of dust passing through the respirator would have proved to be quite different. The tests made are, however, a step in the right direction, and it is possible that with more experimenting an efficient respirator may be evolved.

Mr. H. A. White *(Vice-President)*: I understand that examination of lungs of miners who have died from silicosis shows that the fatal size of the particle of quartz is in the neighbourhood of 2 microns. Larger particles seem to be provided for and are removed from the air passages by natural means. Now, quartz particles of this size, but not much larger, are affected when suspended in water by the well known "Brownian" movement or pedesis. This has the effect of decreasing the rate of fall, and might possibly be utilized for separation of dangerous from comparatively innocuous dust by flotation in an electrolyte free solution of suitable viscosity.

This fact also suggests the advisability of investigating the mechanism by which dust is rejected from the human breathing apparatus in order to determine whether this peculiar coincidence of size limit has any physiological explanation, and in that case if any advantage could be taken of the power of very small quantities of suitable electrolytes to cause aggregation in such suspensions.

CYANIDE CONSUMPTION ON THE WITWATERSRAND.

By H. A. White (Vice-President).

THE PREVENTION OF HYDROLYSIS IN CYANIDE SOLUTIONS.

By H. M. Leslie (Member).

(Printed in Journal, September, 1915.)

DISCUSSION.

Mr. John Watson *(Member of Council)*: In these stirring times of a great world war—of a political, followed by a municipal, election—it is somewhat difficult to settle down to quietly read, digest and criticise a scientific paper, even on the cyanide process—that process which was responsible for the birth of this society, and, which has kept it alive, more than any other, for a period of 21 years.

The author (Mr. Leslie) published a booklet, some three years ago, on "The Closed System for the Treatment of Ores by Cyanide." This booklet was kindly shown to me by Mr. K. L. Graham, and, partly on his suggestion and with the consent of our general manager (Mr. C. Glyn) I tried an experiment, which extended from 23rd June to 11th September, 1913, to check the results which I see are embodied in table 7, p. 39 of this paper. The author says:—"The tests, although conclusive to me, knowing how they were conducted, I recognise would not serve to convince others." . . . . .

This modesty and candour, on the author's part, are worthy of a fellow-countryman of the great David Hume, and it was partly in this spirit that intervals of spare time were found by the writer, for the sake of science, to make the necessary tests.

My tests were carried out on a working cyanide solution from the City and Suburban

Cyanide Department; this was divided into two parts.

The first half was kept in a large glass beaker, of about 3 litres capacity, uncovered; whilst the second half was placed in a 6-litre glass bottle fitted with tubulure and glass stopcock, for drawing off, near bottom. The mouth at top was closed with an india-rubber cork, fitted with glass and india-rubber tube and a pinchcock. These solutions were kept in a shaded room in laboratory and temperatures taken by a self-registering thermometer. Each morning, after pipetting or running off 100 cc. from each, when the tests were completed, the contents of beaker were stirred with a glass rod, with I.R. cover, and the bottle was agitated, so as to cause proper diffusion in the liquid.

The methods I used in testing for cyanide and protective alkali were those in use, so far as I know, in most Rand extractor houses, with the exception, perhaps, that I added 5 cc. N/10 K_4FeCy_6 solution as recommended by L. M. Green and G. W. Williams. (See "Rand Metallurgical Practice," vol. I., p. 326.)

The following table shows the results obtained:—

TABLE.—Experiment Checked on Working Cyanide Solution on City and Suburban Mine, 23rd June to 4th September, 1913.

Interval.	KCN Test (with KI)		Loss Percentage		Temperature Fahrenheit.		Protective Alkali (stated as CaO %).		Interval.
Days.	Open Vessel.	Closed Vessel.	Open Vessel.	Closed Vessel.	Max.	Min.	Open Vessel.	Closed Vessel.	Days.
	0·228%	0·228%					0·014%	0·014%	
1	0·213	0·226	6·58	0·87	64°	53°	0·014	0 012	1
2	0·192	0·224	9·21	0·87	59	53	0·016	—	2
2	0·190	0·222	0·87	0·87	61	51	—	—	2
2	0·188	0·221	0·87	0·43	63	52	0·008	0·018	2
3	0·161	0·220	11·84	0·43	63	55	0·004	0 012	3
2	0·155	0·219	2·63	0·43	62	50	0 003	0 010	2
2	0·145	0·218	4·38	0·43	56	48	0·003	0·010	2
2	0·142	0·218	1·31	nil	60	49	0·003	0·011	2
2	0·140	0·217	0·87	0·43	63	52	0·003	0·010	2
3	0·137	0·217	1·31	nil	62	51	0·004	—	3
2	0·128	0 217	3·94	nil	62	49	0·003	0·011	2
2	0·123	0·217	2·19	nil	62	50	0·004	—	2
3	0·118	0·217	2·19	nil	65	52	0 003	0 016	3
3	0·115	6·217	1·31	nil	65	51	0·003	0·018	3
4	0·102	0·217	5·70	nil	69	55	0·003	0·019	4
7	0·095	0·217	3·07	nil	59	54	0·002	0·021	7
3	0·092	0·217	1·31	nil	69	53	0·003	0·020	3
6	0·090	0·217	0·87	nil	61	47	0·002	0·017	6
5	0·089	0 217	0·43	nil	71	55	0·006	0 020	5
7	0·087	0·217	0·87	nil	73	53	0·009	0·017	7
10	0·085*	0 213	0·87	1·75	72	54	0 009	0·017	10
Total = 73 days			62 62 (By addition)	6·51 (By addition)					Total = 73 days
			62·72 By calculation direct Difference or balance = 37·28% of original strength remaining	6·57 By calculation direct Difference or balance = 93·49 % of original strength remaining					

*By this time the solution in open vessel was practically finished and began to crystallise. One more test after 7 days showed 0·100 KCN%.

Points of Difference.—It will be noted from the author's table No. 7, that in his series of tests the cyanide solution in open vessel, dropped from 0·234 KCN to *nil* in a period of 25 days, whereas my tests showed a drop of 0·143 KCN only, after 73 days, or in other words our solution lost only 62·7 of its cyanide in a period nearly three times as long as that covered by Mr. Leslie's experiment. On his theory of hydrolysis, the missing 0·143 parts of KCN should give the equivalent 0·123 of KOH 0·061 CaO. My protective alkali, stated as CaO, dropped from 0·014 to 0·009 in the open vessel, probably due to carbonation by atmospheric air; but one half of the alkali of K_2CO_3 or Na_2CO_3 should be indicated by phenol-phthalein.

Dr. Caldecott's question at the end of his contribution, p. 79, is much to the point. Clennell, in "The Cyanide Handbook," pp. 108-111, gives about eleven different headings of causes for loss of cyanide. Mr. Andrew F. Crosse read a paper in 1912, before this Society, on "The Action of Mineral Sulphates and Arsenates on Cyanide Solution." The writer then tried some experiments (see Vol. XIII. pp. 55-57) which showed that the presence of $CaSO_4$ and $MgSO_4$ do appreciably increase the loss of cyanide.

These salts, especially the former, are present to a large extent in the mill circuit water of most Rand mines.

The author appears to take it as granted that the great cause of loss of cyanide is *hydrolysis*; the title of his paper not only begs the question but has a remedy for it.

After perusing his paper I should return the Scottish verdict of "Not proven."

Mr. A. Whitby *(Member of Council):* In reviewing these papers, both excellent in kind and intention, one is, however, struck by a diversity of opinion on the main issue. Mr. White positively affirms that increased alkalinity effects a saving of cyanide, whilst Mr. Leslie equally positively affirms that hydrolysis of the cyanide is unaffected by increased alkalinity. Mr. White's paper presents such an amazing mass of figures, and is so wanting in a defined summary of the results of his investigation, that it makes difficult reading. It wants a comparative statement of possible saving in cyanide on the three plants dealt with by the increase of lime from, say, a normal protective alkalinity of 0·005 to 0·021, reckoned as NaOH. Here I wish to raise a point already made by Mr. Toombs, but in another way. Why, when lime deposits pursue us from launder to sump should we call it soda? Is it because the factor 0·004 is simpler than 0·0028? Does it not chemically and physically burke the issue between calcium and sodium hydroxide? Calcium forms the insoluble carbonate, and precipitates zinc from solution even as cyanide. Calcium acts (as sodium does *not* to any extent) as a coagulant. If any of you have ever examined the scale on an old cyanide pipe you will probably have found it to be nothing but carbonates of lime and zinc with a minimum of ferrocyanide. Personally, I agree with Mr. Leslie, and prefer to give my alkalinity in terms of lime. Anyhow, the conversion of one reading to the other should be a simple matter. The reading of cyanide as KCy is to be deprecated, but it is certainly convenient and common. There is never, therefore, any real need to change it to NaCy until we come to the footing of the bill. These, however, are but side issues. The main point I wish to raise is why does Mr. White brush on one side the influence of alkali on the zinc-lead couple? He talks vaguely about temporary accumulators, and makes one experiment to determine the total nitrogen in a box precipitate, that, too, taken at a mid-month clean-up, and then actually applies that figure parenthetically to two other mines.

My own experiments, interrupted and futile as they may seem, point to increase in zinc in solution, and its deposition as insoluble compounds by added alkali.

My first experiment carried out in Griffin beakers covered with clock glasses, was undoubtedly vitiated by the laboratory atmosphere; but I give my figures.

500 cc. of solution acting on 50 gm. of zinc dipped in acetate of lead and washed.

Experiment 1.

	1	2	3
KCy original ...	0·092	0·092	0·092
CaO original ...	0·0067	0·0173	0·037
KCy after	0·030	0·038	0·052
CaO after	Nil.	tr.	0·003
Zinc dissolved gm.	0·08	0·112	0·11

Unfortunately no record was made of the total cyanide in solution after action.

Further experiments gave interesting results, being carried out in the open and under different conditions.

Fifty gm. of zinc dipped in 2·5 lead acetate and washed till free from soluble lead covered immediately with 600 cc. of cyanide

solution, carrying progressive proportions of alkalinity. Forty-eight hours contact.

Experiment II.

Original		After			
KCy	CaO	KCy Total	KCy	CaO	Zn
0·1	0·0045	0·046	0·1	0·008	0·032
0·1	0·009	0·048	0·1	0·007	0·029
0·1	0·018	0·052	0·1	0·011	0·029
0·1	0·036	0·054	0·1	0·022	0·040

These tests were conducted in 1lb. jam-jars with screw covers, but these were simply laid on and not screwed. The tests point to the cyanide and lime, both interacting in the dissolution of zinc. The zinc determination in this case were held over for a week, and a noticeable precipitate occurred in all cases, but especially in Nos. 2 and 4. The precipitate consisted of zinc cyanide, but whether in combination as a double cyanide was not determined. I do not consider that the estimated zinc is a true measure of what was dissolved out and afterwards precipitated on the couple and in the solution. Closer investigation than I was able to give is necessary.

This experiment further seems to confirm Mr. Leslie in that covered vessels, even when imperfectly closed, do not permit loss of cyanide.

To carry the matter further, working solutions were used strengthened with pure caustic lime. The original values being:

Experiment III.

	KCy Total	KCy	CaO	Zn
Strong box	0·03 (?*)	0·09	0·007	0·027
Slime box ...	0·02	0·02	0·009	0·006

When strengthened with CaO.

Before Treatment

1. Strong box	0·04	—	0·026	—
2. Strong box	0·066	—	0·042	—
3. Slime box .	0·02	—	0·028	—
4. Slime box .	0·02	—	0·046	—
After Treatment as in previous Experiments.				
1.	0·028	0·07	0·010	0·031
2.	0·038	0·07	0·013	0·040
3.	0·012	0·02	0·010	0·011
4.	0·014	0·02	0·024	0·018

These tests were carried out in tall battery cells covered with clock glasses but with a large air surface above the solution.

To carry on the solutions were immediately transferred to the zinc in the jam jars of experiment 2, Nos 2 and 4 being raised in the alkalinity 0·01 . After 60 hours.

	1.	2.	3.	4.
Zinc	0·027	0·032	0·010	0·017

It was unfortunate that owing to a flood of other work these experiments were interrupted, and that values could not be further investigated. It might seem that these last results show no action, but there was undoubted deposition of zinc compounds on the couple and separation of the same in Nos. 2 and 4 solutions.

I lay no great stress on these experiments except to comment on the inadequacy of the tests made on behalf of the Chamber of Mines, in so far as, rightly or wrongly, the influence of alkali as lime plus cyanide on the solution of zinc has not apparently been investigated. Where zinc is standing at 1s. 3d. a lb. surely an investigation of this nature should have reckoned in this factor. It is idle to say that our consumption of zinc is 0·3lb. per ton, but what factors enter into this computation? Can Mr. White say how much zinc is due to clean-up emergencies and how much to his unnecessary lime? I say unnecessary, because I do not believe that the saving in cyanide effected by the superlative use of lime is in any way commensurate to the fouling of solutions, coating of pipes and dissolving of zinc arising therefrom. The more zinc there is in solution the more trouble there is likely to be if the alkalinity goes down from causes unspecified. In fact, I may conclude my remarks by saying that a very much closer study of the interactions of lime, cyanide, zinc, and atmospheric influences is required before we come to any real estimate. A copper-zinc couple will reduce the nitrates and nitrites in water to ammonia even in the cold. Add an electrolyte like KCy and couple it to an alkali like calcium hydrate in solution, and is it possible at this early stage to say that a surplus of the latter does not involve greater destruction and subsequent loss of zinc? I fully appreciate the value of Mr. Meikle-john's painstaking work, but regret that it has followed the one channel. I do not exactly know, but it may be possible that the conclusions drawn are like those of the dog in the fable, who dropped the bone for its shadow.

Referring to Mr. Leslie's paper, I am sorry that I have been unable to experiment on his lines. He certainly makes out a strong case for the closed system. As one of the editors of the *Journal*, however, I may remark on the unnecessary duplication of the formula for the hydrolysis of cyanide, and also for six nils in table 6, which have no significance. Under column 1, second line, test nil loss 100% seems correct, but 3rd and 4th line test nil, loss nil, expresses nothing. The same occurs on line 4 of column 2.

The meeting then terminated.

* There was some difficulty in obtaining a clear end point.

Obituary Notice.

DON MIGUEL TORRENTE.

Don Miguel Torrente was born on the 16th March, 1862, at Madrid, Spain. After a liberal education in the best schools of Spain, France and Germany he returned to his native country, where he spent a couple of years at office work. He then came to this country, arriving at Capetown in September, 1881. From Capetown he proceeded to Delagoa Bay and Lydenburg, where he managed the Spitzkop Lydenburg G.M. Co. for a time. When gold was discovered on the Rand he came to Johannesburg but returned to Lydenburg after a few months. In 1888 he returned to Europe and resumed the study of Mining and Inorganic Chemistry. On his return to South Africa in 1889 he was engaged in prospecting, diamond drilling and mining in different parts of the Transvaal and Free State until 1892, when a serious illness laid him aside for a year. In 1893 he returned to Johannesburg and became associated with Mr. Charles Butters, of the Rand Central Ore Reduction Co. When the Siemens Halske process was introduced he became cyanide manager at the Worcester G.M. Co. plant, and left there in 1895 to take a similar position at the May Consolidated G.M. Co.'s new cyanide works. In 1897 he became inspector for the Gold Patents Co., the owners of the Siemens Halske patents. In 1898 he left the Rand and proceeded to Siberia on some special work. On his return and following the resignation of Mr. A. von Gernet, he became manager of the Rand Central Ore Reduction Co., which position he held until the liquidation of that company in 1901. From that time he engaged in private consulting work, and was for some time associated with Mr. Alexander Heymann. In 1910 he was appointed Consulting Metallurgist to the General Mining and Finance Corporation, by whom he was employed at the time of his death on the 17th November, 1915. For his country, Spain, he acted for some years as Vice-Consul. Don Miguel became an Associate of the Society in 1897, transferring to membership in 1902 and was a member of Council during the years 1904-7, and also at the time of his death. The funeral at Brixton was very largely attended, the President, Secretary, and a number of members being present. His death is deeply regretted by all who knew him, and the Society tenders its sincere sympathy to his wife and family.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

REPORT OF THE RESEARCH COMMITTEE OF THE INSTITUTE OF CHEMISTRY. "The Glass Research Committee was appointed in October last to conduct investigations with a view to arriving at suitable formulas to be freely available to manufacturers willing to assist in maintaining the continued supply of laboratory glassware. The committee consisted of Prof. R. Meldola (then President of the Institute), Mr. B. Blount, Mr. O. Hehner, Prof. H. Jackson, Mr. W. C. Hancock, and Mr. T. R. Merton.

The chief aims of the research have been: (i.) to produce working formulas for all glasses used in laboratory work, and (ii.) to ascertain the influence of various ingredients on the physical and chemical properties of glasses. The work was extended to include glass for miners' lamp glasses, at the suggestion of the Home Office; and also glass for ampoules, to meet the needs of wholesale pharmaceutical chemists engaged in the production of Army Medical requirements. The committee have also examined and reported on samples of British and French laboratory glassware, produced since the beginning of the war, a number of the specimens being made from formulas similar to, and in some cases almost identical with, those recommended by the committee. Mixtures prepared in accordance with the analytical results were not always satisfactory; but the analyses were helpful in suggesting synthetic experiments.

Up to the present time, the Research Committee have reported eleven formulas for glasses for various purposes based on the results of about 400 experimental melts on a scale large enough for drawing rods and blowing small vessels. No formula has been issued without submitting the specimens made to rigorous tests to prove their suitability for the purposes for which they are intended. The question of workable temperatures has been carefully considered and, so far as it is possible to judge, the melts on a small scale indicate that even better results will be obtained on the industrial scale. This view has been justified by the samples already received from manufacturers who have tried some of the formulas.

Special attention has been paid to the proportions of basic and acidic substances in respect of the action of glass mixtures on clay crucibles during fusion, and it has been shown by careful investigation that the formulas proposed give melts in which the influence of the ingredients of the crucibles is very slight and in some cases practically inappreciable.

The following formulas have been communicated to a number of manufacturers who have expressed their interest in the progress of the investigation and to scientific workers who are conducting similar experiments.

Soft Glasses, suitable for ordinary chemical laboratory ware:

(1) Sand, 67·0 parts; sodium carbonate (Na_2O), 31·2; calcium carbonate, 11·6; alumina (Al_2O_3), 6·5. A soft glass which does not give up alkali readily to water, works well in the blowpipe and does not devitrify readily.

(2) Sand, 67·0 parts; sodium carbonate, 29·0; calcium carbonate, 9·6; calcium fluoride, 1·6; alumina, 8·3; boric anhydride (B_2O_3), 2·0. A soft

glass of higher quality. Does not give up alkali under severe tests. A "kindly working" glass before the blowpipe, and very difficult to devitrify.

Resistant Glass suitable for pharmaceutical purposes, ampoules, etc.:

(3) Sand, 67·0 parts; alumina, 1·0; calcium carbonate, 12·5; magnesia, 0·5; potassium nitrate, 1·0; sodium carbonate, 17·0; boric anhydride, 8·0. This glass is intermediate in hardness between soft glass and combustion tubing, is highly resistant to chemical action, withstands changes of temperature well, and should be a very suitable glass for high class beakers, flasks, etc.

Glasses for Combustion Tubing:—

(4) Sand, 68·2 parts; alumina, 6·2; barium carbonate, 8·8; calcium carbonate, 13·0; potassium nitrate, 4·3; sodium carbonate, 5·5; boric anhydride, 5·5; calcium fluoride, 1·0. This glass resembles Jena combustion tubing very closely indeed. It has practically the same fusing point. It fuses on to Jena glass perfectly, and is indistinguishable from it before the blowpipe and in its behaviour on prolonged heating below its fusing point. The presence of the small quantity of calcium fluoride facilitates the incorporation of the ingredients. The sodium carbonate can be reduced to 1·34 parts provided 7·93 parts of anhydrous borax be used in the place of boric anhydride.

(5) Sand, 68·2 parts; alumina, 6·2; barium carbonate, 8·8; calcium carbonate, 14·2; potassium nitrate, 4·3; sodium carbonate, 5·5; boric anhydride, 5·5. This glass is practically of the same composition as (4). It is not so easy to make or to work, but it does not become so opaque as Jena combustion tubing on prolonged heating. As in No. (4), the proportions given for sodium carbonate and anhydrous borax can be substituted for the figures for sodium carbonate and boric anhydride.

Miners' Lamp Glasses:—

(6) Sand, 65·0 parts; alumina, 1·0; calcium carbonate, 0·6; arsenious oxide, 2·0; antimony oxide, 1·0; potassium nitrate, 3·0; sodium carbonate, 14·0; boric anhydride, 24·0. A colourless and fusible glass withstanding rapid changes of temperature exceptionally well.

(7) Sand, 15·0 parts; alumina, 1·0; calcium carbonate, 0·6; arsenious oxide, 2·0; antimony oxide, 1·0; potassium nitrate, 3·0; anhydrous borax, 26·68; boric anhydride, 5·5. The same glass as (6) but the ingredients have been varied to avoid the use of so much boric anhydride, which is at present apparently difficult to obtain on a commercial scale.

Resistance Glass:—

(8) Sand, 65·5 parts; alumina, 2·5; magnesia, 5·0; zinc oxide, 8·0; sodium carbonate, 10·2; anhydrous borax, 13·0. A glass almost identical in its general behaviour with Jena resistance glass; withstands changes of temperature well, but, like Jena, is not suitable for working before the blowpipe. It darkens and tends to devitrify; operations such, for instance, as sealing side tubes into flasks—are difficult, if permanent and neat joints are required.

Formula No. 3 recommended for pharmaceutical purposes, ampoules, etc., may be substituted for the resistance glass with advantage, as the ampoule glass lends itself very well to blowpipe work, and is also especially resistant chemically.

Alternative for Combustion Tubing:—

(9) Sand, 72·0 parts; alumina, 10·0; calcium carbonate, 11·0; magnesia, 0·5; potassium nitrate, 3·0; sodium carbonate, 11·2; borax anhydrous, 7·2. This glass is capable of withstanding high temperatures and rapid changes of temperature; works well before the blowpipe and is free from the chief defect of Jena glass, namely, the readiness with which it becomes cloudy and finally quite opaque after prolonged use.

By slight modifications of this formula, almost any degree of hardness can be obtained.

In formulas (8) and (9) substances such as magnesia and zinc oxide can be added in the form of carbonates if the actual percentages of MgO and ZnO respectively present in the carbonates are known.

Soft Soda Glasses suitable for tubing and for X ray bulbs:

(10) Sand, 68·0; alumina, 1·0; calcium carbonate, 12·8; potassium nitrate, 11·5; sodium carbonate, 26·0; and

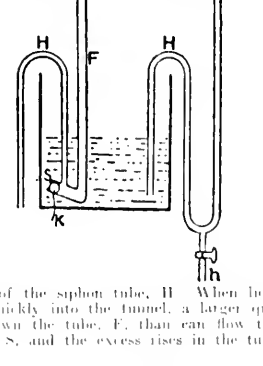
(11) Sand, 68·0; alumina, 1·0; calcium carbonate, 12·8; potassium carbonate, 10·0; sodium carbonate, 26·0. These glasses do not lose their easy working qualities after repeated heating and blowing, and are plastic over a long range of temperature. They require a temperature of at least 1400° to 1500° C. for complete incorporation of the ingredients in order to obtain that homogeneity which is necessary for resistance to rapid changes of temperature and ease of working before the blowpipe. No. (10), containing potassium nitrate, is considered the better of the two, and is more easily incorporated.

The committee is of the opinion that there is now information available for the manufacture of all the important glasses used in the laboratory and for industrial purposes, which have hitherto been mainly obtained from abroad."—*Journ. Soc. Chem. Ind.*, April 30, 1915, p. 424. (J. A. W.)

Contributions to Water Analysis.—"In testing for hypochlorite in potable waters sterilized by means of bleaching powder, two drops of methyl orange (1 : 5000) followed by 2 cc. to 3 cc. of 10% hydrochloric acid are added to 250 cc. of the water; hypochlorite effects decolorization in a few seconds. Ozone acts similarly. The test is sensitive to 0·1 mgm. of active chlorine per litre; methyl red is 20 times as sensitive as methyl orange. In presence of manganese the water must be cleared with alum (0·1 gm. per litre) before applying the test. Active chlorine is estimated by adding 0·2 gm. of very pure potassium iodide, 1 cc. of starch solution, and 2 cc. to 3 cc. of 25% phosphoric acid to 100 cc. of the water and titrating the liberated iodine with *N*/200 sodium thiosulphate; nitrite is destroyed by bleaching powder, and ferric iron does not liberate iodine in presence of phosphoric acid. Traces of manganese as small as 0·1 mgm. per litre may be detected by adding 1 drop of methyl orange to each of two portions (250 cc.) of the water; 1 cc. to 2 cc. of 10% sodium hydroxide is added to the first portion, and after a few minutes both portions are acidified with 5 cc. of 10% hydrochloric acid; in presence of manganese the first portion is decolorized within 2 minutes. The reaction cannot be applied in presence of hydrogen sulphide; nitrite renders it less sensitive; but it is unaffected by iron. Phosphoric acid is precipitated as ferric phosphate by warming the water (1000 cc. in some cases 5000 cc.) on the water bath for an hour with 1 cc. of 10% ferric chloride solution and 2 cc. of 10% alum solution. The flocculent precipi-

tate is collected on a moistened wad, and the flask and filter are washed three or four times with warm 30% nitric acid in quantities of 10 cc., whereby the precipitate is completely dissolved. After the addition of a drop of hydrochloric acid the filtrate is evaporated to dryness on the water bath, the residue dissolved in distilled water containing a few drops of nitric acid, the resulting solution filtered, evaporated to 5 cc., and treated with 5 cc. of molybdic acid reagent (prepared by adding 500 cc. of 20% nitric acid to a filtered solution of 25 gm. of commercial ammonium molybdate in 100 cc. of distilled water and 100 cc. of 10% ammonia and allowing to stand for several days). Even 0·05 mgm. of phosphoric acid produces a precipitate after some time. After standing overnight the precipitate is collected upon a wad, washed with 10 cc. to 20 cc. of the above reagent, and dissolved in the least possible amount of ammonia. The solution is made up with the distilled water washings to 50 (or 100) cc. and treated with 5 (or 10) cc. of the molybdic acid reagent. To 55 (or 110) cc. of distilled water contained in a similar beaker are added 1 drop of ammonia followed by a standard potassium chromate solution until the tint is matched. This operation must be accomplished rapidly. Twice the number of cc.'s of potassium chromate solution required gives the number of mgm. of P_2O_5 per litre of the original water. The gravimetric method yields high results for quantities of less than 5 mgm. P_2O_5 per litre, owing to the presence of molybdic acid in the magnesium pyrophosphate. Of six samples of potable water examined, one gave 0·02 mgm., four gave 0·05 mgm., and one gave 0·11 mgm. of P_2O_5 per litre."—L. W. Winkler. *Z. angew. Chem.*, 1915, 28, 22—23.—*Journ. Soc. Chem. Ind.*, March 15, 1915, p. 243. (J. A. W.)

Acid Siphons.—"Two patterns of acid-siphons constructed entirely of metal (lead or aluminium) are illustrated. Both are started by pouring some of the acid into the funnel at the top. In Fig. 1 the vertical charging tube, F, is of larger bore than the siphon proper and is bent at an angle at the bottom terminating in a hole, S, constricted to



the size of the siphon tube, H. When liquid is poured quickly into the funnel, a larger quantity passes down the tube, F, than can flow through the hole, S, and the excess rises in the tube, H, starting the siphon. A glass ball may be placed in the short branch to act as a check-valve in the hole, S, during starting, falling back again when the siphon starts. Fig. 2 shows a pattern constructed of aluminium for nitric acid, with a tap, *h*, for intermittent use without recharging."—W. Szigeti.—*Chem. Zeit.*, 1915, 39, 122.—*Journ. Soc. Chem. Ind.*, March 31, 1915, p. 278. (J. A. W.)

Detection of Bromide in the Presence of Thiocyanates, etc.—"Thiocyanates, cyanides and ferrocyanides interfere with the test for bromide even when excess of chlorine-water is employed. The neutral or slightly acid (with sulphuric acid) solution (containing, *e.g.*, 500 mgm. of any one of the above radicals) is heated just to boiling with saturated sulphur dioxide solution (15 cc.), and treated while hot with 2-*N* copper sulphate solution, which is run in until the supernatant liquid becomes blue. A greenish liquid in the presence of thiocyanate indicates insufficient addition of sulphur dioxide. The hot filtered solution and washings are concentrated to 5 cc. to 10 cc. in a conical flask, washed into a test-tube, and shaken with 1 cc. of 3-*N* sulphuric acid and 1 cc. of 1% potassium permanganate; 0·5 cc. of carbon bisulphide is then added and the mixture again shaken: a yellow colour in the carbon bisulphide layer indicates presence of bromide. Quantities of 2 mgm. of bromine were thus detected in the presence of 500 mgm. of the pure or mixed interfering radicals. The range of the above interference was estimated for each radical."—L. J. Curtman and A. G. Wikoff.—*J. Amer. Chem. Soc.*, 1915, 37, 298—301.—*Journ. Soc. Chem. Ind.*, March 31, 1915, p. 279. (J. A. W.)

Reduction Test for Tungstic Acid.—"Some simple prospectors' reduction tests for tungstic acid are described by G. Torossian (*Am. Journ. Sci.*, December, 1914, p. 537) from which two are selected.

Place the powdered substance on a piece of aluminium plate or foil; moisten with a drop or two of water; then add a drop or two of concentrated hydrochloric acid. A blue colour, which develops even in the presence of oxidizer and ferric oxide, shows tungsten.

Another test is: Place water in single big globules (not spread out) on aluminium foil; sprinkle a little of the finely powdered material on them, and add a little hydrochloric acid to the globules. A blue colour, developed in the presence of insoluble carbonaceous material, copper oxide, hematite, lead oxides, sulphur, rare-earth oxides or calcium fluoride, shows tungsten."—G. Torossian.—*Engineering and Mining Journal*, March 20, 1915, p. 537. ((G. H. S.)

Detection of Small Quantities of Hydrocyanic Acid.—"The Prussian-blue method is recommended, the authors having determined the conditions necessary to obtain a constant shade as well as density of colour from a given quantity of hydrocyanic acid. The distillate containing the hydrocyanic acid is rendered slightly alkaline with sodium hydroxide and evaporated under reduced pressure and below 70° C. to a volume of 1 cc.; 0·2 cc. of 3% ferrous sulphate solution and 0·05 gm. of potassium fluoride are then added, and the air is exhausted from the flask. After 10 minutes, the mixture is acidified with 30% nitric acid, and the blue coloration obtained compared

with a suspension of Prussian blue made from a know weight of potassium cyanide. The use of nitric acid or sulphuric acid is preferable to that of hydrochloric acid for acidifying the mixture, and the presence of potassium fluoride has a remarkably favourable effect on the formation and colour of the Prussian-blue. It is not necessary to add a ferric salt, and any considerable formation of ferric salts should be avoided."—A. VIEHOEVER and C. O. JOHNS.—*J. Amer. Chem. Soc.*, 1915, *37*, 601—607.—*Journ. Soc. Chem. Ind.*, April 15, 1915, p. 351. (J. A. W.)

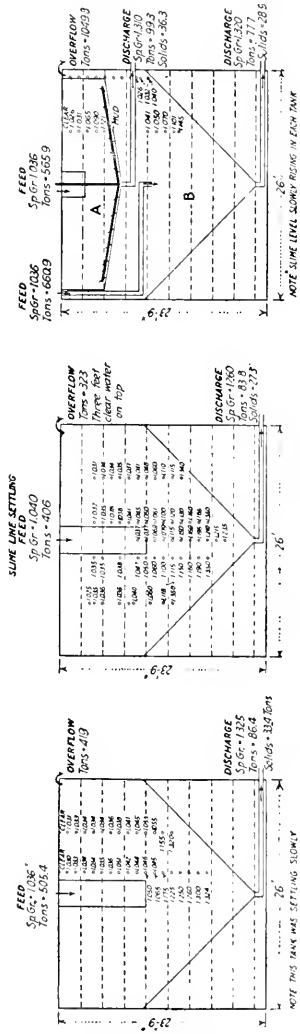
METALLURGY.

CONTINUOUS DECANTATION.—Increasing popularity of continuous decantation, with counter-current circulation of the cyanide solution, as applied to the treatment of slime, is evidenced by the manner in which it is being solved and adopted by Black Hills operators. Following the example of the New Reliance, the first plant in the district to be equipped for this work, the newly constructed Black Moon mill of the Mogul Company was constructed on similar lines. Now comes the Golden Reward, where, after considerable testing, the process is to be applied, and the Moore filter dispensed with. In preparation for the new process there have been installed in this mill a 24 by 16-ft. tank, with Dorr thickeners; a 24 by 16-ft. tank with Dorr agitator, and a similar sized tank equipped with a pipe agitator made on the premises. Tanks for the slime department as now arranged consist of four 20 by 16-ft., equipped with Dorr thickeners (two of these are steel, recently installed to replace wooden tanks), the two agitators as above, the Trent tank and the large thickener. As the slime at this plant amounts to less than 100 and over 75 tons daily, it is calculated that the equipment will be ample to fill the requirements. To handle the slime between the various tanks, diaphragm pumps are used, and in order to overcome the excessive wear on the driving gear, an objection to the pumps as delivered by the manufacturers, at this mill a new scheme of transferring the energy has been adopted. A frame of ½ by 3-in. angle iron is fastened to a small platform, at one end of which is the pump and at the opposite end, 5 ft. distant, is the drive shaft to which a pulley is attached. An eccentric on this shaft is connected by a rod to a walking beam, which in turn drives the pump. It makes a smooth-running rig, does away with the gears and pinions often used, and appears admirably suited to the purpose."—J. SIMMONS.—*Mining and Scientific Press*, April 10, 1915, p. 575. (G. H. S.)

SETTLING OF SLIME.—"The accompanying diagrams record determinations of the specific gravity of pulp in clarifying or thickening vats under certain definite conditions of operation. Whatever interest they may possess lies in the fact that few determinations of this nature have been recorded, rather than in any unusual or unexpected conditions developed by the determinations themselves.

Two types of vats are illustrated: One a simple hopper-bottomed type, automatic in action and discharge; the other a double-decked arrangement of a shallow Dorr thickener, mechanically discharged, superimposed upon a vat of the first type, which is necessarily modified as to the method of introducing the feed of pulp.

The samples were taken partly through a tube, ¼ in. in diameter, the lower end attached to a rod to insure against lateral deflection, inserted over the rim of the vat and operated as a siphon; and for greater depths and particularly for determinations in the lower member of the double-decked vat, through a ¾ in. pipe, equipped with the necessary fittings and connected with a small bicycle



Record of Slime Settlement and a Different Conditions

pump. In either case flow ceased when a specific gravity of 1·35 was reached.

The slime measured was derived from the crushing of oxidized ore. It contained much iron oxide and was exceptionally slow settling. Water was the suspending liquid, and there had been no lime or other alkali added. Moreover, the work was done during the winter and the temperature of the water was low.

Too much emphasis cannot be given to the fact that conditions of testing were not necessarily parallel. Although Test 3 illustrates practically synchronous conditions in the two compartments, it should be remembered that neither compartment (and more particularly the lower one) is often operated with the upper line of solids in exact equilibrium, but rather, as illustrated in the two series of observations on the single vat, either overloaded (Test 1) or underloaded (Test 2) with the clear water line either rising or falling in consequence." A. J. CLARK. *Engineering and Mining Journal*, Feb. 27, 1915, p. 412. (G. H. S.)

PLATINUM IN LITHARGE.—"Litharge (625 gm.) was reduced by flour (15 gm.) with a flux composed of 3 parts each of pure sodium and potassium carbonates, 1·5 of pure borax and 1·5 of pure sodium chloride. The buttons from 8 charges of the same sample, *e.g.*, from ½ kilo., were scorified together, the resulting button cupelled, and the remaining platinum, gold, and silver estimated. The amounts (*a*) of Pt, Au, and Ag found in three different samples of litharge, together with the amounts (*b*) found in the litharge formed during the scorification of these samples, are shown in the table.

Litharge from	Pt, mgm. per kilo.		Au, mgm. per kilo.		Ag, mgm. per kilo.	
	(a)	(b)	(a)	(b)	(a)	(b)
Freiberg	0·417	0·423	0·916	0·200	25·33	5·988
Frankfurt a/M	0·350	0·060	1·950	0·362	11·22	2·666
Mulheim	0·111	0·031	0·666	0·142	10·00	2·446

F. MICHEL. *Chem. Zeit.*, 1915, *39*, 6. *Journ. Soc. Chem. Ind.*, March 31, 1915, p. 285. (J. A. W.)

EXPERIMENTS WITH COLLOIDAL GOLD AND SILVER. "The mineral chalcocite (Cu_2S) and such metals as zinc, copper, tin, and cadmium precipitated gold from warm dilute (N/10) aqueous solutions of auric chloride as a dark brown or black coating. In the presence of an emulsoid solution of gelatin stable colloidal solutions of gold resulted, which were deep blue by transmitted and bright brick red by reflected light in the first three cases, while tin yielded "purple of Cassius" solution and cadmium gave a solution which was dark brick red by reflected and purple by transmitted light. In some cases the precipitation of the gold was similarly delayed in the presence of a colloidal solution of silica. On the other hand, minerals and metals such as pyrites, galena, and bismuth, which precipitated gold in the lustrous yellow or orange condition, failed to give colloidal solutions in the presence of the gelatin or silica solution. It is hence suggested that the dark brown or black gold represents a state of fine division approaching that of a coagulated colloid. Analogous results were obtained in precipitating silver from silver sulphate solutions. The author concludes that transport of these metals in the colloidal condition may occur in the process of primary ore deposition."—E. S. BASTIN.—*J. Wash. Acad. Sci.*, 1915, *5*, 64—71.—*Journ. Soc. Chem. Ind.*, March 31, 1915, p. 285. (J. A. W.)

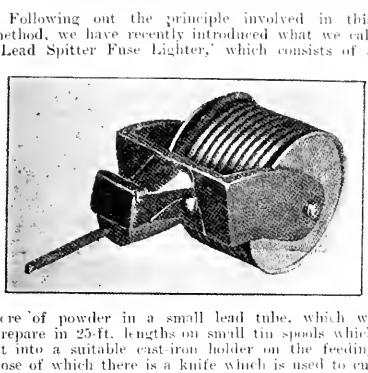
MONEL METAL.—"Monel metal is a natural alloy of approximately 67 per cent. nickel, 28 per cent. copper and 5 per cent. other metals, containing no zinc, tin or antimony. The alloy is natural in that the important elements—nickel and copper—bear the same homogeneous relation to each other in the refined metal as in the ore, the refining processes merely removing the undesirable elements. The mines from which the ore is obtained are located in Ontario, Canada, and are owned and operated by the Canadian Copper Company, a subsidiary of the International Nickel Company, who are the sole refiners of Monel metal. The ore is reduced to the form of matte at the smelter, and is then shipped to the Orford works of the International Nickel Company at Bayonne, New Jersey, where the process of refining takes place. The refined metal is then ready to be manufactured in the form of rods, castings, forgings, tubes, wire, strip stock, sheets, etc.—*Canadian Mining Journal*, July 1, 1915, p. 399.—(J. C.)

ETCHING REAGENTS AND THEIR APPLICATION.—"The principal reagents which have been found specially suitable for different metals and alloys are the following:—*Copper.* Ammonia, ammonium persulphate, bromine (with ammonia), copper-ammonium chloride. *Brasses.* Ammonia, ammonium persulphate, copper-ammonium chloride, electrolytic etching, ferric chloride, chromic acid, nitric acid. *Bronzes.* Ammonia, ammonium persulphate, ferric chloride. *Copper-aluminium alloys (aluminium bronzes).* Ammonium persulphate, ferric chloride, copper ammonium chloride, nitric acid. *German silver.* Ammonium persulphate, ferric chloride. *Nickel-copper alloys, monel metal, etc.* Electrolytic etching. *Gold and rich gold alloys, platinum and its alloys. Aqua regia* (dilute). *Aluminium and light aluminium alloys.* Sodium hydroxide, hydrochloric acid, hydrofluoric acid. *Lead, tin, and their alloys (white metal, etc.).* Chromic acid in nitric acid, ferric chloride, hydrochloric acid, nitric acid, silver nitrate. *Zinc and alloys rich in zinc.* Sodium hydroxide, iodine, Tmofeef's reagent (91 gm. HNO_3+6 gm. CrO_3), a few drops in 50 cc. of water. *Iron and steel.* Copper ammonium chloride, copper chloride, hydrochloric acid in alcohol, iodine, nitric acid in alcohol, picric acid in alcohol."—O. F. HUDSON.—*Inst. of Metals*, March, 1915.—*Journ. Soc. Chem. Ind.*, April 15, 1915, p. 362. (J. A. W.)

MINING.

SPITTING FUSES. "In correspondence and discussion in the *Journal* of April 17, 1915, we notice an item on Spitting Fuses. We are entirely familiar with the methods which are there referred to, and particularly the one recommended—the old-fashioned way of spitting with a piece of notched fuse. It is our observation that this method is still used to a large degree in metal mining and doby work, and we quite agree that it is the most effective method which has been in use until quite recently.

Following out the principle involved in this method, we have recently introduced what we call 'Lead Spitter Fuse Lighter,' which consists of a



core of powder in a small lead tube, which we prepare in 25-ft. lengths on small tin spools which fit into a suitable cast-iron holder on the feeding nose of which there is a knife which is used to cut off the spitter after the round of fuses has been lighted.

The operator in using this lighter simply draws out through the nose 12 or 15 in. of the lead spitter and ignites the end of it with his candle or match. It then burns at a steady, slow rate and throws a long spit from the end which easily and quickly ignites the safety fuse in wet weather, wet workings, or under other adverse conditions."—THE ENSIGN-BICKFORD CO. *Engineering and Mining Journal*, May 15, 1915, p. 874.—(G. H. S.)

MISCELLANEOUS.

MINE TOOL STEEL USED OVER AGAIN.—"An interesting example of the reclamation and re-use of scrap tool steel, and one which offers a suggestion of value to mine shop superintendents, is to be found in the experience of one leading company.

The steel includes four grades, No. 1 being used for machine of the higher speeds, No. 2 for taps, reamers, etc., No. 3 for chisels, punches, rivet sets and the like, and No. 4 for sheet metal and blacksmith tools.

All lathe and similar cutting tools are made from bars having the following dimensions: $1\frac{1}{2}\times3$, $1\frac{1}{4}\times2\frac{1}{2}$, 1×2, $\frac{5}{8}\times1$, $\frac{1}{2}\times\frac{1}{2}$ and $\frac{3}{8}\times\frac{3}{8}$ in.

As tools become too short or unfit for further service, they are placed in the scrap bin provided for the purpose, corresponding to the number on the tool. When a sufficient quantity of scrap tools has been accumulated, it is then sent to the smithshop, weighed, receipted for and placed in bins in sections labelled Grade 1, Grade 2, Grade 3, etc., corresponding to the grade of the scrap. This department is equipped with a furnace that will maintain a heat of 2500° F., with a reheating chamber attached suitable for annealing purposes. There is also a 1200 lb. steam hammer for the work of the department. This work should be handled by an experienced steel worker; otherwise the results will not be satisfactory.

When a sufficient quantity of scrap of Grade No. 1 has been accumulated, it is placed in the preheating chamber of the furnace, and allowed to heat slowly and uniformly to a dark red, about 1000° F.; the steel is then transferred, at the convenience of the toolsmith, to the other chamber and brought to a bright red heat, 1550°, and forged to the required size. Pieces $1\frac{1}{2}\times3$ in. are forged to $1\frac{1}{4}\times2\frac{1}{2}$; $1\frac{1}{4}\times2\frac{1}{2}$ in. to 1×2; 1×2 in. to $\frac{5}{8}\times1$; $\frac{5}{8}\times1$ in. to $\frac{1}{2}\times\frac{1}{2}$ and $\frac{3}{8}\times\frac{3}{8}$ in. sizes. After forging and dressing for service, the steel is placed in the annealing box and allowed to cool before hardening to relieve the forging strains. From 600 lb. of scrap steel thus treated there are usually 500 lb. of serviceable tools delivered to the tool room, at a reclamation cost of 5 cts. per lb. From the service rendered, they are the equal of tools made from the original bar; hence the reclaimed value of the steel after deducting the cost of reclaiming, 5 cts. per lb., is 15 cts. per lb.

The following actual examples are interesting: Three hundred and fifteen lb. of Grade No. 1 were worked into cutters for various of the heaviest. After these cutters had served to the limit, there were 250 lb. of scrap steel annealed and redressed for six $\frac{1}{2}$ in. cutters for boring bars at a reclamation cost of 5 cts. per pound, representing a reclaimed value of $112.50. After having served to the limit there were 180 lb. of scrap which was redressed and again used, the reclaimed value being $81. After the limit of boring tools was reached, there was on hand 150 lb. of pieces $\frac{7}{8}\times2\times2\frac{1}{2}$ in., which were sent to the smithshop and forged into lathe tools, $\frac{1}{2}\times1\times7$ in. long, with a reclaimed value of $56.25.

Chaser dies for $1\frac{1}{4}$ in. heads on pipe threaders and cutters were from $\frac{3}{8}\times1$ to $\frac{3}{8}\times5$ in. bars. After a number of re-cuttings for the $1\frac{1}{4}$ in. machine, there was on hand 200 lb. of scrap dies, which quantity was annealed and reworked for the 1 in. machine at a reclamation cost of 2 cts. per lb., or a reclaimed value of $95. After this service there were on hand 150 lb. of scrap steel, which were then worked into tool holder bits, of which there were 50 lb. $\frac{3}{8}\times\frac{3}{8}\times4$ in. at a reclamation cost of 5 cts. per lb., or a reclaimed value of $22.50.

From 786 lb. of Grade No. 2 steel placed in service at an original cost of 17 cts. per pound, there was 437 lb. of scrap annealed for reworking at a reclamation cost of 2 cts. per pound, with a reclaimed value of $65.55, the tools consisting chiefly of taps and reamers; the taps ranged in size from 4 in. in diameter down. Four-inch taps were reworked to the next smaller size, say $3\frac{3}{4}$ in. and so on down. On $1\frac{1}{2}$ in. taps for flexible staybolts $1\frac{3}{4}$ in. diameter, the threads were removed and used for reamers for the same purpose. In like manner, after having the threads removed, staybolt taps were used as reamers on boiler work preliminary to tapping for staybolts.

From 1147 lb. of Grade No. 3 steel put in service at 12 cts. per pound there were reclaimed 534 lb. at 2 cts. per pound, or a reclaimed value of $53.10, consisting of punches, punch dies, rivet sets, and flue expander pins. By the use of a punch post and nut to fit the spindle of a lathe, all punches were dressed to a smaller size on the point. All punch dies of standard and uniform dimensions were re-bored and faced to any convenient size needed. Expander pins were returned to the next smaller size. Rivet sets, due to the drawing of temper and losing shape in cup, were re-dressed again, hardened and replaced in service, with satisfactory results.

Thus it will be seen that by the systematic recovery of scrap tool steel, and following a plan of utilizing the material for the smaller sizes and shapes for which it is perfectly applicable, there is an opportunity in a mine repair and maintenance shop to save a great deal of tool steel.—GEO. E. EDWARDS.—*Mining and Engineering World*, July 24, 1915, p. 143.—(H. R. A.)

Abstracts of Patent Applications.

(C.) 164 15. Vilyn Clifton Tainton. Improvements in the electrolytic recovery of metals from their solutions and in apparatus therefor. 17.7.15.

This application is for apparatus for the carrying out of the process described in Patent 341 13, consisting of electrolytic cells in which the various members (anodes, cathodes, distance pieces, etc.), are combined in the form of a filter press. Two distinct designs are described and shown in the drawings.

(C.) 174 15. Lion Saint Denis Roylance. Ball-bearing gland. 28.7.15

This application relates to ball bearing glands, and especially to glands intended for use in agitating amalgamators.

The object of the applicant is to construct a gland so as to prevent the passing solution from entering the ball race. This he accomplishes by trapping and compressing air in the space occupied by the ball race.

The example given refers only to glands of the vertical type, and it involves a system of mechanical construction which deliberately introduces the defective element requiring the protection which constitutes the subject of the application.

(C.) 186 15. Harry Dalton Rankin. Improvements in method of recovering values from metalliferous material. 7.8.15

This application relates to improvements in the wet method of treating metalliferous material for its metal contents, and consists in a process where in nitric acid or its equivalent is employed in a suitable closed apparatus for controlling temperature and pressure to oxidize the acid forming material (such as sulphur) and produce acid salts (sulphate) of the contained metal; the nitrous fumes evolved in the operation are drawn off through condensers for subsequent reoxidation and re-use. Modifications of the process are given to deal with cases where the acid forming material (such as sulphur is insufficient or absent. The particular novelties claimed, apart from the regeneration of the nitric acid, are: Control of temperature below melting point of sulphur to avoid the formation of encrusted sulphur, and pressure to accelerate the reaction.

(C.) 185 15. Benjamin Augwin. Improvements in the utilization of compressed air for motive power purposes. 7.8.15

This application relates to a method of re-heating compressed air used for motor power by means of a jet of combustible vapour, introduced into the pipe leading to the air motor.

(C.) 191 15. Robert Alfred Lehfeldt. An improved process for the recovery of zinc from solution. 14.8.15

This application is practically fully described in Claim 1: "The process of recovering zinc which consists in electrolyzing a solution of zinc with a cathode consisting of or containing mercury or liquid zinc amalgam."

(C.) 210 15. Noel Vyv Robertson (1), Herbert Moss (2). Improvements in rollers, idlers or pulleys for supporting, hoisting, hauling and like ropes, conveyor belts and the like. 1.9.15.

This application refers to haulage rope rollers, conveyor idlers and similar devices.

(C.) 229 15. The Koering Cyaniding Process, Ltd. (1), Bruno Richard Koering (2). Improvements in process and apparatus for the recovery of metals from their ores. 15.9.15.

Claims 1 and 2 of this application refer to a process of treating ores in which a metal extracting solution is added to the said ores and tumbled together therewith so as to obtain the contact necessary to dissolve the metals therein contained. Simultaneously the solution is filtered and delivered to a precipitation device. The metal extracting solution is repeatedly passed through the filter medium.

Claims 3 to 15 deal with the apparatus necessary to carry on the process and describe its construction and uses.

(C.) 238 15. Henry Candy (1), William Young (2). Improvements in stamp guides. 21.9.15.

This application refers to a form of guide for stamp mills, consisting of metal bushings in halves, one-half being bedded into a timber guide block and the other kept in position by a metal cap or strap fitting round the half bushing and secured by two bolts passing through the timber guide and the guide gut with a solid square collar sunk in the front of the timber guide.

It is claimed that this construction enables any one stem to be taken out without disturbing the remainder.

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Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DETERMINATION OF ZINC AS A DOUBLE SULPHO- obtained by the action of the vapour of silica or silicon on carbon; the product retains the original shape of the carbon, which is penetrated to a depth of about ½ in. The appearance and character of the product vary with the temperature of production, but non-crystalline (yellow) and crystalline (steel black) varieties, obtained by heating carbon in the amorphous and crystalline zones of a carborundum furnace (*i.e.*, at 1150°—1820° C. and 1820°—2220° C., respectively) were both found to consist essentially of the same silicon carbide as carborundum, viz., SiC. By the combined action of silicon vapour and molten silicon, carbon is silicidized in the same manner as when placed in a carborundum furnace; but the resulting product is carborundum containing varying proportions of pure silicon, these two components being mutually soluble in all proportions. In considering the difference between silfrax and ordinary carborundum, the reactions of the carborundum furnace are discussed, the author being of opinion that the equation, $SiO_2+3C=SiC+2CO$, explains only the conversion of carbon particles into carborundum *in situ*, but not the formation or growth of crystals larger than the original carbon particles, since at the temperature and vapour pressure of the carborundum furnace, carbon is generally regarded as having no appreciable vapour tension. The equation, $3Si+2CO=2SiC+SiO_2$ is suggested to account for all the phenomena of the furnace, the forward or low temperature reaction (which is exothermic and occurs throughout the crystalline zone) explaining the building up of the large from the small crystals formed by the first equation, and the reverse reaction (occurring in the hottest part of the furnace and next to the core) explaining the phenomena of the local disappearance and re-crystallization of carborundum. On these grounds, silfrax is regarded as differing fundamentally from carborundum in its mode of production, being formed mainly by the action of silicon vapour on carbon *in situ* and not from gaseous carbon compounds. In the production of silfrax, the density of the original carbon article is such that when the initial coating is formed the pores are almost completely filled with silicon carbide, so that further circulation of gases is made very slow and difficult. Hence the vapour reaction indicated in the second equation cannot occur to any extent, and the crystals remain small and closely compacted so that the product acquires toughness and mechanical strength superior to those of carborundum. Silfrax is used commercially as a refractory material for the manufacture of pyrometer tubes, electrical heating elements, and chemical ware."—F. J. TONE.—*Trans. Amer. Electrochem. Soc.*, 1914, *26*, 181—198.—*Journ. Soc. Chem. Ind.*, April 15, 1915, p. 352. (J. A. W.)

DETERMINATION OF ZINC AS DOUBLE SULPHOCYANIDE.—"In a recent paper before the American Institute of Metals (Chicago meeting, September, 1914), G. E. F. Lundell and Nai Kim Bee suggested as a method for the analysis of brass and other alloys the precipitation of zinc with potassium-mercuric sulphocyanide. The precipitate has the composition $ZnHg(CNS)_4$ and may be dried at 100-108° C. and weighed.

The precipitating agent is made by dissolving 39 gm. of KCNS in 200 cc. water and then stirring in 27 gm. of $HgCl_2$, slowly diluting to 1000 cc. as this is done. If a residue remains, filter this solution.

Prepare the zinc solution to be assayed as though the pyrophosphate method were to used, although the acid present need not be carefully neutralized. However, the solution should not contain nitrous acid nor over 5% by volume of free acid. Then add 25 to 30 cc. of sulphocyanide reagent for each 100 cc. of the solution. After adding the reagent, set the solution aside for from five to 30 min., during which time it should be well stirred three or four times. Filter on a gooch, dry at 100-108° C. and weigh

Wt. × 0·1266 = Zn.

The interfering substances are Hg′, Cu, Bi, Cd, As′″, Co, Mn and large amounts of Ni. These give high values. K_2SO_4, $NH_4C_2H_3O_2$ and NH_4CNS give low results, as does over 10% of free acid. Alkaline solutions cannot be used. H_2S and oxides of nitrogen are undesirable. The solution must be filtered cold. The precipitate is slightly soluble in water, but practically insoluble in water containing 2% of the precipitating solution; it begins to decompose at 110° Centigrade."—G. E. F. LUNDELL and N. K. BEE.—*Engineering and Mining Journal*, April 17, 1915, p. 701.—(G. H. S.)

RAPID ANALYSIS OF ALLOYS FOR TIN, ANTIMONY AND ARSENIC.—"The following methods have been used for six years with satisfactory results:—*Alloys.* 0·5 gm. of fine shavings or drillings is dissolved in 8 cc. of concentrated sulphuric acid, the solution is boiled, cooled, treated with 5 cc. of cold water, cooled, and 20 cc. of concentrated hydrochloric acid and about 0·5 cc. of granulated pumice are added. The solution is

then distilled gently for 12-15 minutes, the temperature being kept at 107° C. (bulb of thermometer 1 in. above surface of liquid) for at least 5 mins.; the exit tube for the vapours is bent downwards and connected with an S-tube, one bend of which is nearly filled with water and submerged in cold water, whilst the free end dips into about 75 cc. of water in a beaker. The distillate, amounting to about 200 cc., is treated with an excess of about 2 gm. of sodium bicarbonate, warmed to 27° C., and the arsenic determined by titration with iodine solution (1 cc. = 0·010672 gm. I or 0·00315 gm. As), in presence of starch, to a blue tint corresponding to that obtained with 0·2 cc. of iodine solution in a control determination. The residue in the distilling flask is diluted with about 130 cc. of water and titrated with permanganate solution (2·6315 gm. per litre; 1 cc. = 0·005 gm. Sb). The solution is then, if necessary, mixed with sufficient antimony trichloride to bring the antimony content up to 0·07 gm., 6 cc. of concentrated sulphuric acid, 60 cc. of concentrated hydrochloric acid, and about 6 ins. of clean, soft No. 14 pure iron wire are added, the solution is boiled gently for ½ hour, and after adding 4–6 ins. more of iron wire, again boiled for ½ hour. A rubber stopper is placed loosely in the neck of the flask, and, after two minutes, the flask is stoppered tightly, cooled in water, the solution transferred to a beaker, the flask rinsed with 150 cc. of recently boiled water, and the solution (300–350 cc.) titrated with the iodine solution (see above: 1 cc. = 0·005 gm. Sn) to a blue colour corresponding to that given by 0·1 cc. of the iodine solution. When more than 3% Cu is present 0·5–1 gm. of potassium iodide crystals is added just before titrating the tin.

Mattes, soluble furnace products, chilled slags, etc.—Antimony and tin may be determined directly and rapidly by the following method, even when much copper, iron, and arsenic are present: 0·5 gm. of the sample is heated for at least 15 mins. with 10 cc. of concentrated sulphuric acid; if much silicon is present 15 cc. of water is added before the acid. The cooled solution is treated with 10 cc. of water and 10 cc. of hydrochloric acid, boiled for 2 mins., treated with at least 1·5 gm. of potassium chlorate, heated to expel chlorine and chlorine compounds, treated with about 0·1 gm. of sulphur, and boiled for ½ hour. After cooling, the solution is diluted with 6 cc. of water, cooled, treated with 20 cc. of concentrated hydrochloric acid and some pumice, boiled to expel arsenic, and antimony and tin determined as described above. If much carbon is present, nitric acid is used in place of potassium chlorate, and somewhat more sulphur is added subsequently. Slags or ores containing insoluble stannic oxide or silicate must first be fused with caustic soda." F. A. Smith. *J. Ind. Eng. Chem.*, 1915, 7, 211–212. *Journ. Soc. Chem. Ind.*, April 30, 1915, p. 428. (J. A. W.)

Purification of Drinking Water by Means of Calcium Hypochlorite.—"Calcium hypochlorite possesses undoubted practical advantages over solutions of alkali hypochlorites for sterilizing water. It is conveniently used in the form of compressed tablets, each containing 0·015 grm. of calcium hypochlorite (= 3·5 mgm. available chlorine) and 0·08 gm. of sodium chloride, the latter being added to promote the diffusion of the active chlorine into the water. Each tablet is sufficient for 1 litre and is added to the water without previous crushing. After 20 minutes practically the whole of the chlorine has diffused into the water and merely a skeleton of calcium carbonate remains. The tablets keep quite well, a diminution of only 0·0003 gm. of active chlorine being observed within 2 months. Drinking water sterilized in this way can be consumed 20 minutes after the addition of the tablet. Its mineral composition is thereby scarcely altered, and its taste is inappreciable. Ammoniacal and albuminoid nitrogen were found to diminish by one half to three-quarters after the sterilization with the hypochlorite tablet, whilst all pathogenic organisms, including *Bacillus coli*, were killed in 10 to 12 minutes, even in the pressure of much organic nitrogen."—H. Vincent and Gaillard.—*Comptes rend.*, 1915, *160*, 483—486.—*Journ. Soc. Chem. Ind.*, May 31, 1915, p. 569. (J. A. W.)

Penot's Method of Analysing Hypochlorites.—"All hypochlorite solutions contain chlorites, and these react with hypochlorous acid to form chlorates and chlorine peroxide. This accounts for the lower results (about 2%) obtained by Penot's method of titration with sodium arsenite solution with starch-iodide paper as indicator, as compared with those calculated from the amount of gas liberated by interaction with hydrogen peroxide, or by Mohr's modification of adding an excess of arsenite and titrating with standard iodine solution. The following modification gives correct results:—The volume of arsenite solution required in Penot's original method, is added at once to the hypochlorite solution: the liquid will still give a blue colour with the starch-iodide paper, and the titration is completed with the standard arsenite solution till the paper ceases to turn blue."—J. Clarens. *Ann. Chim. Analyt.*, 1915, *20*, 81—84.—*Journ. Soc. Chem. Ind.*, May 31, 1915, p. 547. (J. A. W.)

METALLURGY.

Testing Oils for Flotation.—"It has long been recognised that a well-equipped experimental testing laboratory is necessary for the successful working of a flotation concentrating plant. Of the many various tests which are required from time to time, the most frequent and perhaps the most important is the testing of oil, or active floating medium. The following remarks refer chiefly to eucalyptus and resinous oils:—

The first material necessary is a standard ore sample. For the purpose of oil testing, a thoroughly representative sample of the material to be treated is dried, crushed to pass 60 mesh and bagged. For convenience, a supply ready for use may be weighed off in 1 lb. lots and put up in small tins.

A sulphuric acid solution, containing 405 gm. of H_2SO_4 per litre, is generally used, 1 cc. of such a solution containing 2 lb. of pure acid per ton, when working on 1 lb. of ore sample.

A standard oil sample is that oil which has been found to fully meet the requirements of the proposition, upon which all future calculations are based and comparisons made. It may be stored ready for use in bottles.

Preliminary Examination. For specific gravity tests hydrometers reading to 0·001 are required. In all cases it is necessary to ascertain the specific gravity of the oil, with the view, at least, to future calculations. This may be carried out at any suit-

able temperature which has been fixed upon as standard. It has been found advisable to check the specific gravity of the standard oil simultaneously because of the gradual increase in specific gravity which takes place owing to the loss of lighter oils by volatilization. A correction for temperature is made by allowing 0·00015 for each degree Fahrenheit.

A small burette is used for counting the number of drops in 1 cc. of the oil, also for admitting the oil to the machine during testing operations. The greater the number of drops delivered by the burette, the greater is the accuracy of the test. To obtain a suitable dropper, cut a burette about 8 in. above the cock, almost close the discharge orifice by dumping up the glass with a blowpipe flame, then grind the outside back to a point so that a minimum surface is presented to the oil drop. The burette should at normal temperatures give between 80 and 90 drops per cubic centimetre when run at the rate of 1 drop per second. The temperature of the oil during the dropping test should correspond with the temperature during the flotation test.

Having obtained the number of drops per cubic centimetre and the specific gravity, it is easy to calculate the number of pounds of oil per ton of ore, when working on 1 lb. of sample.

$$\text{Thus} \frac{2240 \times sp.gr.}{453{\cdot}6 \times drops\ per\ cc.} = lb.\ of\ oil\ per\ ton.$$

It is sometimes desired in practice to use a mixture of oils. When the oil under examination is to be used in conjunction with other oils, these should be wholly miscible in the proportions in which they are to be used.

The following classification and explanations will serve to give a general idea of the methods employed when carrying out various tests: (1) Flotation of lead, zinc and other sulphides, as a mixed concentrate; (2) differential separation, or selective flotation of one sulphide in the presence of other sulphides (the term 'differential separation' is usually applied to the selective flotation of lead sulphide from zinc and other sulphides); (3) flotation of copper and iron sulphides.

Outline of Test Process.—The testing of oils in the laboratory is carried out by comparing measured quantities (from 3 to 6 drops) of a standard oil, with a similar quantity of the oil under examination, the values being arrived at by comparing the results obtained from each series of tests. Tests are usually made on 1 lb. of standard ore sample in 4 lb. of water at a standard temperature, acidulated with a definite quantity of sulphuric acid. The oil then being admitted, the mixture is agitated in a specially constructed agitator machine, the principle of which is dependent on the object of the test. The float produced is skimmed off, dried, weighed and assayed.

Test 1.—The Flotation of Mixed Sulphides.—Almost any eucalyptol oil which produces a persistent froth and leaves a gummy residue on evaporation is suitable for this class of work. An agitating machine may be constructed by cutting a packing bottle about 10 in. above the neck (a bell jar of suitable dimensions can be obtained). Fit four copper baffles, 4×1½ in. wide, to a copper band of the same width and push this arrangement hard down into the bottle (the band being first bent to fit the inside circumference of the bottle). The lower ends of the baffles will jam hard at a point where the concave glass begins. The band is then expanded hard against the glass and held in position by soldering the separated ends. The mouth, or discharge end, is closed with a rubber stopper, through which is passed a glass or metal tube fitted with a short rubber tube and clip. The bottle with the baffles in position is inverted and clamped centrally under two pairs of suitable bearings, which carry a ½ in. impeller shaft. At the upper end a four-bladed impeller which just has clearance between the lower points of the baffles and the glass. The blades of the impeller have a lateral angle of about 45° and should be driven at about 1200 r.p.m. in a lifting direction.

Test 2.—Differential Separation.—For differential separation, an oil high in phellandrene which leaves a gummy residue on evaporation is used. Phellandrene may be tested for by a polariscope. Differential separation is worked in acid and neutral and in hot and cold liquors, and being still in its infancy, allows of many types of machines and schemes. Each different ore requires some modifications, but the principal in main is the addition of medium and aeration from below, which is effected by air jets or suction created by the impeller.

Test 3.—Flotation of Copper and Iron Sulphides.—An oil which gives a deflection by the polariscope of 60 or over is considered sufficiently high in phellandrene for use in copper flotation. Tests are usually made with the apparatus described in Test 1, using cold circuit liquors made slightly acid. In practice the mine water usually contains sufficient acid for the purpose."—J. Coutts.—*Engineering and Mining Journal* (and *Australian Mining Standard*, April 8). June 19, 1915, p. 1079 (G. H. S.)

Recent Progress in Flotation.—"In the *Engineering and Mining Journal* of May 29, 1915, recent progress of flotation was discussed by O. C. Ralston and F. Cameron. After an interesting review of various subjects pertaining to flotation methods, the flotation of nonsulphide minerals was discussed. The writer has obtained a basic patent on the sulphidizing of nonsulphide mineral for purposes of flotation in 1905, and has since then done extensive work in this line, some of which is of recent date, and has not been published. Messrs. Ralston and Cameron make a number of statements in their article which might unintentionally create the impression that only low-grade concentrates can be obtained by this process. Copper concentrates assaying 23·40%, 24·60%, and 43·40% in copper, which I have made, can hardly be called low grade. Nor should lead concentrates assaying from 53% to 71·50%, obtained from lead-carbonate ores, be classed as low grade. In the latter case extractions of 76% can be made commercially without difficulty. Zinc concentrates obtained from carbonate and oxide ores, some of them carrying some sulphide, assaying 52% to 57%, can also be classed as good smelting products, and commercial extractions of 80% and better are obtainable when the proper methods are employed. It might be well in publishing definite conclusions to state under what conditions such tests were made. The flotation of nonsulphides is a field even more difficult than the flotation of sulphides, and surely those who have not made exhaustive tests on the subject might obtain poor results, while when the proper conditions are provided excellent results may be obtained both in the way of getting high-grade concentrates and good extraction. It is here

pointed out in the previously mentioned article that a great deal of gangue, especially iron, is carried up. This also is quite contrary to my experience. The H_2S process lends itself particularly well to selective flotation more so than any other method I have known, and I have separated complex ores with great success. The application of the H_2S process is particularly important for the flotation of those sulphide ores which either carry a percentage of carbonates and oxides or else do not lend themselves well to flotation. This is invariably due to the fact that the sulphides are slightly coated with monsulphides. Metallic copper and native silver are susceptible to this treatment, and the writer begs to differ with Messrs. Ralston and Cameron in their estimation that the present outlook limits the application of the H_2S process to copper ores. It rathers seems that the injection of H_2S gas, which is inexpensive and can be made from waste products which are usually on hand, and the fact that this process can be used in connection with any suitable method of flotation, should secure for it almost universal application." A. SCHWARZ.—*Engineering and Mining Journal*, June 19, 1915, p. 1084. (G. H. S.)

NICKEL-PLATING ALUMINIUM.—"The difficulty of electro plating aluminium with nickel lies not in devising a suitable electro chemical process, but in securing a coherent deposit. An intermediate coating has generally been considered indispensable between aluminium and nickel, and copper, zinc and iron have been proposed and tried in various ways. In the first place, however, a chemically clean surface must be secured on the aluminium and to the well known difficulty of obtaining this must be attributed failure to secure a deposit which will stand mechanical working. A new process described by M M. Canac and Tassilly before the Société d' Encouragement pour l'Industrie Nationale, is claimed to overcome this defect. The aluminium piece to be plated is passed first through a boiling potash bath, then brushed with milk of lime, dipped in a 0·2 potassium cyanide bath for several minutes, and finally subjected to attack by an iron chloride solution (500 gm. hydrochloric acid, 500 gm. water, 2 gm. iron), till the appearance of the aluminium resembles crystallized tinplate. After each stage in this treatment the work is thoroughly washed in water. The following is found to be a satisfactory nickelling solution :— Water, 1,000 cc.; nickel chloride, 50 gm.; boric acid, 20 gm. Worked at 2·5 volts and 1 ampere per sq. dcm. The deposit obtained is matt-grey on leaving the bath, but easily takes a good polish under the scratch brush. Above all, it is claimed that the coating will stand bending and hammering without peeling or cracking. The reason for the adherence of the deposit appears to lie in the strong reducing action of the nascent hydrogen evolved when the aluminium is plunged in the ferrous chloride pickle. Metallic iron is deposited on the aluminium and, though its amount is so small (0·25 to 0·5 gm. per sq. metre) that it cannot form a continuous intermediate deposit, the iron forms with the aluminium a number of tiny couples favouring attack by the acid pickle. The surface of the aluminium is thus very minutely but very completely pitted so that, besides being deposited on the nearest practicable approach to a chemically pure surface, the nickel coating is actually locked or "rooted" into every part of the aluminium surface, and it is on this mechanical action that the tenacity of the deposit depends. Aluminium thus protected is said to be unaffected by moist air, dilute and boiling soda, acetic acid, sea salt, wines and foodstuffs, and does not permit leakage of petrol. It is recommended as useful for electrical conductors and is claimed to extend greatly the possible applications of aluminium.—*Ind. Engineering*, June 12, 1915, p. 347. (J. A. W.)

MISCELLANEOUS.

ACCIDENTS FROM POISONOUS ASPHYXIATING GASES IN MINES.—*"Causes of Gas Poisoning in Coal and Metal Mines.*—Accidents from asphyxiating gases are much more common in coal mines than in metal mines. The risk of the occurrence of explosions and fires is the peculiar danger of coal mining, a danger due to three well known facts—first, that many coal measures give off an inflammable gas ('fire damp'), which when mixed with air in certain proportions is explosive on ignition; second, that the fine dust of bituminous and lignite coals, when suspended in air as a dust cloud, is also explosive when ignited, even without any inflammable gas being present; and third, that coal itself is not only combustible, but is capable of undergoing spontaneous combustion.

The important abnormalities and impurities which may be found in the air of mines are the following :—

1. Deficiency of oxygen.
2. Carbon dioxide, CO_2.
3. Methane, CH_4 ('Fire damp').
4. Carbon monoxide, CO.
5. Nitrous fumes, mainly NO_2.
6. Sulphuretted hydrogen, H_2S, and sulphurous acid, SO_2.

Apart from nitrous fumes poisoning, which only occurs in the use (or abuse) of explosives, the danger to life in accidents from gas-poisoning in mines arises almost entirely, either from a deficiency of oxygen in the atmosphere breathed, or from the presence of carbon monoxide.

Let me illustrate this general statement by a brief reference to the main circumstances in which one or other of these conditions may be met with.

(1) In cases of asphyxiation due to serious local vitiation of mine air by 'black-damp' or by 'fire-damp,' it is the deficiency of oxygen in such atmospheres which constitutes the actual danger. This is obviously so in the case of fire-damp, since methane is itself not a poisonous gas.

'Black-damp' is simply the residual gas resulting from processes of oxidation occurring in coal, minerals, timber, etc., when exposed to air. It is almost universally present in some amount where air penetrates underground, as in mines or wells, and is the commonest impurity in the air of coal mines. But it is very frequently present in the air of metal mines as well, sometimes in quantities sufficient to cause fatal gassing. Pure black-damp consists of nitrogen with varying proportions of carbon dioxide, the amount of the latter gas depending on the nature of the substance undergoing oxidation. Thus there may be as much as 20 per cent. of carbon dioxide, or there may be practically none.* In air gravely contaminated with black damp there is therefore always a deficiency of oxygen, and there is commonly an excess of carbon dioxide. It is seldom, however, that carbon dioxide is present in sufficient quantity to

* *Pure* carbon dioxide, unmixed with other gases, is very rare in mines, but large outbursts have occurred in some French coal mines

be itself a source of danger, and, consequently, although the immediate warning symptoms of breathlessness, panting and dizziness, which occur on exposure to black-damp, are usually due to the carbon dioxide present, the real danger to life commonly lies in the possibility that in atmospheres heavily charged with black-damp there may be a serious deficiency of oxygen also.

If the oxygen percentage in air falls to 17 or 16 the flame of a candle or oil lamp will be extinguished, but an oxygen deficiency of this amount will have no effect on men breathing such air. Unless a man is exerting himself, symptoms of embarrassment do not appear until the percentage has fallen to about 10. At this point exertion becomes more difficult, and is apt to produce dizziness and shortness of breath.

At 7 per cent., *i.e.*, when the oxygen percentage has fallen to about one-third its normal amount, symptoms become urgent. There is panting, palpitation, and headache, with mental confusion and great impairment of muscular power. At a slightly lower percentage complete loss of consciousness occurs, often suddenly, and there is then imminent danger to life. Death will quickly ensue if the unconscious man is not brought to fresh air. On the other hand, if this is done, recovery also is rapid.

The condition known as 'mountain sickness' is exactly comparable to that produced by simple deficiency of oxygen in mines.

It is most dangerous to penetrate into atmospheres seriously deficient in oxygen, unless special precautions are taken to test the air. If the deficiency is very great from the outset, rapid disablement and unconsciousness will occur. But if it is not, the first symptoms produced are slight and insidious, so that a man may go some distance into such an atmosphere, without realising his danger, until it is too late to effect an escape. The extinction of lights, and, when carbon dioxide is present, the preliminary symptoms produced by that gas, will give warning of oxygen deficiency, but *they give no measure of its amount*. Therefore, although an atmosphere which is extinctive to lights may be perfectly safe to breathe, both signs are to be regarded as urgent danger signals.

Sometimes there may be little or no carbon dioxide in black damp, and in such cases the warning given by the effects of that gas will be absent.

(2) On the other hand, in explosions of fire-damp and coal-dust in coal mines, and in mine fires, nearly all the deaths are due to poisoning by the carbon monoxide, present in the 'after-damp' produced by explosions, and in the fumes from fires. Direct physical injury accounts for only a small percentage. Carbon monoxide is therefore also the great source of danger to rescue and exploration parties. Carbon dioxide is also present in considerable quantities in these circumstances, but not in sufficient amount to be in itself dangerous to life. Sulphuretted hydrogen, and sulphurous acid, both powerful poisons, are also found in small quantities in the air in underground fires and after explosions. But they are practically never present in dangerous amounts, although they may cause smarting of the eyes and throat and irritation of the air passages.[2] 'Heating' of coal is also a source of carbon monoxide which may cause trouble.

(3.) In accidents due to the fumes of explosives carbon monoxide again accounts for most of the deaths, but nitrous fumes poisoning may also result, and is the characteristic form of poisoning in accidents due to the inhalation of the fumes of burning nitro-glycerine explosives.

(4) These are the commonest and most important causes of accidents which produce gas poisoning in mines, but other rare conditions contribute a small proportion. In mines where compressed air is used underground, for example, accidents have from time to time occurred from overheating of the air compressors, and consequent 'firing' of carbonised oil residues. The air distributed to the faces in these circumstances may become charged with dangerous quantities of carbon monoxide. In one accident of this nature, which occurred some years ago on the Rand, ten lives were lost from carbon monoxide poisoning. The fumes of hydrocyanic acid, given off from the wet sands in the process of sand filling in gold mines, have also been the cause of occasional accidents, but so far as I know only one death has been traceable locally to this cause.

Gassing Accidents Arising from the Fumes of Explosives.—With a very few exceptions, due to one or other of the rarer causes just mentioned, all of these accidents arise from the fumes of nitro glycerine explosives, of which very large quantities are employed daily in mining operations. The explosives in ordinary use are blasting gelatine, gelatine dynamite, and gelignite. The theoretically perfect detonation of these explosives should produce no injurious gas except carbon dioxide, but in ordinary mining practice a theoretically perfect detonation is, to say the least, quite exceptional, and the ordinary blast produces in addition varying proportions of the much more dangerous gases, carbon monoxide and 'nitrous fumes.' It may in fact be regarded as certain that a dangerous amount of carbon monoxide is not only an occasional but practically an invariable accompaniment of the use of explosives in the mines of the Rand, and that nitrous fumes are also not infrequently present in dangerous amounts. Where definite partial burning of the explosive takes place (the so-called 'soft shot'), the amount of carbon monoxide and of nitrous fumes is proportionately increased. Imperfect detonation may be caused by the use of detonators, which are not powerful enough for the amount of explosive used, or which have become deteriorated by damp through temporary storage underground, or by a disproportion between burden and charge, or by faulty methods of charging and firing. Finally, when the explosive is not detonated at all, but is accidentally set on fire and burns, large quantities of these dangerous gases are evolved.

Fumes produced by blasting in the end of a drive may drift along and settle in the nearest winze, and there give rise to 'gassing,' although no blasting may have taken place recently in the winze. This possibility must always be borne in mind.

Carbon Dioxide.—With regard to the part played in gassing accidents by the three gases mentioned, that of carbon dioxide may, I think, be readily disposed of.

First let me say that I know of no evidence that any important amount of black damp is in general found in the mines of the Rand. In the records of gassing accidents extending over four years (1904-1907) I could find none which pointed to vitiation of the air by black damp as the cause. The instance which I am about to quote, which

was of more recent occurrence, shows, however, that accidents due to this cause *may* occur under exceptional conditions. The circumstances were these :

Case 1. Poisoning by Black Damp. 5–6–13.—A native wishing to shirk his work ran up a raise. A little stoping had been done from the raise some considerable time before, but since then the place had lain entirely idle, and the inspector stated that fumes from blasting in neighbouring workings did not pass the foot of the raise. The place was roughly in the shape of an inverted bottle, some 50 feet in vertical height

The native was seen at 9 a.m. and missed at 10 a.m. Another native then went up after him for 20 or 30 feet, but his lamp was extinguished. He came down and recharged it, but on his going up again it was once more extinguished. He went back and called a white miner, who went up the raise with the native, taking his acetylene lamp. This also went out. He sent the native for a lighting stick, and they returned with another miner. The later's acetylene lamp was also extinguished.

The first miner reached the boy, and endeavoured to pull him out, but was himself overcome by the gas. He made another attempt to reach him, but became unconscious, and was carried out to the station, where he revived. A shift boss with fresh boys then went up, about 12.15. They reached the native, who was dead, and brought him out.

This is not the account of an ordinary gassing accident. No blasting had recently taken place in the raise, nor was the place exposed to vitiation from that cause. There are two important points to be noted. First, acetylene lamps were repeatedly extinguished. The flame of a candle or oil lamp will be at once extinguished if the oxygen percentage falls to 16 per cent., and at a rather higher point, say 17.5 per cent. more or less, if carbon dioxide is present also. But the flame of an acetylene lamp is much more 'tenacious,' and is not extinguished until the oxygen percentage has fallen to about 10. Secondly, the white miner attempting rescue was reduced unconscious *almost immediately*, and the native died within a short (but undefined) time.

These facts point to the conclusion that oxygen deficiency was the factor which produced the fatal results.

Such an occurrence is, however, quite exceptional, although it certainly suggests that, in 'gassing' accidents from the fumes of explosives, some amount of black damp may sometimes be present also, when the working has lain idle for more than the usual periods. But it does not affect the general statements which follow.

The results of the analyses of mine air after blasting show that carbon dioxide is never present in amounts sufficient to produce unconsciousness or death by itself. One may safely say that carbon dioxide *taken by itself* is never the cause of fatal accidents in the mines of the Rand. The history of these accidents, and the results of post-mortem examinations, are also in complete agreement with this conclusion.

On the other hand, this gas is probably not infrequently present in quantities sufficient to produce symptoms, and to aggravate or accelerate the effects of any carbon monoxide present.

Carbon dioxide acts as a narcotic poison. But it is not a highly poisonous gas. The presence of 5 or 6 per cent. will produce marked panting. At 10 per cent. the distress is very great.[1] But it requires considerably over 10 per cent. to produce unconsciousness, and even so unconsciousness produced by this gas is a much less dangerous condition than that due to carbon monoxide or oxygen deficiency, and recovery may take place after prolonged exposure.

It is certain that deficiency of oxygen combined with excess of carbon dioxide may frequently be present in the atmosphere of close places after blasting in a degree sufficient to extinguish the flame of a candle. The warning given by this occurrence should never be neglected, for although the vitiation which directly causes it may not in itself be dangerous, the extinction of lights give warning of the probable presence of dangerous quantities of carbon monoxide also.

There remain, therefore, the two gases, carbon monoxide and nitrous fumes, as the main active agents in causing 'gassing' accidents.

The former gas, as has been said, is practically invariably produced in dangerous quantities after ordinary blasting, and the latter is also not uncommonly present in dangerous amount, especially where there has been a partial burning of the explosive.

One might expect as a consequence that cases of mixed poisoning by both gases would frequently occur, but in the great majority of cases one or other type is markedly predominant. The symptoms in each case are quite distinctive and may now be described.

Carbon Monoxide Poisoning.—As is well known carbon monoxide is colourless and has no smell. It may be present in dangerous quantities in air in which a candle will burn quite well, unless at the same time there is a great enough deficiency of oxygen and excess of carbon dioxide to affect the flame. Neither the miner's lamp nor his senses give indication of its presence. Hence the victim may have no warning, and may go on working and inhaling the gas, or may penetrate far into a dangerous atmosphere, without being aware of danger until it is too late to effect an escape.

If carbon monoxide is present in sufficient quantity (say from $\frac{1}{2}$ to 1 per cent.) it will produce unconsciousness in a few minutes. But when, as usually happens, it is present in smaller but dangerous amounts, it takes time to produce its effects. The onset of symptoms is insidious and gradual. The first effect of the gas is to produce heaviness and dizziness, with a swimming feeling and often a painful throbbing in the head, and with palpitation and breathlessness. There may be a feeling of constriction in the chest. These are the usual symptoms of onset, and they are increased and accelerated by exertion. Then general muscular weakness ensues, often first shown by staggering in walking. Frequently indeed the first warning the man has is that his legs give way under him. The person affected is still conscious, although judgment is already very commonly impaired, and he is now probably aware of his danger. But he may be unable to help himself because his muscles refuse to serve him. Sometimes the muscular exertion involved in the effort to escape causes death. Soon the affected man becomes insensible, and when any one has been rendered unconscious by carbon monoxide, if he be not quickly extricated and removed to fresh air, he will die.

In gassing accidents due to carbon monoxide, those overcome are frequently already dead when

extricated, or succumb very shortly afterwards. If they survive the first hour or two, recovery, in our experience, is almost invariable. The bodies of those fatally poisoned by carbon monoxide lie lifelike with the cheeks and tongue pink.

There is one other feature of carbon monoxide poisoning which is worth noting, namely, that sudden exposure to cool fresh air may sometimes greatly intensify the symptoms, or cause, for example, a recurrence of unconsciousness after partial recovery. This fact should be kept in mind in treating these cases.

Carbon monoxide is highly poisonous. As little as 0·05% will in time produce symptoms; anything above 0·15% is dangerous, although it will require about 0·5% to produce fatal effects in a moderate time. It is clear from the history of 'gassing' accidents that the latter amount is not infrequently present in places where these accidents have occurred.

The immediate action of the gas depends on the strong affinity which it has for haemoglobin, this being more than 200 times stronger than that of oxygen. Hence when breathed into the lungs carbon monoxide tends to pass readily into the blood, displacing the oxygen from its combination with haemoglobin, and forming carboxyhaemoglobin instead. With each inspiration more of the latter is formed. But carboxyhaemoglobin, although more stable than oxyhaemoglobin, is not a permanent combination, and carbon monoxide unites with or separates from haemoglobin in amounts determined by the partial pressure of oxygen and carbon monoxide in the air breathed. Three practical possibilities may be considered:

(1) If the percentage of carbon monoxide in the air is high, in time too little oxygen is left in the blood to support life, and death results from oxygen starvation. Slight symptoms are noticeable with a 20 per cent. saturation of the blood. When the saturation reaches 50 per cent., symptoms become urgent. 'It is scarcely possible to stand and even slight exertion causes temporary loss of consciousness.'[1] In cases of gradual poisoning death will ensue when about 80 per cent. of the oxygen is displaced, but death may be precipitated by exertion before this point is reached.

(2) On the other hand, when moderate amounts of carbon monoxide are present, in time a balance is struck, and the blood ceases to take up more of the gas. Thus Dr. J. S. Haldane states that roughly a percentage of 0·04 will *finally* produce about a one-third saturation, 0·08 per cent. one-half, and 0·16 per cent. a two-thirds saturation. It would require the absorption of about 300 cc. of carbon monoxide into the blood to produce half-saturation in a man of average weight, and the time, in which the different limits of saturation which may be produced by the percentages of the gas mentioned will be reached, obviously depends, *inter alia*, on the amount of air reaching the air cells per minute, and this varies, of course, with the amount of exertion performed. The time theoretically required can be readily calculated. Thus to produce a 50 per cent. saturation in a man at rest, with 0·2 per cent. of carbon monoxide in the air, would require at least half an hour. Actual experiment showed the period to be considerably longer.,

(5) When a man poisoned by carbon monoxide is removed to fresh air the combination of that gas with haemoglobin gradually gives way in the presence of air, especially if the latter contains an excess of oxygen, such as can be supplied by the oxygen cylinder. The carbon monoxide passes off in the expired air, oxyhaemoglobin is gradually reformed, consciousness returns, and the affected man recovers.

The displacement of carbon monoxide, however, is slower than its absorption. The time taken depends on the original degree of saturation, and, other things being equal, is probably also increased by a longer duration of exposure. Experience shows that even in bad cases half an hour of an hour, less or more, will generally suffice to get rid of enough of the gas to place the patient beyond immediate danger. The results of direct and experimental spectroscopic observations regarding the length of time, during which the presence of carbonyhaemoglobin may be detected in the blood of the living after exposure, vary considerably. Usually in six or eight hours practically all the carbon monoxide is eliminated.[8] The maximum limit is probably considerably longer than this, but all evidence shows that the remote effects of carbon monoxide poisoning cannot be ascribed to prolonged retention of the gas in the blood.

In my experience of carbon monoxide poisoning, if a man is extricated alive, and promptly treated, he usually recovers. If death does not take place within an hour or two, recovery is almost invariable, and takes place within a comparatively short time. Serious after effects are, I think, rare in cases of carbon monoxide poisoning as they occur on the Rand. Severe headache, greatly aggravated by movement, and lasting for perhaps twelve or twenty-four hours, sometimes for several days, is common and characteristic. Vomiting also is common, and epigastric pain is not infrequent. These symptoms are observed when the affected man has been brought to fresh air. I have seen occasionally marked temporary excitement and mental confusion or exaltation, and even definite epileptiform convulsions during the stage of recovery. Drowsiness, giddiness and slight breathlessness on exertion, and irregularity of the pulse, may also persist for some days. Bronchial irritation and cough may be observed, but in some cases this may be due to coincident slight nitrous fumes poisoning. Idiosyncracy counts for a good deal in susceptibility to carbon monoxide. Alcoholics appear to stand 'gassing' badly, so, naturally, do those affected by miners' phthisis. Occasionally I have heard sufferers from miners' phthisis state that they have never been the same after having been 'gassed,' and they have attributed the commencement of their breakdown to this cause.

But the long train of serious after effects which have been repeatedly observed in England and elsewhere in cases of carbon monoxide poisoning after colliery explosions or fires, or after poisoning by the gas from other sources, is in my experience comparatively inconspicuous on the Rand. I should very much like, however, to hear the experience of other medical men in regard to this matter.

I shall therefore note briefly the chief sequelæ which have been observed.[2]

1. *Prolonged coma*, lasting sometimes for several days, and usually ending in death, is described. I have not seen this. Relapses after apparent temporary recovery are frequent, and may be fatal.

2. There may be persistence for considerable periods of the initial *symptoms, chiefly cardio-vascular* in character.

Headache, breathlessness on any exertion, tachycardia, palpitation and precordial distress may per-

sist for weeks or months. Syncopal attacks may recur and end in death. Rarely arterio sclerosis may result later on.

3. *Pneumonia* is a not uncommon cause of death after carbon monoxide poisoning. The type is usually lobar and extensive, sometimes broncho-pneumonic. The condition usually develops on the second or third day, sometimes later, and the disease commonly runs a rapid course, the patient often dying after two or three days' illness. I mention these facts in order to differentiate such cases from those of nitrous fumes poisoning, in which the acute respiratory symptoms always develop within twelve hours. Personally I do not recollect any case admitted to hospital from carbon monoxide poisoning, in which lobar pneumonia has developed within a few days, but I should be glad to hear the experience of others. It is probable that such cases do occur from time to time, but I do not think that they are at all common. In one instance in the records the mining inspector held that the death of a native, who had been admitted to a mine hospital suffering from 'gassing,' and who after three days developed pneumonia and died on the tenth day after the accident, was due to the accident. The point is of medico legal interest. Acute oedema of the lungs has also been described.

Apart from cardio-vascular changes the after effects of carbon monoxide poisoning fall mainly on the *central nervous system*. Symptoms pointing to lesions, both peripheral and central, bulk largely in the literature. They often develop after temporary recovery. Neuritis frequently occurs. Vasomotor disturbances are also frequent, *e.g.*, indurated erythematous patches, lasting for weeks and followed by local anaesthesia. Trophic disturbances may occur, sometimes even deep sloughing of the tissues. Less commonly more extensive motor disturbances, such as a general paresis, paraplegia, or hemiplegia, are observed, even poliomyelitis and disseminated sclerosis.

Changes in the voice and speech, and disorders of the special senses, such as nerve deafness, and auditory and visual hallucinations, are noted.

Psychical symptoms are not uncommon, excitability and irritability, amnesia, change of disposition, loss of 'grip' and initiative, depression, melancholia, even final dementia, are described. These disturbances may last for months or may become permanent.

If I am right in considering that such serious after effects are comparatively infrequent on the Rand, I can only conclude that an important factor in producing them is *duration of exposure*. With us the exposure is short and sharp. After colliery explosions exposure is often prolonged and poisoning is more gradual.

At the great explosion, followed by a serious fire, which occurred at Senghenydd in October, 1913, 440 men were killed, and 18 were rescued alive and survived, *after an exposure to after damp for twenty hours.*

The gross signs found post mortem in fatal cases of carbon monoxide poisoning are well known and quite characteristic. I only refer to them now now in order to indicate the contrast with those shown in cases of nitrous fumes poisoning, which I shall mention later. The general signs are those of asphyxia with engorgement of the right heart and the great veins. The special sign is the characteristic cherry red colour of the blood, the muscles and the tissues generally, which is unmistakeable. The lips and skin may, but do not always, show a strikingly life-like tint. There may be patches of rose pink discoloration here and there on the surface of the body, and the post-mortem hypostatic staining may have a pinkish tint. The blood is cherry red, very fluid in character, and does not readily coagulate. The bronchi may be injected, and the lungs commonly show areas of congestion or hyperæmia. The brain and chord also show hyperæmia, and frequently multiple small haemorrhages. The spectrum of carboxy-haemoglobin may be detected for lengthy periods, even for months, after death, in the blood of those who have died while exposed to the gas, and the percentage saturation of the blood may be estimated by the colorimetric test suggested by Dr. Haldane.

There is only one other point which I wish to mention regarding the symptomatology and pathology of carbon monoxide poisoning. *Chronic carbon monoxide poisoning* occupies a considerable space in the literature of the subject.[2] Very numerous accounts have been published of a similar train of symptoms having occurred in persons, whose occupation or living conditions have exposed them to the inhalation over varying periods of repeated but relatively small doses of carbon monoxide, *e.g.*, in those working in connection with gas engines, or blast furnaces, in laundries, or in places where gas is used for fuel and lighting, or where slow combustion stoves are used, etc.

The train of symptoms described includes headache, giddiness and muscular weakness, palpitation, anæmia and breathlessness, indigestion, loss of weight, neuritis, neurasthenia, even loss of memory and weakening of the mental faculties.

It is characteristic of many of these cases that cessation of work, or change to fresh air, causes a disappearance of the symptoms.

The observed cases of this condition are numerous and well-authenticated. On the other hand, there is equally valid evidence of the acquisition in many cases of *tolerance* to repeated small doses of the gas. Experimental evidence bearing on the matter appears to be small in amount. Nasmith and Grahame,[11] however, found that guinea pigs were not incommoded by a 25 per cent. saturation of their blood by carbon monoxide, and that they could be *gradually* brought to tolerate even a 45 per cent. saturation, an initial extensive degeneration of the red cells, accompanied by a short period of ill health and loss of weight, being rapidly compensated by a great increase in the number of red corpuscles.

The point is of practical interest locally, since intermittent exposure to small amounts of carbon monoxide may not infrequently take place underground, for example in persons waiting to be hauled up the shaft after blasting, or when the ventilation of workings in which blasting has taken place is inadequately carried out. The question of protecting waiting places from the access of dust and blasting fumes is now receiving attention.

(To be continued.)

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Review.

THE METALLURGY OF GOLD.

By Sir Thomas Kirke Rose.

6th Edition, published by Chas. Griffin and Co., London. Price 22 6 nett.

To the members of the C.M. and M. Society the name of Sir Thomas Kirke Rose is so familiar that little need be said introductory to a review of any work of his on gold. There is no student of gold metallurgy who is not familiar with his classic on this subject, and a new edition needs little praise other than to call attention to the many fresh features and up-to-date character of the information embodied in this work.

So evident are these attractions, that a first glance leads to the opinion that a new work has been placed before us, and it is only by going carefully through the book that the reader locates the old familiar order and processes that carry him back nearly quarter of a century. Old things have been revised and new developments treated in a comprehensive manner that quite clearly shows the master hand. The Institution of Mining and Metallurgy is to be congratulated on a President who can write in so authoritative a fashion. References are world wide and embrace particulars of treatment in all goldfields, with special mention of the latest features of Transvaal gold mining practice.

The book contains 600 pages, and is divided into twenty-five chapters, whilst an excellent contents and an index are included. Quite a number of the 222 illustrations are insets, and form most valuable additions to the work.

The contents treat of the physical and chemical properties of gold, its alloys and the chemistry of its compounds; its occurrence and distribution and the treatment of placer deposits; stamp battery crushing and amalgamation; fine grinding; concentration and dry crushing; roasting and chlorination; the cyanide process—chemical reactions, general methods and special details of practice; refining and parting of gold bullion; the assay of ores and of gold bullion; whilst of more special interest is the bibliography—eleven pages detailing, according to subject, all publications on gold metallurgy.

The book is attractively printed and the illustrations clearly defined. The work should attract the practical man as well as the student, and to no one should it be more interesting and instructive than the many men engaged in metallurgical work on the Rand.

A. McA. J.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Production of Argon.—"The cyanamide process has been used recently in America as a means of producing argon in quantity. Nitrogen obtained from the air by the copper process, is repeatedly passed through the cyanamide furnaces, until concentrated argon remains as the residual gas. Thousands of cubic feet of this gas is being sold for the use of the electric lamp industry, chiefly in America, but some is exported."—F. S. Washburn, *Chem. News*, 1915, *112*, 29.—*Journ. Soc. Chem. Ind.*, July 31, 1915, p. 796. (J. A. W.)

Precious Metals in Litharge.—"From 1200 gm. of litharge the author obtained a weighable quantity of gold. Platinum could not be detected in the litharge. The assumption that colloidal platinum occurring in ores cannot be determined by the ordinary assay methods, is disproved by the results of experiments in which colloidal gold and platinum were added to ores and the latter then analysed; from 98% to 99·6% of the two metals was recovered."—J. Loevy, *Chem.-Zeit.*, 1915, *39*, 287.—*Journ. Soc. Chem. Industry*, June 15, 1915, p. 618. (J. A. W.)

VEGETABLE DEFUS AND POISONOUS PLANTS. *Bark and fruits of Strychnos Henningsii, Gilg., from South Africa.* The bark contained 1·8%, the husk of the fruit 0·16%, and the kernels 1% of an alkaloid, different from strychnine and brucine, but which could not be obtained in a crystalline condition, except in one experiment in which a few rosettes of white needles were obtained, which decomposed between 260° C. and 270° C. The husks contained 1·9 of an orange red fat, and the kernels 17·6% of a pale, brownish orange oil.

Specimens of *Acokanthera venenata*, G. Don, from the Transvaal, and of "Cape Slangkop" (*Ornithoglossum glaucum*, Salisb.) from S. Africa were found to contain poisonous, bitter, amorphous substances with a physiological action similar to that of digitalis. The leaves of *Tephrosia Vogelii*, Hook, from Rhodesia, were found to contain 0·15%, and the seeds 0·3% of tephrosin. The leaves also contained 0·66% of tephrosal and 0·05% of the yellow substance, m. pt 228° C. to 229° C., isolated by Harriot (*loc. cit.*), whilst the seeds contained 0·068% of the yellow substance and 0·11% of a new, colourless non nitrogenous substance, m. pt 158° C." BULL. IMP. INST., 1915, *13*, 28–65. *Journ. Soc. Chem. Ind.*, May 31, 1915, pp. 570, 571. (J. A. W.)

PRECIPITANT FOR AMMONIA (Substitute for Nessler's reagent). "When mercuric chloride solution is made alkaline in presence of sodium chloride, a very soluble complex compound, Na_2HgCl_4, is formed, which is stable in presence of large amounts of the sodium salt. This solution precipitates quantitatively very minute amounts of ammonia, producing a bluish white cloudiness, or a white precipitate with larger quantities of ammonia. By adding lithium carbonate the tendency of the precipitated particles to coalesce is prevented, and if the reagent be employed for nephelometric measurements (in certain biochemical nitrogen determinations), a little 1% soluble starch solution is also added to keep the particles in suspension. The proportions required are: 70 cc. cold saturated mercuric chloride solution, 15 gm. sodium chloride, 35 cc. saturated lithium carbonate solution, and 65 cc. water. The water used must be free from ammonia; if a cloudiness appears, the liquid can be easily filtered after shaking with powdered talc. The reagent is more stable than that of Nessler, but fully as sensitive, precipitating ammonia from ammonium sulphate diluted to 1 : 1,000,000. It should be useful in water analysis, and can be applied directly to ordinary and micro Kjeldahl determinations, rendering them less tedious and obviating errors due to distillation." S. S. GRAVES, *J. Amer. Chem. Soc.*, 1915, *37*, 1171–1181. *Journ. Soc. Chem. Industry*, 15th June, 1915, p. 638. (J. A. W.)

COLORIMETRIC DETERMINATION OF PHOSPHORIC ACID. "A colorimetric method is described depending on the blue coloration produced when ammonium phosphomolybdate is reduced by warming with a solution of hydrazine sulphate; the intensity of the colour is proportional to the amount of phosphoric acid present. A comparison solution is prepared by dissolving 0·505 gm. of crystallised disodium phosphate in 100 cc. of water. One cc. of this solution is placed in a graduated centrifuge tube, 1 cc. of nitric acid (sp. gr. 1·125) and a quantity of 30% ammonium nitrate added to fill the tube to the tenth mark, then the tube is heated and 2 cc. of 3% ammonium molybdate solution is added, thoroughly mixed, and the solution centrifuged for two minutes. The clear solution is poured off, and the precipitate shaken with 30% ammonium nitrate solution and again centrifuged for two minutes, and the clear solution poured off. The precipitate is then washed into a 100 cc. flask by means of a 2% solution of hydrazine sulphate (20 cc.), and the solution warmed. The flask is then cooled, and filled to the mark with distilled water; 1 cc. of this solution is equivalent to 0·00001 gm. P_2O_5. The solution to be analysed is treated in the same manner, and the two solutions are compared in a colorimeter.—E. RIEGLER, *Bull. Acad. Sci. Roumanie*, 1914, *2*, 272–275.—*Journ. Soc. Chem. Ind.* July, 31, 1915, p. 796. (A. W.)

REAGENTS FOR USE IN GAS ANALYSIS. I. ALKALINE PYROGALLOL. "The most satisfactory alkaline pyrogallol reagent for absorbing oxygen is one containing 13·6 gm. of pyrogallol and 71·5 gm. of potassium hydroxide per 100 cc., prepared by dissolving 15 gm. of pyrogallol in 100 cc. of potassium hydroxide solution of sp. gr. 1·55; carbon monoxide is formed only in negligible traces, if at all, by this reagent. Using 100 cc. of gas sample in a Hemple double pipette, with 25 cc. of the reagent in the first pipette and 185 cc. in the second, and working at 20° C. to 24° C., the following values were obtained for the specific absorption of the reagent (see preceding abstract):—

In Hempel pipette for use with mercury.

No. of minutes samples were shaken ...	1	2	3	4	5
Samples containing 20·9% O_2	27	30	31	—	—
Samples containing 90% O_2	21	24	24	—	—

In Hempel pipette for liquid reagents.

Samples containing 20·9% O_2	25	29	31	—	—
Samples containing 90% O_2	0	27	30	31	32

The values for the samples mentioned are applicable to all samples containing less oxygen, and the values for samples containing 90% of oxygen could be applied, for practical purposes, to the analysis of nearly pure oxygen. Hempel's objection to the use of potassium hydroxide purified by alcohol could not be confirmed. R. P. ANDERSON, *J. Ind. Eng. Chem.*, 1915, *7*, 587–596. *Journ. Soc. Chem. Industry*, August 16, 1915, p. 853. (J. A. W.)

METHOD OF EXTRACTING POTASH FROM FELSPAR. "The Appalachian Mountains contain large quantities of felspathic rock high in potash. Many attempts have been made to extract this potash in water soluble form in order to use it in fertilizers. The process consists in powdering 100 parts of felspar rock with 20 parts of burned lime and sprinkling a solution of calcium chloride upon a moving layer of the powdered mixture. The calcium chloride unites with the lime to form balls or clumps, and these are screened out and fed to a rotary kiln. The formation of the clumps produces an intimate mixture between felspar and flux and thus a reaction during heating which gives a high yield; it also eliminates the dust nuisance. The clumps passing down the

kiln are heated by the powdered coal blast to a bright red and issue in the same outward form in which they entered but with the potash dissociated from the silica of the felspar and united with the chlorine of the calcium chloride to form potassium chloride.

The lumps fall hot into water and the potash salt is leached out, producing a 10 per cent. solution of potassium chloride. This solution is sprayed down through the hot waste gases of the kiln, the water evaporated and a hot concentrated solution obtained. This solution is dried at this point or by passing through a rotary drier using the hot gases from the rotary lime-burning kiln."—G. W. Coggershall.—*Engineering Magazine*, July, 1915, p. 578.—(H. A. W.)

METALLURGY.

Electric Furnace for Cyanide Gold By-Products.—"The gold precipitate from the zinc-dust presses in the cyanide plant of the Alaska Treadwell Gold Mining Co., Treadwell, Alaska, is treated, in the refinery adjoining, by the Tavener or lead-smelting method. About 3 or 4 tons of this precipitate are produced monthly having a gold assay value of $40,000 to $60,000 per ton.

It was formerly the practice to treat the by-products from this process, consisting of 2 tons of slag, 300 lb. of matte, 500 lb. of refinery refuse, flue dust, etc., in a 24-in water-jacketed blast furnace. This practice was discontinued in the summer of 1914 by reason of the difficulty of keeping the lead-well open when treating a high-grade lead product; of preventing the loss of gold in the flue dust; and of avoiding injury to the general health of the refinery operators; a plain single-phase electric furnace was substituted for the blast furnace.

The furnace was constructed from an old steel acid drum by cutting off the top and introducing a cable, made from strands of bare copper wire, through the bottom and spreading the strands out fan-shaped on the inside of the drum.

Powdered graphite, obtained by grinding up old crucibles, mixed with 10% cement, was tamped wet into the bottom of the drum, around and completely covering the copper wires. The graphite was carried up to the bottom of the furnace, or lead-well, and acted as the lower electrode. The sides were built up of ordinary firebrick forming a melting chamber 14 in. in diameter by 20 in. high.

The upper electrodes, of graphite or carbon, are 3 in. in diameter and 40 in. long, arranged with joints enabling new electrodes to be connected without shutting down or wasting stubs. A screw feed was arranged for raising and lowering the upper electrode. (Fig. 1.)

The cover for the furnace had three openings, one for feeding the charge, one for the escape of gases, and one in the centre for the introduction of the electrode. It was later found more practical to enlarge this centre opening to 6 in. in diameter, to allow of a central feeding of the charge around the electrode.

A 4 in. pipe connected to a ventilating fan carries off the escaping gases.

The furnace is operated on the lighting circuit through a 50 kw. transformer, 60 cycles, 110 volts.

Later, a water rheostat, operated by raising and lowering one disc on another submerged in a barrel of water, was constructed to lower the voltage for use when starting the furnace, or when there was a bath of metal in the bottom of the furnace that would otherwise allow too much current to pass and thereby cause short circuiting.

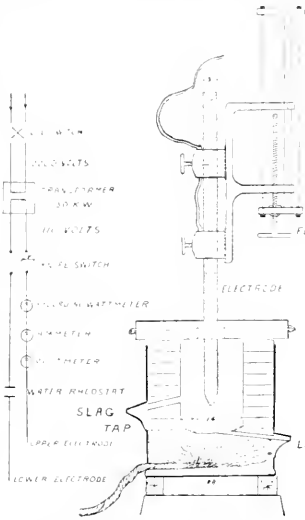

Fig. 1.—Sectional View of Electric Furnace.

The material to be treated in the furnace is mixed in the following proportions: Refinery by-products, 100 lb.; old reverberatory hearths, containing 60% CaO, 20 lb.; litharge, 20 lb.; coke, 2 lb.; scrap iron, 3 lb. About 160 lb. of the mixture is added for a charge. The total amount of material added during the melt is 8,507 lb., of which 5,870 lb. is by-product material (old slag, the residue from burning old mill launders, sweepings, etc.) of the following composition:—

	Per Cent.
SiO_2	40·1%
Fe	16·0%
CaO	16·1%
Cu	5·9%
Pb	3·0%
Zn	5·7%
Al_2O_3	2·0%
S	3·6%
Moisture	4·0%
	96·4%
Gold	$1,095·50 per ton

Coke is added to the charge as a reducing agent, only a quantity sufficient to throw down the lead being used.

The charge is introduced into the top of the furnace through the central opening around the electrode without removing the slab cover.

In starting, the furnace is operated as an arc furnace, until it has become thorough hot, when the charge, consisting of slag, brick dust, etc., is added, the electrodes being raised as the load in amperes increases, until the entire chamber is filled with the charge and the upper electrode extends into the melt almost a foot. As the charge continues to melt the electrode is moved down or up,

keeping the amperage reading as nearly constant as possible. The raising or lowering of the upper electrode decreases or increases the power consumption and thereby affects the amount of heat generated.

At this stage the furnace is no longer operating as a simple arc furnace, but has automatically transformed itself into a resistance type of furnace, the semi fluid or molten charge acting as the resistant, the current passing from one electrode through the centre of the charge to the other.

After the charge is in quiet fusion, which takes on an average 2 hr. 10 min., the power is turned off and the charge allowed to settle for 15 min. before tapping the slag. This allows the lead to settle out, the furnace acting as a forehearth.

In operation, the molten lead and slag are tapped, as in the blast furnace. If the lead freezes it is melted by diverting current from the upper electrode and letting it pass through the lead, the tapping rod playing as an arc on the side of the furnace or into the lead tap.

The operators use coloured or amber glasses to protect their eyes from the glare of the electric arc.

Operating Data.

Total running time	128 hr. 10 min
Average weigth of charge added, lb.	160·5
Average fusion time of charge ...	2 hr. 50 min
Total number of charges	53
Graphite electrodes used	1 per 24 hr.
Total power for 128 hr. 10 min., kw. hours	4,410
Power used per hour, kw.	34·50
Power used per lb. of material fed, kw.-hours	0·52
Power used per ton of material fed, kw.-hours	1,044·0
Acheson graphite used per hour, ft.	0·15
Material melted per hour, lb. ...	67·7
Material melted for 24 hr., lb. ...	1,625·0

The only item of cost for flux is the coke, 2 lb. per 100 lb. of by-product material or 118 lb. total being used, since the old reverberatory hearths furnish the lime, the cupels from bullion refining the litharge, and mill scrap the iron.

Both graphite and carbon electrodes are used, the former costing $2·95 each, and the latter $1·20. Power cost is less than 1 cent. per kw. hour.

The labour required for the operation of the furnace, including the work of charging and tapping, is one-half one man's time, when melting 1 ton in 24 hr.

The furnace was constructed and put into operation by the regular cyanide-plant crew, without the aid of special electricians. The operation is so simple as to require no special training. Although the operators were unaccustomed to handling electrical equipment, no trouble has been experienced from electrical shocks, since the bottom of the furnace acts as the lower electrode, or grounded circuit.

The advantages of the electric furnace compared to the blast furnace for melting high-grade gold slags are: A saving in mechanical loss of gold in flue dust, because the melting is done in a quiet neutral atmosphere, instead of in a rising blast of air; the obtaining of a lower grade slags, free from shot, by reason of the quieter melting action and the higher temperature obtainable, making a more fluid slag, the nicety of regulation of the melting temperature; the benefit to the general health of the operators." W. P. Lass. *Trans. Amer. Inst. Min. Engrs.*, July, 1915, p. 1445. (J. A. W.)

MISCELLANEOUS

ACCIDENTS FROM POISONOUS ASPHYXIATING GASES IN MINES. *(Continued from page 128, Dec., 1915, Journal.) Nitrous Fumes Poisoning.*—We may turn now to the consideration of the symptoms of cases of poisoning caused by the irritant acid fumes, commonly known as nitrous fumes. What is actually inhaled is mainly a mixture of nitrous and nitric acids. Nitrous fumes are the red irritant fumes with the smell of which most miners are familiar. The smell is that of fuming nitric acid. The gas is intensely poisonous, and of all mine gases it is the most treacherous. We do not know the downward limit of its toxicity. Exposure to as little as 0·05 per cent. of nitric oxide for half an hour, was found by Haldane to cause death in mice after about twenty four hours, with the typical sequence of symptoms, and the symptoms were the same whether nitric oxide or the fumes of burning dynamite were added to the air. It was found that with small percentages of nitrous fumes the haemoglobin was normal at the time of death. With higher percentages (and more rapid death) the blood was found to be more or less chocolate coloured, and to contain methaemoglobin. Under the latter circumstances nitrous fumes act like carbon monoxide by disabling the haemoglobin, and may in concentration produce rapid death. Where the exposure is to small amounts, death is due solely to the lung inflammation and its immediate consequences. It is with this latter form of poisoning that we are locally familiar. Half of the amount mentioned is the highest figure (0·026 per cent. reckoned as NO_2) found by Mann in mine air after blasting, but the presence of the gas in poisonous amounts is probably not so very uncommon in ordinary blasting practice. I shall give some evidence for this statement later. What is certain is that quite a brief exposure to small quantities of nitrous fumes is sufficient to produce serious and even fatal poisoning. Air which contains enough nitrous fumes to cause feelings of irritation in the nose or air passages is very dangerous.

The symptoms of poisoning are very characteristic. At the moment of exposure they are slight. A man who inhales nitrous fumes has a sense of irritation in the nose and throat, and of constriction, and perhaps pain, in the chest. There is headache, smarting of the eyes, and there is coughing. The latter is a characteristic sign. But these immediate irritant effects may not be severe, and commonly pass off altogether in a short time. The man may feel quite well and may continue to work. He leaves the mine, has his supper perhaps, and goes to his room. Then, in from perhaps four to eight hours afterwards (very rarely later), acute symptoms suddenly come on, and progress with alarming rapidity. In a typical severe case there is marked and increasing distress in breathing, with coughing, and often severe pain in the chest. The cough is at first dry, and auscultation may at this stage reveal no moist sounds. But this condition is speedily followed by the expectoration of a copious frothy rather fluid blood-stained spit. The lungs becomes waterlogged, and auscultation now reveals copious moist sounds. There is cyanosis and marked dyspnœa and distress, followed, unless the case is promptly treated, and very often in spite of all treatment, by collapse, unconsciousness, and death *within a few hours.* I well remember the first case of this condition which I saw, now more than ten years ago. A

miner was taken ill suddenly in his room on one of the mines at 11.30 p.m. I saw him at 1 a.m. He was cyanosed and in much distress. His lungs were full of moist sounds and one base was dull. The wooden floor of his room was covered with a blood-stained fluid expectoration. He died at 5 a.m. I could not reconcile his condition with that of any form of acute pneumonia which I have seen, but it agreed at every point with the published descriptions of nitrous fumes poisoning. It is probable, I think, that a certain number of cases, returned as 'pneumonia,' amongst underground workers, are really cases of nitrous fumes poisoning.

The typical sequence of symptoms in nitrous fumes poisoning is therefore this: (1) *Initial symptoms* of irritation occurring at the moment of exposure and usually comparatively slight. Nitrous fumes never, in our experience of ordinary gassing accidents, produce partial or complete unconsciousness at the time of exposure, as does carbon monoxide, although they may do so if present in massive amounts, and cases of rapid death from the concentrated fumes of burning explosives may be partly due to this cause. (2) *A latent period of several hours duration*, during which the patient may, and commonly does, feel quite well. (3) The *sudden onset after that interval of acute symptoms*, due to a rapidly progressive inflammatory oedema of the lungs. The appearance of this well-marked symptom-sequence is of great diagnostic significance.

The post-mortem signs are also definite. There is an intense injection of the trachea and bronchi, which is characteristic. The lungs are intensely oedematous, often enlarged and tense with oedema. A copious frothy blood-stained fluid exudes from them on section, and may be seen to occupy the air passages. There may be patches of incomplete consolidation, sub-pleural haemorrhages may be observed, and blood-stained fluid in the pleural cavities or the pericardium. The right heart and the great thoracic and abdominal veins are engorged with dark, thick, sometimes almost tarry, blood. The abdominal veins especially are often very greatly distended.

'There is one other point. Miners from time to time complain that after slight exposure to blasting fumes, perhaps from blasting at some distance from their working place, they are attacked the same night with difficulty of breathing and perhaps slight spitting of blood, which pass off in a few hours. The affected man thinks he has had an attack of "asthma" or "bronchitis." He has really had a slight attack of nitrous fumes poisoning. The most sedulous care must be taken that, if any casual blasting has to take place before the end of the shift, the water blast shall be invariably employed immediately after, and that the fumes are thoroughly dispersed. Precautions must be taken also in the general blasting at the end of the shift, in order to obviate the risk of inhalation of drifting fumes of this dangerous gas by men waiting on the stations before being "hauled" up the shaft.'

Finally, I may state that the intensely irritating character of nitrous fumes has suggested that the repeated inhalation of small quantities of this gas, may, by maintaining a catarrhal condition of the air passages and lungs, be a contributory factor in the development of miners' phthisis.

Relation of the Circumstances of the Accident to the Type of Poisoning Produced. The two types of gas poisoning described are therefore quite distinct, and the circumstances under which they occur differ also in some respects. Some years ago I made an enquiry, in order to determine the relation existing between the particular circumstances of these accidents and the particular type of gas poisoning produced.

By the courtesy of the Government Mining Engineer, I was enabled to read through the reports by the mining inspectors of all fatal gassing accidents, which had occurred on the Rand during the four years 1904-1907. There were in all 61 accidents and 169 fatalities, so that the material was considerable.

The results are shown in the accompanying table, reprinted (with slight alterations) from the 'Statement on Conditions Affecting the Health of Underground Workers on the Mines of the Witwatersrand,' originally submitted by Dr. Macaulay and myself to the Mining Regulations Commission in May, 1908.

The accidents under review fall into two groups —A. Those due to exposure to the fumes of explosives *after blasting*, the ordinary 'gassing' accidents. B. Those due to explosives being accidentally set on fire and burned.

A. Accidents caused by the Fumes of Explosives after Blasting.—I found it desirable to sub-divide this group into two classes.

(1) *The first class comprised those cases where the exposure was immediately or shortly after blasting.*

These accidents occurred, when persons returned too soon to the working places after blasting, or when they were affected by the fumes of explosives arising from blasting taking place in other parts of the workings. *The majority occurred in drives.* I found that in such cases either carbon monoxide or nitrous fumes poisoning might result, but that the latter type was considerably more common.

Whether the one type of poisoning or the other occurs under these circumstances is due in part to the character of the explosion, since if definite partial burning of the explosive takes place the risk of nitrous fumes poisoning is especially great. But the result will depend also on the duration of exposure. Unless carbon monoxide is present in large amounts, it takes time to produce its effects. On the other hand, the records abundantly show that quite a brief exposure to even small quantities of nitrous fumes is sufficient to produce serious or fatal poisoning from this source. For the same reason *drifting* fumes from blasting will typically cause nitrous fumes poisoning, and persons may be 'gassed' in this way by the fumes from blasting which has taken place at some considerable distance. Sometimes one or more of a party working in close proximity may be affected, while others escape.

Mixed poisoning must sometimes occur, but the number in which the evidence points to the probability of such a conjuction is surprisingly small. One or other type is almost invariably markedly predominant. I know of no case, in which a man, who had been rendered unconscious by carbon monoxide and had then recovered, developed later on signs of nitrous fumes poisoning. On the other hand, it is probable that carbon monoxide may co-operate in some cases of nitrous fumes poisoning in aggravating and prolonging the initial symptoms.

Table A. Fatal Accidents from the Fumes of Explosives on the Mines of the Witwatersrand, 1901-1907.

Fatal Accidents.

(A) Due to Exposure to Fumes after Blasting.

Character of Accident.	Number of Accidents.	Fatalities.		Locality.				Predominant Type of Gas Poisoning.							
								(1) Carbon Monoxide.			(2) Nitrous Fumes.			(3) Indefinite.	
		Whites.	Natives or Coolies.	Stope.	Into or Level.	Winze.	Raise.	Shaft or Station.	Instances.	Fatally Gassed.	Temporarily Gassed.	Instances.	Fatally Gassed.	Temporarily Gassed.	Fatally Injured or Gassed.	Temporarily Injured or Gassed.
(a) Exposure immediately or shortly after blasting	19	5	21	1	11	2	2	3	7	9	7	12	17	16		
(b) Exposure several hours (two or more) after blasting	28	5	34	1	3	21	3	—	27	38	47	None	—		1	
Totals	47	10	55	2	14	23	5	3	34	47	54	12	17	16	1	

(B) Due to the Ignition of Explosives.

(a) Explosives ignited, then exploded	2	7	55	—	—	—	—	2	?*	?*	?*	1	1	—	61*	15
(b) Explosives ignited, no explosion	15	7	35	3	6	1	1	1	2	16	—	14	26	9	—	—
Totals	17	14	90	3	6	1	1	6	3	16		15	27	9	61	15
Grand Total all Causes	64	24	145						37	63	54	27	44	25	62	15
		169														

(2) *In the second class of 'gassing' accidents exposure was delayed until several hours after blasting.* They occurred when the working places were entered on the following shift. They were caused by the stagnation of blasting fumes in close places owing to defective local ventilation, due, either to the miner responsible having neglected to use the means of ventilation provided, or less commonly to the fact that the means of ventilation provided were inadequate.

There were twenty-eight accidents of this nature during the four years under review. Two features stand out.

1. *Of the twenty-eight, twenty-one occurred in winzes*, owing to workers, usually natives, entering the winze before the fumes had been blown out, a circumstance which shows the necessity for more particular attention being directed to the ventilation of this type of working. The winze is much the commonest seat of 'gassing' accidents. In one of the original cases, and in two which occurred more recently, the interval between blasting and exposure was as long as 40 hours, namely, from the Saturday afternoon till the following Monday morning. This fact indicates how long the fumes may remain present in dangerous quantities in such workings, unless they are properly dispelled. Post-mortem examination made in one of these cases (2. 2. 14) showed the usual evidences of carbon monoxide poisoning.

2. *In every instance in the class carbon monoxide was the predominant cause of death*, no doubt with carbon dioxide co-operating in some cases. Nitrous fumes poisoning *never* occurs under these conditions. It is only caused when the exposure occurs immediately or shortly after blasting, or after the actual burning of explosives. It is not merely that the carbon monoxide present, by producing rapid and often fatal results, forestals and obscures the delayed effects of the more slowly acting poison. Two of the thirty-nine fatalities, which occurred, happened amongst those engaged in attempting to rescue the men overcome. But there is no history in all these cases of any of those engaged in rescue work, and temporarily affected by carbon monoxide (and there were a good many who suffered in this way), afterwards developing symptoms of nitrous fumes poisoning.

As I have said elsewhere, 'the majority are due *either* (and most frequently) to pure carelessness on the part of miners in neglecting to use the means of local ventilation at hand to blow out close places before setting their gangs to work, and in some cases in returning too soon to their working places after blasting, before the fumes have been dispersed, *or* to the lack of provision of adequate means of local ventilation by those responsible.'

Preventive measures are now adequately covered by the Mining Regulations, and, if these were properly observed, 'gassing' accidents from blasting fumes should become of rare occurrence.

Two points may be emphasised in this respect:—

(1) That the systematic use of the water blast in all close places after blasting will practically remove all danger of poisoning by nitrous fumes, from this source.

(2) That the adequate blowing out of the ends of winzes, raises and drives and other close places *for which proper means should be systematically*

provided, and the thorough wetting down of the broken rock will practically remove all danger of poisoning by carbon monoxide from blasting fumes. 'Blowing out' must be carried out both after blasting and before the working places are re-entered.

There is no doubt also that gases tend to be imprisoned in the broken rock, and may readily become a danger when they are liberated as shovelling proceeds. Hence the air should be kept turned on for some time after work has been started, and the rock should be kept wetted down.

It would be wise to provide *all* winzes with a separate ventilating pipe leading compressed air as near to the face as practicable, controllable *only* from the level above, and altogether independent of the pipe supplying air to the machines. For winzes of ordinary depth this is quite sufficient Where *deep* winzes (say from 250 to 500 feet or over) are being sunk, as is sometimes done, additional means of ventilation will probably be necessary, and, if so, should be supplied. Finally, all development work should be so organised, that, as far as possible, the practice of blasting the 'cut' and 'round' during the same shift should be done away with.

B. Accidents due to Explosives being Accidentally Set on Fire and Burned.—The number of observations which have been made on the composition of the fumes actually evolved when nitro glycerine explosives are burned in air is not large. Recorded results vary considerably, and the actual results, which occurs when accidents from this cause happen underground, also probably varies between rather wide limits.

Dr. Moir has supplied me with the following figures :—

TABLE B.

Quantity in cubic feet of gases produced by 1 lb. of blasting gelatine under different circumstances.

Conditions	CO_2	N	CO	NO
A. Perfect explosion, with out wrapper	6·2	2·9	—	—
B. 'Practical' explosion, with wrapper	5·5	2·9	0·9	?
C. Burned without wrapper in excess of air	5·7	2·3	0·7	1·2
D. Burned in vacuo	—	—	6·0	5·0

When burning takes place in the dead end of a drive or similar place, combustion ought to be expected to begin as in C, and to finish as in D, according to the available air. The calculations are for pressure and temperature at a depth of 1,500 feet.

These being the determining conditions, one might conclude that nitrous fumes poisoning would be the type of gassing most frequently met with when explosives are set on fire underground, but that carbon monoxide poisoning may also occur. Experience shows that this is actually so.

Treatment of Cases of Gas Poisoning.—First Aid Treatment.—In all cases of gas poisoning prompt and efficient First Aid treatment is of the first importance. The records, even of recent accidents, show that the practical knowledge of rescue and first aid methods in these emergencies, amongst those employed underground, still leaves a good deal to be desired. This state of matters will, it is to be feared, continue, until all mine managers set themselves to make a well-equipped underground first aid service part and parcel of the routine working organisation of the mine. Several of those attempting rescue, working with misdirected heroism, have in the past lost their lives, and many more have been temporarily overcome, through ignorance or heedlessness of the risks they have run, or sometimes from lack of the provision of adequate means of local ventilation

The following points are to be remembered in attempting rescue :—

1. When a gassing accident happens, the first thing to do is at once to turn on compressed air into the place in which it has occurred, and to keep it turned on throughout. This at once supplies those overcome with purer air, and makes it safer for rescuers to enter the danger zone. It seems an obvious precaution to take, but neglect to do so has not infrequently resulted in disaster to those attempting rescue, and in fatal delay in extricating those already overcome. No one should enter the danger zone until it has been well blown out for several minutes.

2. *No man must enter the danger zone alone.* Loss of life has occurred from this. Four men are the proper number to form a rescue party, so that if one is disabled the others may be able to help him. Men who have already been engaged in rescue work should not enter the danger zone a second time if fresh men are available.

3. Rescuers should carry with them one or two candles to test the air, and also electric lamps, or, failing these, acetylene lamps. *Never* take it for granted that the air is safe because candles continue to burn, but *always* regard it as an urgent danger signal if candles burn low or go out. If this happens, rescuers should at once withdraw, and wait until the place has been more thoroughly blown out, unless rescue can be effected immediately and easily.

4. The next thing to do is to bring the affected man as quickly as possible to the *nearest* place where the air is fresh, and then *at once to attend to the breathing.* To bring the man to the surface before commencing treatment will, in bad cases, give that treatment little chance of success, and in cases where artificial respiration is required the delay will render treatment hopeless. Yet this sort of thing still happens.

The routine first aid treatment recommended is as follows :

(a) In slight cases where the man is conscious and quite able to swallow, the usual treatment is to give a dose of emetic (copper or zinc sulphate are commonly employed), and when that has acted to follow with a dose of sal volatile.

This procedure is to be adopted both in cases of carbon monoxide poisoning and of suspected nitrous fumes poisoning, the object being that the act of vomiting will help to express mucus and gas from the lungs. There is no doubt that effectual vomiting gives great relief, and is a valuable safeguard against after-effects, particularly in nitrous fumes poisoning.

(b) In all cases of carbon monoxide poisoning of any severity oxygen should be administered.

(c) When the man is unconscious oxygen should invariably be given, and persevered with until consciousness is restored. When breathing is obviously failing, artificial respiration must be employed as well. Either Sylvester's or Schäfer's method may be used. The former is commonly more convenient on the wet and uneven surfaces of mine ground, but the latter is rather more effectual and should be used when circumstances permit.

In desperate cases, where breathing has ceased, the use of artificial respiration may sometimes be possible on the spot where the man is first found, *provided that the place has first been well blown out*. To do so requires courage and judgment, but it has saved life in at least one case within my knowledge.

(d) The man must be kept warm throughout by extra covering, friction of the limbs, etc. He must not be allowed to exert himself too soon, since the carbon monoxide poisoning exertion may readily cause a dangerous exaggeration of the condition. On no account, even in comparatively slight cases, should the affected man be allowed to walk home. Hot strong coffee should be given and alcohol should be avoided.

(e) All cases of carbon monoxide poisoning of any severity should be sent to hospital, and this rule should *invariably* be followed where the circumstances suggest that nitrous fumes poisoning may have occurred. It is most important to keep these cases under observation, since the danger of acute symptoms setting in after apparent temporary recovery is always present. The medical attendant of the patient should be notified in all cases of gassing, particularly when poisoning by fumes is suspected.

I may offer a few additional notes. First, regarding the use of oxygen. The simple oxygen cylinder with tube and mouthpiece is efficacious if properly used. A more elaborate and useful type of oxygen reviving apparatus has a mask fitted with straps, by which it is fastened closely on the face of the disabled man. The mask has a non-return outlet valve opening to the air, by which the expired air escapes. It has also a tube having an inlet valve connected with a breathing bag. With this apparatus oxygen can be delivered to the mask under a slight pressure.

Several mechanical appliances have been introduced to take the place of artificial respiration, or for use in cases, *e.g.*, of extensive fracture of the ribs, where that procedure cannot be employed. In ordinary accidents the combination of gassing with broken ribs is, however, very rare, and can usually be met by the use of a modified Howard's method of artificial respiration.

The appliances mentioned are based on the principle of alternate suction from the lungs and introduction of oxygen under pressure through an air-tight mask.

One of these devices, the "Pulmotor," has had a considerable vogue, but it should be stated that the members of a recent American Committee, appointed to investigate methods of "Resuscitation from Mine Gases," have reported adversely on the device, and on the "Brat Reviving Apparatus," which is similar in principle, on the ground that repeated suction of air from the lungs is likely to result in injury to these organs, and inadequate inflation. The U.S.A. Bureau of Mines, therefore, does not recommend the use of appliances of this type.

Mr. A. M. Anderson, Ambulance Officer, Crown Mines, states that in sharp cases of gassing he employs the movements of artificial respiration, even where the breathing is to cease, to aid the expulsion of minute and gas from the lungs. This is a useful suggestion, and I see that it has been recommended also by Dr. J. S. Davies in the treatment of bad cases of carbon monoxide poisoning, and by Drs. Elliot Black, Glenny and McNee in the treatment of cases of acute oedema of the lungs occurring amongst our troops, and caused by the use of poisonous gases by the enemy.

A word about emetics. Those used are not always to be relied on to produce prompt and efficient emesis, and if they do not do so may cause nausea and depression. In slight cases of carbon monoxide poisoning two drachm doses of bicarbonate of soda have been found a useful substitute. The eructation of gas which it causes acts in a similar way to the act of vomiting.

For the more severe cases, and in all cases of nitrous fumes poisoning, I should suggest the use of apomorphine hypodermically. This drug is more reliable, and produces a prompt and not too prolonged emesis. Ambulance officers could easily be instructed in its use. Apomorphine should be preceded or followed by an injection of strychnine or (preferably) pituitrin." L. G. Irvine, M.D.—*Med. Journal of S.A.*, Sept., 1915, p. 17. (J.A.W.)

Abstracts of Patent Applications.

(—) 37/15. George Vickery. Mechanical device or contrivance for the effectual laying of dust, smoke, or other injurious fumes in mines or underground workings. 6/2/15.

(—) 263/15. Martyn & Holford. Improvements relating to boiling tanks. 23/10/15.

This application relates to tanks for baling liquids from mine shafts and the like, especially to those for use during sinking operations.

(C.) 278/15. E. H. Wilhelm. Improvements in valves for percussive tools. 11/11/15.

This application is for the piston valve comprising two end cylinders and the central flange. This valve operates in the usual manner, the difference being in its form.

(—) 293/15. Oliver T. Jenkins. Method of hardening rock drills and apparatus therefor. 27/11/15.

Claim 1 of this application refers to the method of hardening the cutting edges of drills which consists in heating the cutting end of the drill and thereafter cooling it up to a predetermined depth in a fast flowing stream of water.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

Adamson, R., *c/o* Francistown; P.O. Box 581, Bulawayo, Rhodesia

Allen, H. E., *c/o* Krugersdorp; Figgs Peak Development Co., Ltd., Swaziland, via Barberton

Evans, R. C., *c/o* Benoni; Machavie G.M. Co., Potchefstroom

McCormick, A. R., *c/o* Elandsdrift; Machine Gun Section, School of Musketry, Cantonments, Potchefstroom.

APR 1 1920
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Vol. XVI. FEBRUARY 1916. No. 7.

Proceedings
AT
Ordinary General Meeting, February 19, 1916.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, February 19th, 1916, Mr. J. E. Thomas (President) in the chair. There were also present:—

29 Members: Prof. J. A. Wilkinson, Messrs. J. Gray, H. A. White, H. S. Meyer, C. Toombs, F. W. Watson, John Watson, E. M. Weston, Dr. W. A. Caldecott, Dr. J. Moir, Prof. G. H. Stanley, Messrs. A. McA. Johnston (Members of Council), H. A. Adam, H. C. Boydell, B. J. Hastings, W. G. Holford, C. E. Hutton, A. King, E. J. Laschinger, F. G. Macdonald, P. T. B. Morrisby, J. M. Neill, S. Nettleton, S. Newton, W. S. V. Price, A. F. Rose, A. Thomas and D. Wilkinson.

11 Associates and Students: Messrs. J. Davidson, O. A. Gerber, J. Gibson, J. Ingram, R. W. Irwin, H. L. Krause, D. W. Pugh, T. L. Thorne, H. Ward, L. A. Womble, and J. A. Woodburn.

30 Visitors and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBER.

Dr. J. Moir and Dr. W. A. Caldecott were elected scrutineers, and after their scrutiny of the ballot papers it was declared that the following candidate for membership had been elected:

PUESGLOVE, HENRY WRIGHT, Geduld Propy. Mines, Ltd., P.O. Box 41, Springs. Assayer.

GENERAL BUSINESS.

THE DEATH OF SIR HENRY ROSCOE.

Mr. John Watson, F.I.C., *(Member of Council):* Since our last meeting one of the greatest British chemists of our time has passed away, in the person of the Rt. Hon. Sir Henry Enfield Roscoe, P.C., F.R.S.

I beg to move that we, as a Chemical Society in a British Dominion, send a letter of condolence to the family (two daughters) of this distinguished chemical teacher, at the same time stating, briefly, our appreciation of his life and work.

Roscoe's name will live, I think, not only for his research work in chemistry and his published volumes, but also as a great pioneer in technical and scientific education. With great distinction he filled the chair of chemistry for some 30 years at Owen's College — the pattern and fore-runner of many other British, Colonial and American science colleges. He then sat as M.P. for South Manchester for ten years, and did much valuable work on several Royal Commissions, including one for the re-modelling of the Scottish Universities. Prof. Roscoe was one of the founders and the first President of the Society of Chemical Industry.

We older chemists, associate the name of Roscoe with chemistry, just as we think of Geikie and geology, and as we connect Huxley' name with the science of biology.

Dr. Jas. Moir *(Past-President):* I beg to second the motion.

The President: It has been moved and seconded that a vote of condolence be sent to Sir Henry Roscoe's two daughters. I would ask all members present to rise as a token of sympathy.

All present rose; the motion was unanimously adopted,

USE OF WELDED CAM SHAFTS.

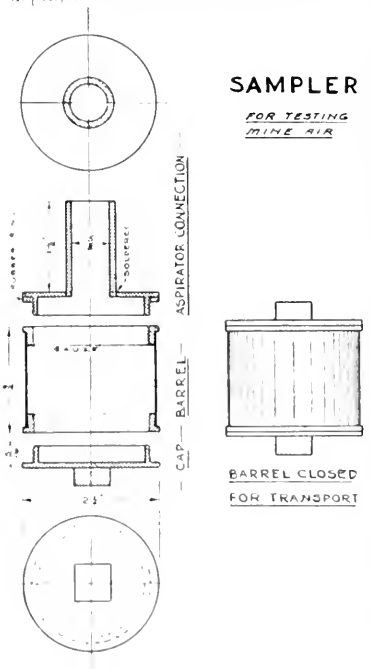
The President: I have a small item to bring up under General Business that may be of interest during the present time, when we are all trying our best to economize in material and labour required in the work-

shops in Britain. During the last three years twenty broken cam-shafts have been welded in the mine workshops, and put into use in the Simmer Deep mill. These shafts are each 7 in. diameter and 16 ft. 3 in. long, and run ten stamps. The life of these shafts varied from 491 days to 8 days actual running life, and averaged 100 days. I have no doubt that if sufficient skill and care is brought to bear a much better average might be obtained, particularly if electric welding were employed.

DUST SAMPLE TUBES.

Mr. F. W. Watson *(Member of Council):* In view of the difficulty of obtaining glass tubes for sampling dust underground, and rubber corks for Lauf tubes, we considered that a sampling tube could equally well be made of metal. I have here a specimen of a tube, which we have experimented with and it has so far given us every satisfaction.

The tube which contains the sugar is 2 in. in internal diameter and about $1\frac{3}{4}$ in. in depth, and holds about 50 gm. of sugar.



It is fitted with flat screw caps at either end, and another screw cap with a small pipe, $1\frac{1}{2}$ in. long and $\frac{1}{2}$ in. internal diameter, can be interchanged. This last cap (the adapter) connects the tube to the suction pump. One adapter is sufficient for a whole series of tubes. A diaphragm of gauze, 260 apertures to the linear inch, is fixed in the tube $\frac{1}{2}$ in. from one end, and on this the sugar rests.

When taking a sample, the flat caps are removed and the tube screwed into the adapter, which is connected to the pump. After taking the sample the tube is detached from the adapter and the flat caps replaced. When taken to the laboratory the outside of the tube is washed before the caps are removed and the tube placed bodily in a beaker of warm water, care being taken that the water does not reach the top of the metal tube. The sugar is dissolved and the dust passed through the fine gauze in the one operation, thus obviating the necessity for having a separate funnel with a gauze screen. When solution is complete, the tube is lifted out and washed, and the solution in the beaker is then ready for filtering. The tubes can be packed in a small compass and the filtering area is large enough to allow a large sample to be taken without causing a partial vacuum between the tube and the pump.

The President: We all thank Mr. Watson for this little contribution which, no doubt, will be most useful in these times, when it is so hard to obtain supplies of such material. A lot of these adapted methods may be found more successful than the original ones so long in use.

Dr. J. Moir *(Past-President):* This new tube seems to be a real improvement, but I would like to know whether there has been any oxidation of the metal at all?

Mr. F. W. Watson *(Member of Council):* We have tried it and there does not seem to be any whatever; but, if so, it is a thing which is easily met.

NON-INFLAMMABLE EXPLOSIVES BAGS.

Mr. F. W. Pursglove *(Member):* Some weeks ago the Prevention of Accidents (Safety First) Committee issued a circular letter to the mines, asking that tests should be made towards the discovery of a cheap chemical that would make ordinary fibre bags non-inflammable. On some of the mines of the Central Rand, when the reef is highly inclined, the miners are in the habit of using these bags in which to carry their explosives, this prov-

ing a more convenient method than by carrying the explosives in the packets or cases. It appears that an accident occurred on one of the mines, owing to the bag, when full of explosives, becoming ignited by a candle flame. It was to prevent similar accidents in future that the above request was made.

been awarded the Military Cross for gallantry in the field. Our list of members serving with the Forces, both in this country and oversea is rather incomplete, and we should be very glad if any members knowing of any of our members or associates serving anywhere with the Forces would send in their names to our Secretary.

Saturated Solution of Chemical. Fibre 10in. x 10in.	Contact with Candle flame. Minutes.	Result: Inflammable?	Result: Smoulders?	Length of canvas destroyed by smoulder Inches.	
10% Sol. Bleaching Powder	5	No.	No.	——	Destroys the canvas
Ferric Sulphate	5	No.	Yes.	7 inches.	
Ferric Chloride	5	No.	Yes.	4 inches.	
"Hypo"	5	Yes with difficulty.	Slightly.	2 inches.	
Zinc Sulphate	5	Yes.	Yes.	8 inches.	
Alum	2	Lights easily and rapidly spreads.		10 inches.	
Sodium Tungstate	5	No.	Slightly.	1 inch.	
Ammonium Phosphate ...	5	No.	No.		
Sodium Silicate	5	No.	No sign of charring whatsoever		

The writer makes no claim that the following tests are original; they were made to ascertain in what degree the treatment with chemical in each case affected inflammability, and to note the area of fibre destroyed by the smouldering action.

The canvas was similar to that used in the manufacture of mealie bags, and pieces 10 in. x 10 in. were immersed in saturated solutions of the chemicals mentioned, dried and then subjected to the flame of an ordinary candle. It will be seen that the most successful tests were those using sodium tungstate, ammonium phosphate and sodium silicate. The latter is much to be preferred as it can be easily and cheaply manufactured on the mine by fusing silica and sodium carbonate together

The President: I should like to thank Mr. Pursglove for his valuable contribution. I am sure anything tending in the direction of making the handling of explosives safer than it has been in the past will meet with the appreciation, not only of the members of this Society, but of everybody employed in handling explosives.

In mentioning this matter, I should also like to congratulate the Geduld Mines in having won a prize from the Safety First Committee in connection with the reduction of accidents on mines of the Witwatersrand.

ASSOCIATE AWARDED MILITARY CROSS.

The President: I have no doubt that members will be gratified to learn that one of our Associates, Lieutenant H. Ruggles-Brise, formerly of the Ferreira Deep, has

Prof. Wilkinson has just told me the incident for which Lieut. Ruggles-Brise was awarded the Military Cross. It seems that he was going in to explode a British mine when he met two Huns coming along who had broken into the place. He disposed of both in a very summary way, then exploded the mine and got out safely.

THE UNIVERSITY BILLS.

The President: In connection with the University Bills, the Council has not had sufficient opportunity of going into the matter which is at present agitating the minds of many on the Rand. The idea of a great many is that the establishment of a University on the Witwatersrand should be emphasized to the Minister of Education. There is no doubt that this centre offers an exceptional opportunity in South Africa for the study of anything in the scientific line, much more so than any other centre in South Africa. In that connection, I would ask our Secretary to read a letter which has been sent to the Minister of Education on the 17th inst:—

Sir,

I am directed by the Council of this Society to urge that, in view of the extreme and ever increasing importance of the application of scientific knowledge and methods in industries of all kinds, indeed in all fields of human activity, adequate and specific provision be made, in constituting any University or Universities by the Bills now before Parliament, for the prosecution of scientific research. For example, by insuring that the instructional

staff shall be given the necessary facilities; by enabling graduate students to undertake research; by the establishment of research Professorships and Fellowships; and by the provision of funds and appliances.

In view of the scope in South Africa for industries, agriculture, etc., it becomes increasingly essential that further steps be taken whereby the vast natural resources of the country should be thoroughly and scientifically investigated, in order that these resources should be made available for adequate development, and that the youth of the country may be thoroughly well equipped to take full advantage of every opening made available by such development.

My Council is of opinion that if more definite provision for scientific research be made than the mere mention of "promotion of research" in the preamble of the Bills, it would be of incalculable value to South Africa.

I have the honour to be, Sir,

Your obedient servant,

(Sgd.) Fred Rowland,
Secretary.

Mr. H. A. White *(Vice-President):* I noticed in the *S.A. Mining Journal* of this week an article in the course of which the editor pointed out that the public of this town looked to the technical societies for some indication of their view as to the vocation of the Universities. In view of the useful work accomplished by the South African School of Mines and Technology since its inception, and the fact that the Rand is not merely the Industrial centre of South Africa, but also one of the principal places in the world where scientific and technical knowledge has the most vigorous practical application, it is to be hoped, in the interests of the whole Union, that whatever scheme of University Education is adopted for South Africa, the exceptional advantages which Johannesburg offers in this respect will be fully utilized by carrying out the training in all branches of applied science in this most favourable environment.

It is the more necessary to insist on this in view of the increasing dependence of the prosperity of South Africa upon the application of scientific training and knowledge to the utilization of its vast natural resources.

Dr. W. A. Caldecott *(Past-President):* I have much pleasure in supporting Mr. White's remarks, which I have no doubt are in accordance with the views generally held by members of this Society.

Mr. E. J. Laschinger *(Member):* I think this Society certainly should take vigorous action with regard to University training which, of course, should embrace technology as well as the older classics. No modern University can be dissociated from the advancement of science as a part of its work and, therefore, it behoves this Society, as well as other technical or scientific societies, to take a very deep interest in this matter, and to make their voices heard at the present time, because this is really a crucial moment with regard to Higher Education in South Africa. If we can, as a Society, aid the other efforts which are being made in order to give this subject of University training in South Africa more mature consideration, instead of rushing through the proposed Bills, then, I think, we are doing a very useful work. I hope that this Society will fall into line with other resolutions that have been proposed by other bodies of various kinds, asking for a postponement of the passing of the University Bills, because, as we know, this present war has upset the usual routine of Parliament so that calm and mature deliberation can scarcely be brought to bear on the Bills; which are of such a contentious nature, and which certainly should be discussed free from political party influence. Not only that; it has been pointed out, and, I think, very properly so, that this war has altered, or is going to alter very materially, our ideas and procedure with regard to education generally, more especially Higher Education. It has certainly pointed out, if anything could do so, the preponderating predominance, I might say, of applied science in modern warfare, and not only in modern warfare but also in industrial warfare, which makes the victories of peace. I think, therefore, if we could make our voice heard, and add to other pressures which have been brought to bear, especially on our members of Parliament from the Witwatersrand, to postpone the passing of the University Bills till after the completion of the war, we will have done useful work.

The President: I understand these Bills are coming up for second reading before Parliament next Monday. The copies of the letter which our Secretary has read have been sent to all the members of Parliament representing the Witwatersrand. I personally have written to another fairly influential member to do what he can in the

matter. I quite agree with Mr. Laschinger and Mr. White that technical societies should take very strenuous action in this matter, because, once the matter is arranged, it is one that will take a lot of re-arranging if it does not suit the conditions as it should do.

SCHOOL OF MINES EVENING CLASS EXAMINATIONS.

The Secretary: The following Students at the School of Mines and Technology have been awarded the book prizes offered by the Society for competition at the examinations held in December:

Inorganic Chemistry.

Stage I.—Theoretical, G. W. Bond £1 1s.
Stage I.—Practical, G. W. Bond £1 1s.
Stage II.—Theoretical, R. Railton £1 1s.
Stage II.—Practical, H. Parker £1 1s.

Metallurgy.

Stage I.—R. Railton ... ... £1 1s.
Stage II. T. L. Thorne ... ... £1 1s.

Metallurgy of Gold.

Prize divided between—
D. Ordman ... ... 10s. 6d.
D. W. Pugh ... ... 10s. 6d.

Assaying.

Stage I.—A. R. Keay ... ... £1 1s.
No award was made in Assaying, Stage II.

The President: I might say that the Council of your Society, after considering the matter very carefully, have decided to continue the grants to the School of Mines for these prizes, as they feel that, at the present time particularly, anything in the way of encouraging Higher Education in this way should not be neglected.

INDUSTRIES DEVELOPMENT COMMITTEE.

The President: Last December a circular was sent out by this Society to all members asking them for suggestions in connection with the work of the Government Committee of Industries and Munitions. I regret to say that, so far, no contributions of any sort at all have been received. I felt this matter so much that I called a special meeting of Members of the Council, under authority given to me by a previous Council meeting, to consider the matter to see if we could not do something practical. I am glad to be able to announce that we have several Members of the Council actually engaged in investigating subjects which show a very promising outlook towards achieving something practical in relieving the Industry and the factories in Britain.

MINE AMBULANCE COMPETITIONS.

The Secretary: I have to announce that the competitions for the Ambulance Shields presented by this Society will be held under the management of the South African Red Cross Society next month, entries closing on the 4th March. We hope that every mine will do its best to enter at least one team. The dates for the Competitions are, for Surface Workers' Shield, 12th March; Underground Workers' Shield, 19th March, and for the Colliery Workers' Shield, presented by the Transvaal Coal Owners' Association, on Sunday, the 26th March.

COMFORTS FOR MINING MEN AT THE FRONT.

I have been asked to draw the attention of members to a paragraph which appeared in the *Mining Magazine* last month, asking for comforts for the Engineering Company under Capt. Ralph Stokes, who, as you all know, is one of our Corresponding Members of Council. To this I have been desired to add the name of Capt. Gerard Williams, another of our Corresponding Members of Council. Both these officers are well known to many of you, and in the Royal Engineer Companies under their command, which are fighting strenuously in France, are a number of Rand men. A donation of £2 2s. has already been sent to me to forward, and if any other member would care to add anything to this sum, I am sure it would be most gratefully appreciated on the other side. The fund will remain open for a few weeks longer.

SUBSCRIPTIONS.

There are a large number of subscriptions to the Society still outstanding, and in the course of a few days every member in arrear will receive a note of his indebtedness. The Council trusts that members having had now seven months' grace, will remit the amounts outstanding as soon as possible, as it will be impossible to continue issuing the *Journal* to those in arrear any longer.

The President: In connection with what your Secretary has just said, I should like to point out that the subscriptions overdue at present amount to over £800. This is a very serious state of affairs, and I hope the members of this Society will do something to rectify it, as we cannot possibly carry on without money any more than anyone else can.

RECENT INVESTIGATIONS ON DUST IN MINE AIR AND THE CAUSATION OF MINERS' PHTHISIS.

By James Moir, M.A., D.Sc., F.R.S.S.Afr.

(*Printed in Journal, July-August, 1915.*)

REPLY TO DISCUSSION.

Dr. J. Moir (*Past-President*): In replying to this discussion I wish first to express my appreciation of the praise and encouragement of old friends in the Society, in spite of my absence from meetings.

At the same time, I may point out that it is now thirteen years since I first published on Miner's Phthisis, which, with my later publications—apparently unknown to Mr. Pam—certainly gives me the right to discuss every branch of the subject.

I regret that mining men have had so little to say, even in the fourth year of the existence of the Committee. I think I have done good by intervening, and the discussion has been valuable and enlightening.

Mr. Weinbren asks whether a quartz particle of 3 microns diameter (which would require an hour or so to fall through 5 ft. in still air) would fall quickly when thoroughly wetted. The answer to this evidently depends on what amount of water sticks to the particle. If the particle is caught by a large drop of water, it will, of course, fall quickly; but this condition cannot be achieved in mine air full of fine dust, simply because an equal number of large drops of water cannot be crowded together close enough to catch all the dust particles. To lay dust of this sort requires a special degree of atomising of the water, neither too coarse nor too fine: if too fine the water particles themselves cannot fall (*e.g.*, the fog of defective Leyner drills) owing to the eddying of the air. The miner's breath is sufficient to cause eddying of this sort. I think the best size of the water drops would be about 20 microns.

Mr. Weinbren's suggestion of using sugar in a respirator is, of course, very sound on the theoretical side, but it is obviously too messy as well as bulky.

In regard to Mr. Nettleton's enquiry about the state of the air of stopes after blasting, I am sorry that I cannot give him any accurate information. The figure is probably 5,000 to 10,000 particles per cc. of air varying with the volume of the stope and the amount of gelignite used. The use of gelignite instead of blasting gelatine in stope blasting makes only a small difference in the amount of dust raised, say, 25% less so far as we know. The Phthisis Committee has taken the line that since water blasts would be too inconvenient in stopes, there is nothing for it but ventilation and the abolition of the double shift, and I am assured that it is quite rare for blasting dust to be seen hanging about in stopes, though I suppose Mr. Nettleton has some actual case in mind.

As regards the apparent inconsisteney of my results for dust 5 minutes and 15 minutes after blasting, I have to point out that the figures refer to three different mines, and that different loads of explosives were used: they are not three consecutive results in the same place, hence the percentage by weight of very fine dust is bound to be rather variable.

Mr. Nettleton's views on the necessity for increased ventilation to cope with stope-blasting are very sound, though he seems to be rather hard on the brains of the Rand in suggesting that "every mine official should be *compelled* to take an intelligent interest in this subject." The suggestion of compulsion would seem to indicate that it is not so much lack of intelligence or interest that is the trouble, as sheer laziness or absorption in outside affairs.

Mr. Pooler has certainly gone too far in assuming that dry drilling and blasting are still common, merely because they appear in my synopsis of results, the fact being that the conditions were specially constituted for the experiments.

The question what is the amount of air inhaled by a person doing light work is still dubious. Haldane in 1911 (Encyclopædia Britannica), states that ½ cubic foot is the amount breathed during *rest*, whereas Notter and Firth give the same amount for light work, and Haldane more recently states that 1 cub. ft. is required for a person walking at 3 miles per hour.

Mr. Pooler is of the opinion that quartz particles less than 15 microns in diameter ought to be more spherical than larger particles. I do not know why he thinks so: they are certainly more rounded in outline, but it appears to me that thin scales or flakes are more likely to be formed by blasting a conchoidal mineral than small spheres. More recent experiments have indeed raised the factor which we are using to calculate the diameter from the weight per mg: we are now using the formula $d^3 = \frac{2{\cdot}6 \times 10^9}{N}$ where N is the number of particles in 1 mg.

I agree with most of Mr. Adam's criticisms. My table of results is certainly too abbreviated, but the Miner's Phthisis Prevention Committee, thinking that it would report last year, considered that I was trespassing on its rights in publishing even a synopsis. For the same reason I do not wish to discuss laying devices except to say that nothing is any good except wetting the point of the drill, and that molasses or chemicals added to water for spraying have no marked advantage. Steam-laying cannot be used as the mines are already inefficient through heat and damp. Nearly half the mines are still dependent on natural ventilation giving about 10 cub. ft. a head per minute instead of the 30 or so which are really necessary and which are given by mines with decent-sized fans.

Prof. Wilkinson hopes that a simple technical method for directly estimating dust in mine air will be discovered. I may say that Mr. Kotzé, who has all through displayed the greatest genius in inventing new apparatus for the Committee, has been engaged on this problem, and we hope that when he is happily restored to health he will be able to perfect his invention.

By the way, Mr. Gray rather objects to having been described as my 'collaborator' by Prof. Wilkinson. The facts of the case are that in the preliminary stages I did the gravimetric work required myself, but after the method had been proved to be sound, Mr. Gray did the rest of the gravimetric work—by far the largest proportion—in the ordinary course of business as the analyst to the Miners' Phthisis Prevention Committee. He, therefore, does not wish to pose as a purely scientific collaborator, even though the tables I published contained both his work and mine.

The whole of the rest of the discussion dealt with respirators. Mr. G. H. Smith is to be praised for his enthusiastic efforts to make them a successs, but I really cannot see how they can compete with efficient watering. If we have demonstrated that 97% of the dust under dry conditions can be laid by the conscientious use of water in the proper devices, then there does not seem to be scope for a "general improvement underground to come." If the dust, both coarse and fine, is *everywhere* reduced to 2 or 3% of what it used to be, silicosis will be stamped out—though there may be residual ailments only capable of amelioration through ventilation. What we really want is that word *everywhere*, and what often stands in the way is slackness and too great a regard for working costs.

Mr. Smith may, of course, still attempt to invent a respirator to remove the remaining 3% of the dust, and good luck to him! I fancy that the only way of doing that would be to employ about ten layers of flannel at once, putting the upper half of the miner inside a framework carrying several square yards of the fabric so as to give him enough air! I am sorry that I cannot yet give the details of the Committee's tests, or its method of testing: they will appear in the Report which is expected to be printed about June.

The results which Messrs. Smith and Rusden publish were done on air of the sort which I have just described, viz., air containing only very little dust—which I consider is not worth purifying any farther. The experimental error in testing respirators on such air is very great, probably 50%, consequently I place no great reliance on the results. It is particularly unfortunate that they used a narrow-bore stopcock in the experiment (see figure, p. 107) because its friction would cause a considerable vacuum: consequently their measurement of the air used may have been seriously incorrect.

Curiously enough Mr. Weston's respirator, which contains no fabric at all, has given remarkably good results on testing: nevertheless I should not depend on it unless I knew that watering had been neglected, and there was therefore no alternative. I fully agree with the rest of Mr. Weston's remarks.

Mr. White is probably mistaken in supposing that the predominance of the particles of 2 or 3 microns diameter in silicotic lungs is *entirely* due to physiological removal of large particles. Mine dust itself shows about the same predominance of small particles, and I think the effect of the Brownian movement is quite the reverse of Mr. White's idea, viz., that it prevents the phagocytes from catching the particles *less* than 2 microns (or so) in size. The 1-micron particles in silicotic lungs appear to be fewer in proportion than they are in mine dust. There are thus two selections of the particles, viz., (1) by phagocytes, (2) by Brownian movement: the result is that most of the particles lie between 1½ and 8 microns in size. There is still much to clear up in connection with this disease. Meanwhile I express my best thanks to those who have shown their interest by discussing it.

THE CONGLOMERATES OF THE WITWATERSRAND.

By E. T. Mellor, D.Sc. (Lond.), M.Inst.M.M.

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1.—Geological Features of the Witwatersrand System bearing on the Conditions of Deposition of the Gold-Bearing Conglomerates.

Introductory. Objects of Paper.

For nearly thirty years the conglomerates of the Witwatersrand have occupied a position of continually increasing importance as the source of an output of gold which has now for some time been in the neighbourhood of £40,000,000 per annum. It is not surprising, therefore, that the conglomerates have attracted a considerable amount of attention from geologists and mining engineers, and that the literature referring to them has already grown to very considerable proportions[1].

In spite of this fact, however, considerable differences of opinion still remain as to the precise nature of the conglomerates and to a still greater degree as to the origin of the gold associated with them. The extensive developments in prospecting by boreholes and in mining in the Eastern Rand during the past few years have added very considerably to our knowledge of the Upper Witwatersrand System, particularly with regard to certain members of the Main Reef Group. The bearing of this additional information appears scarcely as yet to have received anything like the consideration it deserves, and it seems extremely likely that when this evidence becomes more widely known many accepted ideas on the subject, originally mainly based upon experience in the Central Rand, will be considerably modified. Of the various theories which from time to time have been advanced to account for the association of the gold with the conglomerates, two may now be said to hold the field, namely, the infiltration theory, and the theory which regards the conglomerates as placer deposits, whose original character has been modified by subsequent re-crystallisation of many of the constituents, including the gold contents. The arguments in favour of this latter view have been clearly and concisely stated by Gregory[2], while those in favour of the former have been more recently recapitulated at great length by Horwood[3], and supported by a formidable collection of all those references which could be regarded as in any remote way affording support to the infiltration hypothesis. A review of the question of the origin of the Rand gold as it stood in 1911 was given by Young in a presidential address to the Geological Society of South Africa[4].

In view of these recent presentations of the available evidence for and against the more important theories it is proposed to confine the present paper more particularly to certain geological aspects of the question in connection with which a considerable amount of evidence has accumulated as a result of a survey of the Witwatersrand System carried out during the past five years in connection with the mapping in detail of the whole of the Witwatersrand area.

In the case of most previous observers attention has been largely, or in some cases almost exclusively, confined to a study of the conglomerates themselves, and in many instances such study has been still further limited to the consideration of the Main Reef and its immediate associates whose vast economic importance has resulted in their receiving an undue share of attention, to the exclusion almost of many of the other members of the system, and the important evidence which a study of the latter is capable of affording.

In my own survey the process has been somewhat different, and a comprehensive examination of the Witwatersrand System as a whole preceded any close concentration on the Main Reef and its immediate associates. In this way it was possible to view this group of beds in its proper relationship to the larger succession of which it forms geologically only a comparatively insignificant portion, and it is hoped that the results so attained may add fresh interest to an old subject, and bring new light to bear on the long debated question of the origin of the most productive gold deposits known to us.

(1) A list to the year 1909 may be found in an appendix to Hatch and Corstorphine's "Geology of South Africa."

(2) "The Origin of Gold in the Rand Basket." *Trans. I.M.M.*, 1907.

(3) "The Rand Basket." *Mining and Scientific Press*, 1913.

(4) "The Problem of the Rand Basket." *Proc. Geol. Soc., S.A.*, 1911, pp. xxi.–xxix.

Owing to the great importance of the economic interests connected with the Witwatersrand Goldfield and the enormous amount of information, exact and otherwise, available from many sources, which required examination, in addition to the actual work of mapping in detail over a thousand square miles of country of which no very accurate topographical map exists, a general geological survey of the Rand carried out in some detail has necessarily occupied a considerable length of time. Had the resources of the Transvaal Survey allowed of several geologists being engaged on the work, it might have been completed much more expeditiously. There is, however, one great advantage in the whole of a work of this kind being carried out practically by a single geologist in that he must necessarily become familiar with the whole extent of the field, and he is thus able to base his conclusions on a wider personal knowledge than would otherwise be the case, and to apply more uniformly and consistently the geological terms which he uses in his descriptions. At the same time in an area like the Rand, every individual portion of which is familiar in its smallest detail to some mining engineer or geologist, the descriptions published by an official observer are not likely to lack that full share of scrutiny and criticism which should be sufficient to free them from the danger of presenting the subject too much from an individual standpoint.

Some of the results of the Survey carried out during the past five years have been published in more or less detail from time to time[5], and have already been subjected to discussion and criticism. In the present paper, therefore, it is proposed with regard to many geological questions rather to give a summary of the main conclusions than to repeat the evidence upon which these conclusions have been based. It will thus be possible to devote more space to the consideration of questions affecting the gold contents of the conglomerates.

In the many descriptions of the Witwatersrand System which have been published the conglomerates have always received a very large amount of attention, as will also be the case in the present essay. It must, however, not be forgotten that they form only a small proportion of the system, and that the remaining members are as a rule more important as sources of information concerning the nature of the formation as a whole and the conditions under which it was laid down.

To some it may appear sufficient to consider the reefs and their immediate associations and to accept the fact of the existence of their widespread gold contents without further question, but there are no doubt many others, who are interested in the wider geological aspect of the subject. Indeed, if the question be asked—"Are the Rand gold deposits more or less fortuitous occurrences, as incapable of explanation as many true lode deposits still are, or are they the result of any definite combination of geological events and processes which may perhaps be found to be duplicated in other areas occupied by the same sytem in South Africa, or even in similar formations in other parts of the world?"—the answer appears largely to lie in a consideration not only of the Reefs themselves but of the sequence of events which led up to the deposition of the conglomerates and the circumstances under which they were laid down, and it is with a view of supplying information bearing on this aspect of the subject that some of the more general features of the Witwatersrand System have been briefly dealt with in the description given below.

General features of the Witwatersrand System—Upper and Lower Divisions.—In the Rand area the Witwatersrand System embraces a thickness of sedimentary beds amounting to over 20,000 feet, the general sequence of rocks and the stratigraphical positions of the various horizons at which conglomerates occur will be best understood by reference to the section given in Fig. 1, which will also serve to indicate the system of nomenclature which has been adopted.

A glance at the section will suffice to show that the whole system consists of alternations of fine and coarse sediments most of which are of the type characteristic of sedimentation in comparatively shallow water, and none of which points to deposition at any great distance from land. On the other hand, contrary to what is probably a very generally accepted view, there are, except at the base and towards the uppermost portions, scarcely any members of the

(5) "Some Structural Features of the Witwatersrand System, etc." *Trans. Geol. Soc. S.A.*, Vol. XIV., 1911, pp 24—42.
"The Normal Section of the Lower Witwatersrand System, etc." *Trans. Geol. Soc. S.A.*, Vol. XIV., 1911, pp. 93—131.
"Structural Features of the Western Witwatersrand." *Trans. Geol. Soc. S.A.*, Vol. XVI., 1913, pp. 1—32.
"The Upper Witwatersrand System." *Trans. Geol. Soc. S.A.*, Vol. XVIII., 1915, pp. 11—57.
"The East Rand." *Trans. Geol. Soc. S.A.*, Vol. XVIII, 1915, pp. 57—71.
"On the Mode of Deposition of the Auriferous Conglomerates of the Witwatersrand" *Compte rendu du Congrès geologique international*, Toronto 1913, pp. 895—900.

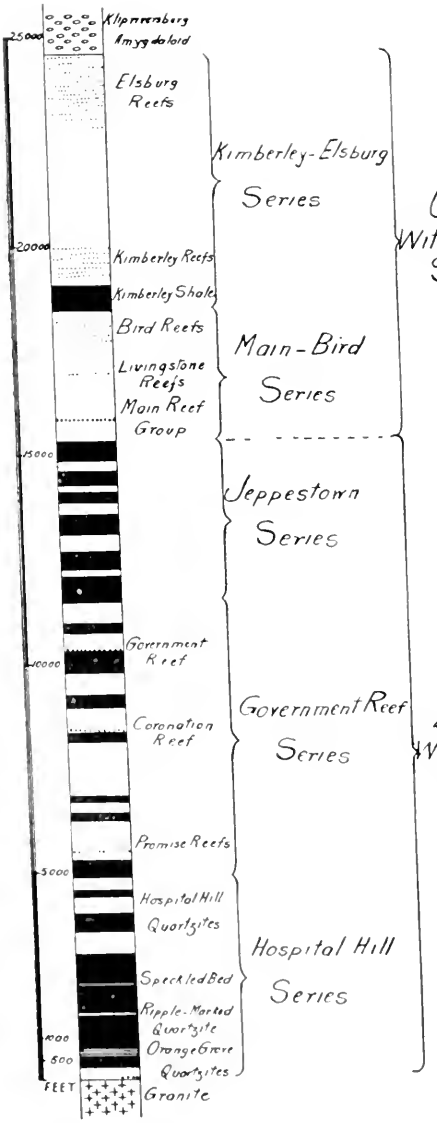


Fig. 1. *Columnar Section of the Witwatersrand System* as represented on the Central Rand, showing the succession of strata and the relative position of the various groups of conglomerates. Black = "slates" or very fine-grained quartzitic beds. Blank = quartzites and grits. ..... = conglomerates.

system, even among the conglomerates, which can with certainty be regarded as actual beach deposits, although some of them were no doubt accumulated along or in close proximity to a shore line. It will further be noticed that in the lowest third of the system finer-grained or argillaceous members predominate, in the middle third fine-grained types and coarse sandy or gritty members are about equally represented, while in the upper third the continuity of such coarser types is only broken by one band of argillaceous beds—the Kimberley Shales.

This uppermost third is usually spoken of as the Upper Witwatersrand System, the base of this division being often placed at the horizon of the Main Reef. Personally I have preferred to place the dividing line on the Central Rand, at the base of the thick series of quartzites which lies below the Main Reef, as being a less artificial limit, and one more likely to be recognisable in other areas. In any case, however, it must be remembered that the division is more or less one of convenience, that there is no corresponding geological break, and that the Lower Witwatersrand sequence finds a very natural continuation in the rocks of the Upper Division.

Conglomerates of the Upper Division.—This Upper Division consists almost entirely of a succession of sandy, gritty and pebbly beds, about 9,000 feet in thickness, interrupted only by the Kimberley Shales, some 500 ft. thick, which include a large proportion of beds approaching more nearly to fine-grained quartzites than to true shaly rocks. Throughout this great series of coarse sediments pebbles are of very general occurrence, either in well defined bands and lenticles, or scattered more or less freely through the coarse gritty beds which form the bulk of the series. Well marked pebble beds are, however, particularly abundant in certain zones which have received distinctive and well-known names. It will be convenient to deal with these from the top downwards, since we shall then proceed from very ordinary types of conglomerates to those of the Main Reef Group, which possess certain characters less generally met with, and perhaps not yet fully appreciated.

The Elsburg Reefs include a group of large-pebbled conglomerates which attain their most conspicuous development to the south of the Central Rand. Here the Elsburg conglomerates are usually very conspicuous at the surface both as actual outcrops and on account of the belt of weathered-out pebbles of large size which everywhere marks the course of the conglomerate zone. Pebbles of six inches in diameter are abundant, and individuals up to 10 inches or so are met with. In addition to pebbles of light and dark quartz which form the most numerous type, others of quartzite and of fine-grained banded rocks are common. The large-pebbled bands are most numerous over a thickness of 350 to 500 ft., above and below which pebbly bands, though frequent, are not so numerous—and the contained pebbles are of smaller size. The whole thickness of conspicuously pebbly deposits is upwards of 1,500 ft. In strong contrast to other conglomerate beds which will be considered later, individual bodies of pebbles are rarely persistent for more than a few hundred feet, and usually for much smaller distances, forming irregular lenticular deposits which vary rapidly in thickness and character. A body of pebbles 2 to 3 ft. in thickness frequently dies out or passes into bands of widely-scattered pebbles within a few feet. The beds with which the conglomerates are associated are almost exclusively very coarse grits, characterised by very irregular and impersistent bedding planes. Current bedding is frequent and characteristic. Here then we have a group of conglomerates of very ordinary type, and very usual associations, such as will be familiar to geologists in almost every part of the world. They bear evidence of having been laid down at no great distance from a shore-line and by strong but continually varying currents, such as might be expected in such a locality.

Proceeding downwards in the Witwatersrand System, the next well-marked group of conglomerates is that known as the *Kimberley Reefs*. On the Central Rand these are separated from the Elsburg group by nearly 3,000 ft. of grits and quartzites, which frequently contain pebbles as thin washes and scattered individuals, and it is practically impossible to make any divisions within this large series of deposits, or to separate the pebbly grits from the conglomerate zones into which they merge both above and below. In general character the Kimberley conglomerates are very similar to those of the Elsburg group. The chief distinction between the two on the Central Rand is the abundance of very large pebbles in the Elsburg Reefs, but in the extreme eastern and western portions of the Rand, where the Elsburg conglomerates show a considerable falling off in the average size of the pebbles

few geologists with an extended local experience would be bold enough to undertake to distinguish between the two, either in hand-specimens or in outcrops of limited extent, whose relationship to the rest of the section was unknown. The Kimberley conglomerates occur at intervals over a thickness of beds of about 500 to 600 ft. Compared with Main Reef standards, the pebbles are of considerably larger size, frequently reaching 3 or 4 inches in longest dimension. Comparatively small-pebbled bands are, however, also abundant, and the pebbles of all sizes as in the Elsburg group are frequently inclined to somewhat irregular, rather than to well rounded forms. As in the Elsburg Reefs, pebbles of quartzite and particularly of banded rocks are fairly abundant, sometimes occurring in groups. The pebbly bands as a rule are not persistent as individuals over long distances, though more so than is the case with the Elsburg group. In some localities, however, individual bands appear to be persistent over distances of a mile or more; they are then usually well-defined, especially on the foot, and it is worthy of note that such bands, from the extent to which they have been opened up, appear to have given encouraging results to the earlier prospectors. In the Far Western Rand, under the name of the Battery Reef, certain bands in this group have been found to be locally of sufficient value to warrant their being mined to a considerable extent.

In the Central Rand the Kimberley Reefs are separated from the shales of the same name by some 200 or 300 ft. of beds usually devoid of pebbles, and shewing a transition to the finer-grained beds below. Going eastwards these intermediate beds gradually disappear, thus allowing the lowest conglomerate bands of the Kimberley Reefs to rest directly upon a "slate" foot-wall, formed of the uppermost portion of the Kimberley Shales. We have thus a condition resembling that obtaining in connection with the principal reef of the Far Eastern Rand (the Main Reef Leader), and it is worthy of note as shewing the similarity in character of the conglomerates of different zones, that under such circumstances, in bore-hole sections, the lowest Kimberley Reef has frequently been accepted as a Main Reef conglomerate by engineers well acquainted with Rand conditions. In some cases the similarity just referred to has been further accentuated by the lowest Kimberley Reef yielding very encouraging assay values, which, however, appear to be of a very erratic character.

In the next lower group of conglomerates, known as the *Bird Reefs*, we have again a series of pebbly beds very similar to those of the Kimberley zone. As a rule the pebbles in this group are much smaller on the average than those of the Kimberley Reefs, but in at least one locality, lying in the West Central Rand—to the south of the Main Reef West Mine—pebbles up to 2 in. or more in diameter are abundant, and the bed containing these is exactly similar to some of the characteristic members of the Kimberley Reefs, and shows a similar persistency along the strike. The Bird Reefs include a large number of pebbly bands occurring at intervals through a considerable thickness of quartzite—often some 300 to 350 ft.—and the group is persistent throughout the whole extent of the Rand. In the west, the Bird Reefs contain a moderate amount of gold, and have been worked under the local name of the Monarch Reef. Still lower in the succession and about midway in position, between the Bird and the Main Reef groups, are the *Livingstone Reefs*, a feeble counterpart of the Bird Reefs, the smaller members of which group they closely resemble. Persistent throughout the greater portion of the Rand they occasionally carry gold in sufficient quantity to attract attention from a mining point of view, as for example, under the local name of the Chimes or Modderfontein Series on the Van Ryn property, and again in the far west in the neighbourhood of Randfontein. In bore-hole sections in the extreme south-eastern Rand they are usually very feebly represented, although the Bird Reefs are well developed and form a useful marker in that locality.

Next in descending order comes the well-known *Main Reef Group*[a], economically the most important of all the conglomerates of the Witwatersrand, and geologically distinguished by some exceedingly interesting characteristics, as compared with the groups already referred to.

In general appearance, especially in hand-specimens, the conglomerates of the Main Reef Group exhibit no essential difference from many others in the Witwatersrand System, and it is easy to obtain hand-specimens from the Bird or Kimberley Reefs, which cannot be distinguished from

(a) Usually spoken of as the "Main Reef *Series*," a term which has crept into use in such widely differing senses as to introduce considerable confusion into Rand nomenclature. Its use has therefore been avoided in the present paper.

Main Reef material. The group also includes numerous accessory bands of reef or "leaders," which in mode of occurrence, variability over short distances, impersistency along the strike, and in the erratic distribution of a certain amount of gold which they may contain are closely similar to members of the Livingstone, Bird, or Kimberley Reefs. The three principal bands of conglomerate, however,—the well-known Main Reef, Main Reef Leader and South Reef—exhibit certain characters which differentiate them from any known members of the higher groups, and which they share in common with some other remarkable conglomerates in the Lower Division of the System. Chief among such characters is the wonderful persistence of the three reefs as individuals, a quality well exemplified in the Main Reef itself, but more strongly marked in the South Reef, and most remarkably developed in the Main Reef Leader. On this account, in the description which follows, attention will be chiefly directed to the Main Reef Leader, since with the persistence of this body appears to be bound up the solution of many of the difficult questions relating to the origin and distribution of the Rand gold.

The Main Reef and Main Reef Leader.—As seen in average section, perhaps the most characteristic features of the *Main Reef* are the uniform average size of the pebbles, frequently with a preponderance of conspicuous white pebbles of coarse vein-quartz; the absence of any well-marked grading in size of pebbles, the occurrence of partings or lenticles of quartzite or more sparsely-pebbled material separating the reef locally into two or more bands and the absence of a very sharply defined and persistent foot and hanging-wall parting.

The gold values, too, have a limited range of variation, and higher values are not characteristic of any special portion of the Reef. On the other hand, the *Main Reef Leader* very commonly shows a considerable range in the size of pebbles in any particular section, accompanied very frequently, and in some parts of the Rand almost invariably, by a graded arrangement so that the largest pebbles lie on, or in the immediate neighbourhood of, the foot of the reef, while the smallest are found very generally near the top. The foot of the reef is usually very sharply defined, and is marked by a characteristic and persistent "foot-wall parting," which is in many cases accentuated by the fact that the largest pebbles lie directly upon a bed of fine-grained quartzite, "slate," or other material which is in strong contrast to the reef itself. The line of larger pebbles along the foot is often persistent even when the whole reef is reduced to a thickness of 2 or 3 in. or to a single layer of pebbles. Similarly the gold contents of the Main Reef Leader frequently show a strong tendency to concentration along certain lines, the actual foot of the reef being the most generally favoured, the larger pebbles on that horizon being not infrequently associated with "visible gold" and extraordinarily high individual assays.

As a consequence of some of the features above mentioned, it might be said that if a block of reef comprising the whole thickness of the leader be examined it would frequently be possible to say confidently which was the original upper and lower side of the specimen, whereas with a similar block of Main Reef such determination could rarely be made.

The South Reef, although extremely persistent throughout the greater portion of the Rand, shows considerable variation in its degree of development in different localities. In some respects it shows resemblances to the Main Reef, *e.g.*, in the frequent uniformity in size of the pebbles in any particular section and the presence of bands of quartzite separating the pebbly portions. On the other hand, in the frequent presence of a very distinct parting plane below the conglomerate bands, and in the persistence of the various bands over long distances it more nearly resembles the Main Reef Leader, and especially in the latter respect the Main Reef Leader as developed in the Far Eastern Rand.

General distribution of the three principal reefs.—The Main Reef and Main Reef Leader are usually very closely associated. In certain portions of the Central Rand they form practically a single large body of conglomerate, and were these the only sections known to us probably no distinction would have been made between the two reefs. As a rule, however, the two reefs are clearly distinguishable, though they may be separated merely by a clearly defined parting plane or by a few inches or feet of intervening beds. Such beds may be of quite ordinary quartzites not essentially different from those in the midst of which the reefs occur. Eastwards of the centre of the Rand, however, they very generally include or may entirely consist of a dark, originally argillaceous or very fine siliceous, rock, which on

account of its colour or of its resemblance to some of the fine-grained igneous rocks met with in the mines is usually referred to as the "Black Bar," or the "Interbedded Dyke." When present this rock immediately underlies the "Leader," the lowest pebbles in which are frequently partially embedded in the "slaty" foot. As a rule the Black Bar is only from a few inches to 2 ft. or 3 ft. thick, but it frequently swells out into lenticles which, even in the Central Rand, may attain a thickness of a hundred feet or more, and extend over areas occupying a large part of a mine. Going eastwards such developments of the Black Bar increase in frequency, size and thickness, and eventually owing to this increase and to a corresponding decrease in the development of the body of quartzites below, which normally separate them from the first big group of slates of the Lower Witwatersrand System, join with the latter to form the considerable body of slaty rocks which underlies the Main Reef Leader in the Far East Rand—the well-known "slate foot-wall" of that area. The behaviour of the Black Bar and its relationships to the beds above and below will be best gathered from an inspection of Fig. 2. In this diagram there is also represented the relationship of the three principal reefs to each other and their distribution throughout the length of the Rand. This is most clearly shown at A. From this diagram it will be seen that all three Reefs are present in the Central Rand, and that while the Main and South Reefs persist into the Far West, the Main Reef Leader dies out before reaching that area. On the other hand, the Main and South Reefs die out in an easterly direction leaving the Main Reef Leader as the only representative of the three principal members in the Far Eastern Rand. A more detailed account of the evidence upon which this representation of the relationships of the Reef is based has been given in one of the papers referred to[7] and need not be repeated here. It will, however, be profitable to follow the Leader into the far East Rand, and to note some of the extremely interesting features presented by it in that district, where it is usually spoken of as the Van Ryn or Modderfontein Reef[8].

Owing to the original discovery of the conglomerates having been made in the central portion of the Rand and to the work-

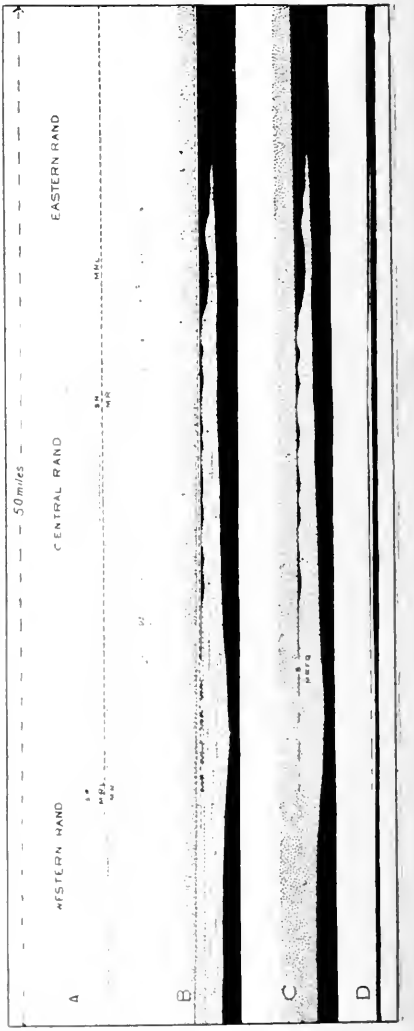


Fig. 2.—*Diagrammatic sections taken in an East and West direction showing the relative distribution of the three principal reefs of the Main Reef Zone.* A. The three principal reefs. C. The Main Reef Footwall Quartzites (MRFQ) and the Black Bar (B) with its enlargements on the Central and Far East Rand. B. Sections A and C combined. (Vertical scale ten times the horizontal). D same as C with the thickness of the section more nearly in true proportion to its length but still somewhat exaggerated.

(7) "The Upper Witwatersrand System." *Trans. Geol. S. S.A.*, Vol. XVIII., 1915, pp. 11–56.

(8) Sometimes also referred to as the "Main Reef," although really to be correlated with the Main Reef Leader of the Central Rand.

ings having gradually extended from that area, the conditions there prevailing have been those most frequently described, and have largely influenced opinion both locally and among the wider circle of geologists and mining engineers in other countries who have taken an interest in this goldfield. It so happens that from a geological point of view many features which are somewhat obscure or complicated in the nearer portions of the Rand are seen under very much simpler conditions in its far eastern portion, and it is probable that had this area been the first to be exploited many of the differences of opinion which have long prevailed on points of Rand geology would never have arisen.

Among the ways in which the Eastern Rand offers a simpler problem may be mentioned—the existence as a rule of a single principal reef instead of three or more as on the Central Rand, the occurrence of this reef at the junction of two large and easily differentiated groups of beds consisting respectively of quartzites and argillaceous rocks so that the reef lies almost invariably directly on a "slate foot-wall" of considerable thickness and can be identified with certainty, and thirdly—the disposition of the formation in a wide and comparatively shallow basin which allows of an examination of the reef and its associated beds by boreholes and mining works for many miles in the direction of the dip, instead of observations being limited to within a few thousand feet of the outcrop as is the case on the Central and Western portions of the Witwatersrand area.

Special features of the reef in the Far Eastern Rand. Existence of patches or shoots.—Owing to the extent to which the Far Eastern Rand has been explored and to the consistency of the results obtained there is very general agreement to-day as to the persistence of the same reef throughout the whole of the Far Eastern area, from its outcrop at the Van Ryn on the north to the corresponding outcrop at the Nigel Mine on the southern limits of the basin.

The study of the distribution of the conglomerates over this wide area and of the variations in its character offers some extremely interesting points. These have been dealt with at some length in a previous description[9], but certain special features may be again briefly referred to. One of these, which is closely connected with the occurrence of gold, is the distribution of the conglomerates in perfectly definite and well-marked patches.

Conditions in the Nigel Mine.—This is best seen on the extreme south-eastern fringe of the eastern basin in the neighbourhood of the Nigel Mine, and comes out with special clearness in the Nigel itself. In this and adjacent mines the conglomerate which constitutes the "Reef" does not form a continuous sheet over the underlying "slate foot-wall," as on the northern margin of the basin, but is distributed in a number of completely isolated lenticles separated by entirely barren ground. As a result of nearly 30 years working the distribution of the patches or lenticles of conglomerate in the Nigel is now known in great detail, and is represented in Fig. 3 which is taken from the mine plan. Each of the shaded patches in the figure represents an individual lenticle of conglomerate, which lies along the plane of junction of a considerable thickness of "slate" as a foot-wall, with a body of some 250 ft. of quartzite which forms the hanging country of the "Reef." The lenticles average about 3,000 ft. in length by 700 ft. in breadth, and when well developed have a thickness of about 2 or 3 ft. Towards their edges they thin out more or less gradually. Smaller patches may exist between the larger ones, but as a rule the intervening spaces are entirely devoid of conglomerates, and nothing remains but the plane of junction of the quartzites and slates known in the mine as the "contact."

The occurrence of gold is practically limited to the conglomerate patches, and in the absence of conglomerate the contact is barren. Similar conditions can be observed also along the actual outcrop of the reef-horizon in many places in the Nigel neighbourhood in a southerly or south-easterly direction, except that the patches of conglomerate show a marked diminution both in extent and in number as well as in the size of the constituent pebbles. On the other hand, in a northerly or north-north-westerly direction the available evidence goes to show that while a similar mode of distribution prevails the patches are probably of larger extent, attain a greater maximum thickness, and include pebbles of larger size. Still further to the north-north-west a similar arrangement is to be observed in the distribution of certain more robust portions of the reef distinguished by greater thickness, by larger pebbles, and other qualities well known to local mining men as indicating "reef" of

9) "The Upper Witwatersrand System," pp. 47—50.

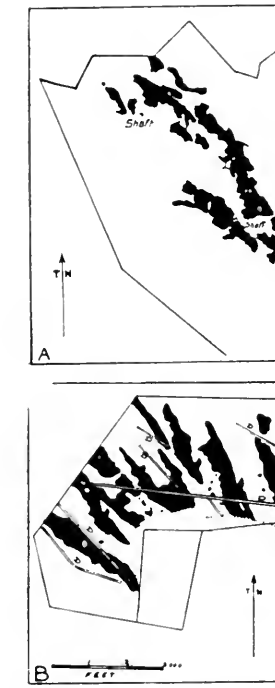


FIG. 3. Plans showing the distribution of well defined patches or "shoots" of conglomerate in the Far East Rand. A. Brakpan. B. Nigel. The dark patches represent stoped areas; in the Nigel these correspond closely with the distribution of the conglomerate, in Brakpan with special areas of conglomerate (see p. 154). The two plans are drawn on the same scale and oriented alike.
Outcrop of reef. D=dyke.
Scale approx. 1 in. = 4000 ft.

superior quality. In the last example, however, the interspaces between such patches instead of being devoid of reef are occupied by conglomerate of an inferior degree of development showing smaller pebbles and a lesser thickness. Thus within the general mass, "shoots" of specially good material are distributed in a manner similar to that shown by the individual conglomerate patches at the Nigel.

Conditions at Brakpan, Van Ryn and Kleinfontein.—These conditions are well seen at the Brakpan Mine, a portion of which is represented in Fig. 3. The shaded portion here indicates approximately the shape and size of one of the characteristic patches, which has been extensively worked. The figure is drawn to the same scale and oriented similarly to the Nigel plan, and the marked similarity in character and parallelism in direction is so obvious as to point unmistakably to the conclusion that the distribution of the better portions of the reef in this locality is due to the same causes as determined the arrangement of the more completely isolated patches of the Nigel.

Still further to the north-north-west in the neighbourhood of the well-known Kleinfontein and Van Ryn Mines very similar phenomena are to be observed in the disposition of certain areas in the mines distinguished by particularly well-developed reef associated with exceptionally good values.

The existence of these "shoots" of particularly good ore has long been definitely recognised by the officials connected with these mines, and it was probably one of these which was referred to by Gregory in connection with the Van Ryn Mine. These "shoots" of exceptional values coincide, as at Brakpan, with corresponding patches in the conglomerate distinguished by larger average size of pebbles, greater thickness, and various other definitely recognisable characters closely connected with peculiarities of original sedimentation. Although clearly recognised in the mine they do not come out very definitely in the mine plans and other records for various reasons. Thus the fact that the richer patches, instead of being separated by intermediate areas of conglomerate of low value, largely left untouched, as at Brakpan, or by areas containing no conglomerate at all as at the Nigel, form part of a wide area of conglomerate nearly all of good value and consequently worked as a whole, prevented such patches becoming evident in the stope plans as at Brakpan and the Nigel. Furthermore, the larger size of the "shoots" and the lesser extent to which they have at present been followed individually in the direction of their long axes further militates against their being so obvious and easy to recognise as in the areas further south. An examination of the better-known examples, however, leads very definitely to the conclusion that we have here "shoots" of an exactly similar nature to those of Brakpan and the Nigel, but forming specially well-developed portions within a general area of conglomerate which, as a

whole, exhibits stronger development than is usual in the mines to the south-south-east.

If now the Nigel, Brakpan and Van Ryn Mines be located on a general plan of the Eastern Rand it will be seen that they lie almost along a straight line drawn across the eastern area from N.N.W. to S.S.E.—that is, in a direction closely parallel to the long axes of the patches themselves, and there can be no doubt that along this line a mode of distribution of the conglomerate similar to that described above generally prevails. Further east, in the neighbourhood of Springs, corresponding conditions are found, and evidence is also forthcoming which renders it highly probable that a similar method of distribution will be found to a greater or lesser degree over the whole eastern area.

Similar conditions probably general throughout the Far East. Indeed, from a consideration of the data at present available with regard to the reef in the Eastern Rand, including many widely separated localities, taken in conjunction with the results of a very extended survey of the Witwatersrand formation as a whole, it seems very probable that the sheet of conglomerate which forms the principal reef in the Far Eastern Rand is very generally characterised by a mode of distribution which is represented in ideal plan in Fig. 4.

Mode of deposition of the reef in the Eastern Rand.—Further, from what we know of the thickness of this sheet in many widely separated localities and of the variations in different directions which it shares in common with many other associated members of the system, it appears probable that the original form in which this particular bed of conglomerate was laid down was a broadly tongue-shaped sheet with its base lying considerably to the north of the outcrop of the reef on the Central and Eastern Rand, and its apex to the south of the town of Heidelberg. The breadth of this sheet from east to west was approximately 40 miles, and its length when complete probably not less than half as much again (*cf.* Fig. 5).

The peculiarities just referred to, shown by the reef of the Far Eastern Rand, are very difficult to account for on any other supposition except that the deposit is one which was laid down by a broad but comparatively uniform current over a very extensive area of almost uniform level, but with a slight fall to the S.S.E., and this view finds support of the strongest character not only from the nature of the sediments whose deposition immediately preceded that of the reef on the Eastern Rand, but also from the evidence afforded by the whole of the Lower Witwatersrand System. Considerations limited to data available from the Central Rand have led practically all previous writers to associate the Rand conglomerates with littoral or even with beach deposits, but the conditions met with in the Eastern Rand can scarcely be explained in such a way. The view expressed in several previous papers that many of the members of the Witwatersrand System, including some at least of the conglomerates, originated as sediments laid down under conditions very nearly approaching those prevailing in or near extensive deltaic areas appears more in accordance with the whole of the evidence now available. Further, the conglomerate sheet which constitutes the reef of the Eastern Rand shews many evidences of having been laid down more or less continuously, and in a comparatively short space of time as a single individual deposit.

Among such evidences perhaps the most striking are its gradual and progressive diminution in average thickness when followed in definite directions away from the area of maximum development, and the corresponding diminution in the average size of the constituent pebbles along the same lines. The distribution of the conglomerates in the irregular elongated patches above described, and the persistence of these as a gradually diminishing fringe beyond the main body, as shown in the neighbourhood of the Nigel, are also very striking features which it is very difficult to explain otherwise than in the manner suggested. Again, were the conglomerate a beach deposit which had gradually advanced over a sinking area, one would expect to find very frequent false bedding and other indications of irregular and interrupted deposition and also of considerable disturbance of the as yet unconsolidated and easily eroded deposits upon which the conglomerate was laid down.

A feature in the Eastern Rand which appears to me to be particularly striking is that in certain mines like Brakpan and Springs, where some of the principal ways follow the reef in the direction of the long axes of the shoots for thousands of feet, and in which particularly clean and continuous sections are visible, anything like well-marked cross-bedding is practically absent, and the bands of

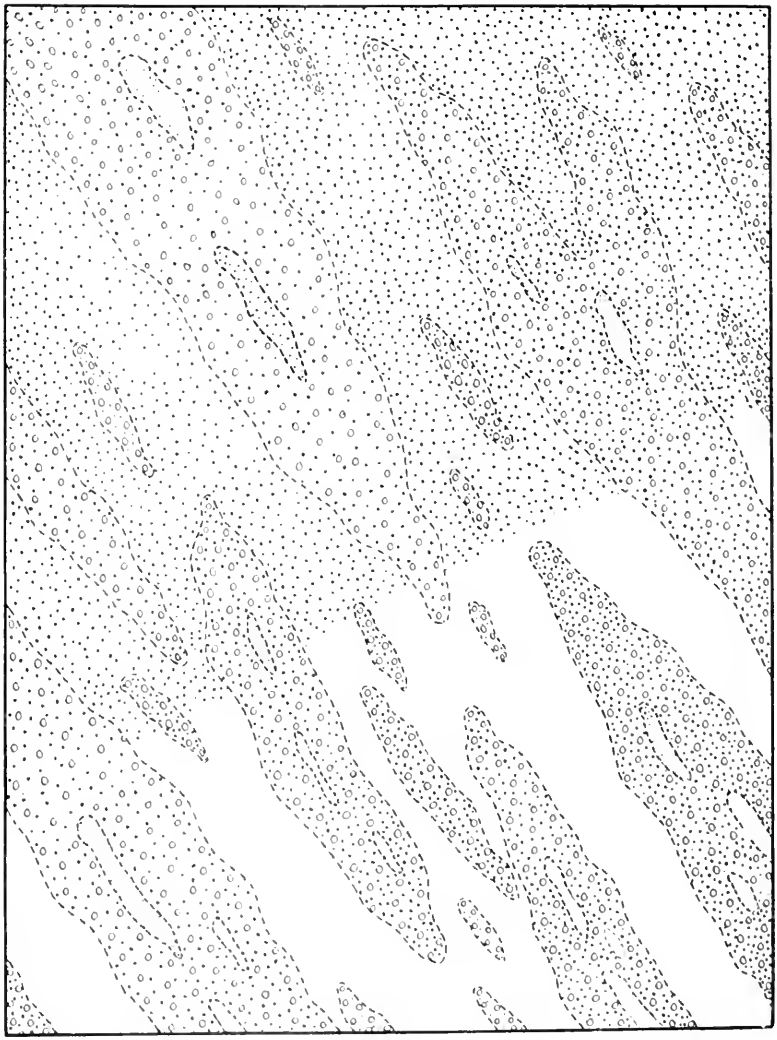


FIG. 4.—Plan showing, diagrammatically, the mode of distribution of conglomerate in the Far East Rand. Areas with dots and circles indicate more robust conglomerate with larger pebbles, usually showing better values; areas with small dots only = conglomerate with pebbles of smaller average size; blank areas devoid of conglomerate. The upper portion of the figure represents the conditions in the northern part of the Far Eastern Rand, the lower those in the southern portion (compare Fig. 3).

smaller and larger pebbles in the reef very rarely show any rapid change in arrangement but are wonderfully uniform and persistent over long distances, any variation in their relative position or character being of a very gradual nature. So also both the average and the maximum size of the pebbles in the reef as a whole is remarkably consistent over areas of hundreds of square miles. We see then that there is a good reason to regard the principal reef in the basin of the Eastern Rand as a single individual sheet of conglomerate, thicker and more continuous in its northern and north-western portion, thinner and more and more broken up into isolated patches in its south-eastern extensions. Also, that this conglomerate sheet lies almost universally on a footwall consisting of some hundreds of feet of originally argillaceous beds so that the laying down of the pebble bed abruptly succeeded a long continued period of deposition of much finer sediments. Further, it appears probable that these peculiarities were due to deposition under conditions more or less like those of an extensive deltaic area and resulted from the rapid redistribution of vast quantities of already graded material which had previously accumulated not very far from the area over which it was subsequently spread in the form in which we now find it.

Conditions on the Central Rand.—The conditions in the area just referred to are much simpler than in the Central Rand, and afford considerable assistance in explaining some of the difficulties of the latter area. If, for example, we follow the Reef of the Eastern basin back into the Central Rand we find that the relationships to the other principal reefs, and to the main groups of strata associated with them in that area, are as shown somewhat diagrammatically in Fig. 2.

No one who has had an opportunity of fully examining the mass of available evidence is likely to come to any other conclusion than that the principal reef of the Far Eastern Rand—the Van Ryn or Nigel Reef—is identical with the Main Reef Leader, which is continuous through the nearer Eastern and Central portions of the Rand, and which dies out in the neighbourhood of Roodepoort, in the West.

One of the main arguments against this correlation has been that the reef in the Far East lies always upon a "slate" foot-wall, while the Main Reef Leader of the Central Rand usually lies directly upon a quartzite. Neither of these statements is universally true, especially the latter, since as has been already pointed out the Main Reef Leader in the Central Rand usually lies directly upon a band of "slate" (the "Black Bar") which, though as a rule only a few feet in thickness, frequently swells out into broad lenticles which even in the Central and nearer Eastern portions of the Rand may occupy a very large proportion of the area covered by a single mine and attain a thickness of from 50 to 200 ft. On account of the variable nature of the Main Reef Leader in the nearer Eastern and Central portions of the Rand, and the fact that it is there usually associated with several other conspicuous conglomerates, including the Main and South Reefs and other less persistent pebble-beds, its continuity is not so obvious as further east, where the almost universal presence of a "slate" foot-wall of considerable thickness forms so prominent a feature. The more the area over which the reef has actually been mined is extended, however, the more certain does it become that the Main Reef Leader maintains its individuality over a very large portion of the Rand, and is to be regarded as a single deposit, for as we have seen, the nature of the immediately antecedent beds as well as the whole previous history of the formation, appear to justify the assumption that at the time of deposition of the Main Reef Leader the conditions in the Rand area were similar to those obtaining in some of the largest deltaic areas of the present day. We know that such areas, in addition to more ordinary floods, are subject at comparatively long intervals to inundations of exceptional violence during which vast quantities of material are transported into new positions, re-arranged and deposited in a very short space of time over new areas, and further, it is to be remembered that it is these exceptional events which leave the most permanent geological records. In connection with the explanation of the conditions under which such deposits as the Main Reef Leader have been laid down, Fig. 5 will probably be found interesting.

If we assume the sheet of gravel known to us as the Main Reef Leader to have been laid down during some such geological episode many of its features find an easy explanation. Thus the widespread extent of what has every appearance of being a single deposit, its gradual diminution in thickness and in the size of its constituent elements from the central portions towards the edges, the shape of the whole deposit as far as it

is known to us, the orientation of special areas and other features within the deposit,

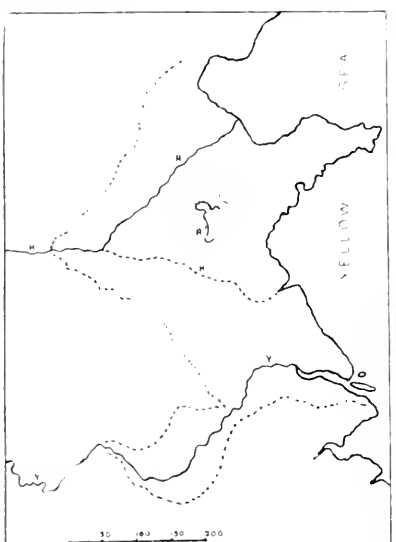


FIG. 5.—Plan showing the deltas of the Hoang Ho (H) and Yang tse Kiang (Y). The extent of the deltaic area is shown by the faint broken lines, the combined frontage being 600 miles. For purposes of comparison there has been drawn on the same scale at R a curved line corresponding in length to the outcrop of the Main Reef Zone on the Rand, while the dotted area shows the probable original extent of the Main Reef Leader. In connection with devastating floods the Hoang Ho has frequently changed its channel from H to H' and *vice versa*, on one occasion after an interval of nearly 1700 years. The area which would be covered by such a bed as the Main Reef Leader is seen to be but small compared with the total area affected by such events.

and its breaking up around the margin into a fringe of isolated patches, which become smaller and smaller and are composed of pebbles of continually diminishing average size the further they are from the main body, are all features thoroughly consistent with this mode of origin.

It is further interesting to note that most of these features may frequently be seen reproduced in miniature where a stream of storm-water of considerable and sustained volume and with a fairly large catchment, carrying a load of sediment down a more or less definite channel, has discharged it in a broadly expanding stream over an almost level area with but a slight fall.* In such an example the more distant portions of the deposit which are laid down during the brief period when the flood covers its maximum area, after being rapidly deposited are soon left beyond the influence of the diminishing stream and remain as originally laid down. On the other hand the stream now ceases to flow as one broad current with minor differences of velocity and gathers itself into a number of more definite channels along which the erosion of some of the material previously laid down is very active, so that in certain areas the finer constituents are carried away and only the coarser elements originally mingled with them remain. The deposits, which can be observed to be formed under such circumstances, appear to find a parallel in certain portions of the Main Reef Leader, as developed on the Central Rand, where a very thin reef carries very high values, and to explain why these are not so frequently met with in the Far East. Such are those portions of the reef in which the thickness is much below the average, but where pebbles of large size are the only ones remaining or form a larger portion of the deposit than usual and are accompanied by a considerable quantity of gold. They represent areas which contain the heavier elements of considerably larger bodies of gravelly material than is now present upon them. On the other hand, in the more distant portions of the field, when the reef is thin or entirely absent as, for example, around the reef-patches of the Nigel, it is very clear that these deficiencies are due to paucity of original deposition, and it is worthy of note that where the reef horizon is devoid of conglomerate it carries little or no gold.

The remarkable persistence of individual bands in the reef and the even character of the bedding over wide areas in the Far East, as compared with the frequent varia-

* The conditions required are not quite those frequently figured to illustrate the formation of miniature deltas, where a cone of detritus is suddenly deposited at the foot of a steep slope. The fall must be much more gradual and the stream more sustained. In a particular example which reproduced in a very striking manner most of the features of the reef in the Far East and Rand, a stream of storm-water with a catchment of several thousands of square yards, after traversing a well defined channel, spread out gradually, and without any very sudden change of gradient, over a broad area covered by a previously accumulated sheet of fine silt sand and pebbles covering about 100 sq. yards. After a storm this material was left in a form very nearly reproducing the arrangement found on a big scale on the reef of the Far East Rand. This was soon destroyed and the various grades of material effectively mixed by the passage of traffic over the spot, but at each repetition of the flooding of the area the arrangement of the material in the manner described above was faithfully reproduced.

tions met with on the Central Rand, are probably to be explained on somewhat similar lines, the deposit in the former area having been laid down beyond the reach of those influences which, in the Central Rand, brought about some modification of the original sediments shortly after they had been laid down.

In the foregoing description a few of the more striking peculiarities of the sheet of conglomerate known as the Main Reef Leader have been dwelt upon, because this bed is certainly one of the most interesting members, not only of the Main Reef group, but of the whole Witwatersrand system. As will be presently seen, it exhibits a number of points which are of great interest as showing the close connection which exists between purely sedimentary characters and the distribution of the gold.

Persistence of the Main Reef Leader.—The persistence of a single bed of conglomerate over so large an area as that here attributed to the Main Reef Leader will probably appear as surprising to most geologists, as it has always done to the writer. The views stated above have, however, only been arrived at after very close scrutiny of the evidence in their favour, and this evidence is now so abundant, connected and consistent that any one having time and facilities for entering into it in detail could hardly come to any other conclusion. If it should seem that the Main Reef Leader is the most persistent bed of the kind known in the world, it must also be borne in mind that no similar bed has ever offered the same inducements for its complete investigation. Moreover, the Witwatersrand System furnishes at least one other conglomerate bed, known as the Government Reef, closely similar to the Main Reef Leader, which can be followed at the actual outcrop much more readily than that reef, and which presents clear evidence of similar persistence over an equally extensive area, and indeed the Lower Witwatersrand System affords a large number of closely analogous occurrences, whose character is entirely consistent with the nature of the whole formation of which they form a part. This point will be dealt with a little more fully later; meanwhile it will be convenient to complete our consideration of the Main Reef group.

Further characteristics of the Main Reef.—To return to the Main Reef itself we have already seen that the general type of section it presents is different in many respects from that characteristic of the Leader. Among such differences are the absence of a well defined foot, the want of grading in the pebbles, the frequent presence of lenticles of quartzite within the reef, and the more rapid and frequent variations shown in the general arrangement and character of the pebble-bands. To these may be added, among others, a tendency to show frequent undulations which are the result of irregular deposition further modified by contemporaneous erosion. Undulations in the reef are particularly apparent in some mines where extensive separate stopes on the Main Reef and Leader lie close together and can be viewed almost simultaneously. Under such circumstances one frequently sees the Leader stope forming a perfectly even plane over large areas, while the Main Reef stope below is characterized by numerous undulations as a result of which the distance between the two stopes is continually changing. These conditions are sometimes so marked as to have given rise to the statement that the two reefs are unconformable, or that the Main Reef has been folded before the deposition of the Leader. The evidence, however, is perfectly clear that the undulations of the Main Reef are due to circumstances connected with its original deposition, and that the depressions were in many cases levelled up by deposits of the Black-Bar type before the Main Reef Leader was laid down. This is particularly well seen in the Primrose and adjacent mines. Contemporaneous erosion of the Main Reef is also clearly observable in the same neighbourhood and thence eastwards. This erosion appears to have been the work of broad streams which broke across the gravel-covered area represented now by the Main Reef and cut definite channels through it, which in some cases penetrated to the sandy beds below. These streams appear to have carried a load of abundant mud and silt, by which ultimately their channels were in some cases completely filled. In other cases the gravels were not entirely swept away, but were caught up by the current and intimately mixed with the invading silt, giving rise to a type of rock well known in the East Central and Eastern Rand under the name of "*Bastard Reef*," which consists of pebbles identical with those of the Main Reef scattered more or less widely through an abundant silty matrix. This matrix, which consists of a mixture of very fine muddy and sandy material, is really similar to that of which

the Black Bar is composed, and not only are all intermediate grades between typical Main Reef, Black-Bar, and Bastard to be found, but also those three bodies themselves can be seen to pass into and replace one another. The degree to which the material derived from the erosion of the gravel-sheet which is now the Main Reef, was scattered through the invading silt varies considerably, and the individual included pebbles may be separated by feet or by inches of matrix or remain sparsely aggregated in washes and clusters. Again, in rare instances masses of the original gravel have been completely enveloped in the silt without separation of the pebbles, so that we have compact bodies of Main Reef material enclosed in the Black-Bar or in Bastard. Of such a nature was probably the example (now no longer to be seen) quoted by Gregory from the May Consolidated, where a very definite channel certainly traverses the Main Reef.* Some very interesting examples of similar invading masses of sediment at the Main Reef horizon may be seen further east in the East Rand Proprietary Mines and at Brakpan.[19]

Persistence and extent of the Main Reef Leader. Compared with the Main Reef Leader, the much less characteristic sections presented by the Main Reef, its original irregularities of deposition and the subsequent contemporaneous erosion to which it was subjected are all circumstances rendering this reef more difficult to positively identify in isolated exposures than is the Leader, and this difficulty is further increased by the fact that its lower values have led to its being left practically untouched over large areas, and also that the presence of other similar bands of conglomerate below the actual horizon of the Main Reef itself is a common occurrence. Such bands are usually described as "Main Reef footwall leaders." Moreover, we know that in certain extensive areas, even in the Central Rand, the Main Reef is very poorly represented, as, for example, immediately below some of the thick lenticular developments of the Black Bar.

In the Far East Rand also the Main Reef is practically unrepresented, appearing to die out before reaching that area. In this connection it must be remembered that in the mines it is usual to regard as Main Reef any fairly well-developed conglomerate occurring not far below the Leader, and it is likely that many isolated exposures described as Main Reef do not belong to that body. These considerations, taken in conjunction with the whole characteristics of the deposit, do not allow of such positive statements with regard to the actual continuity of the Main Reef throughout the greater part of the Rand, as seem to be justified in the case of the Main Reef Leader. There can, however, be little doubt that the body of conglomerate known as the Main Reef in the Central, Western, and nearer Eastern portions of the Rand, occupies practically the same horizon over the whole of those areas, and that the body becomes intermittent and dies out gradually in the east. In seeking an analogue for the Main Reef, I have been much struck with its resemblance to some of the widespread gravel deposits which occur along the shores of Alaska, and which include the gold-bearing gravels in the neighbourhood of Nome. These deposits form extensive low-lying coastal plains or tundras occurring in many places around the Alaskan coast. Individual stretches of such deposits may extend, as at Nome, for 30 miles or so along the coast line, with a width of five miles or more. The sections and descriptions given by Brooks, Moffit, and others, show that in the character and mode of distribution of the various materials of which these plains are composed they bear considerable resemblance to the Main Reef and the beds by which it is underlain, and if we allow for the fact that the Nome coastal gravels represent an accumulation of no great depth upon a floor of older rocks, whereas the Main Reef and the beds below it were laid down as part of a much larger and more widespread succession of conformable deposits already of great thickness, the difference between the two seems rather of degree than of kind.

Conditions prevailing immediately before the deposition of the Main Reef Leader.— The coastal plains of Nome are traversed by the channels of streams eroded in the gravels, and occupied by deposits comparable to those found in the channels traversing the Main Reef. It now, owing to slight temporary submergence or other cause, the outer fringe of one of these plains could receive a deposit of finer grained muddy

* The occurrence referred to by Gregory has been described to me in detail by Mr. Cameron who saw it frequently during its removal by mining, and from his description it was closely similar in character to the examples mentioned above, but on a somewhat larger scale.

[19] "The Upper Witwatersrand System." *Trans. Geol. Soc. S.A.*, Vol. XV., 1913, p. 27.

material, a few inches to two or three feet thick for the most part, but thicker in the hollows of the surface and towards the seaward edge, merging there into pre-existing off-shore muddy deposits, we should have on a smaller scale conditions which would be extremely like those which probably existed in the Rand area immediately prior to the deposition of the Main Reef Leader; and if now an unusual discharge from one of the neighbouring rivers were to rapidly distribute over such an area a load of pebbly material gathered from its lower courses and from portions of the gravel plain traversed before deposition began we should have reproduced on a smaller scale, a very close approximation to the deposits which include the Leader and the beds below it. For the sheet of gravel so deposited would rest in places directly upon the previously existing gravels of the coastal plain traversed by stream-channels, in other places upon a thin sheet of silt swelling out at intervals to larger dimensions, and on its furthest seaward extension upon thicker and more continuous argillaceous beds. In the resulting deposits, counterparts of the Main Reef and the Main Reef Leader, the Bastard and the Black Bar would all find a place, and the variable character of the footwall of the Leader in different localities would also be reproduced. If now during a period of continued slight subsidence, there were deposited further sediments mainly of sandy type, of considerable thickness nearest the point of discharge, and thinning gradually towards its remoter portions, and these were again covered by a widespread pebbly deposit originating in a similar way to the previous one, we should then have a counterpart of the South Reef of the Rand and of the underlying beds which separate it from the Main Reef Leader. For the South Reef shows many features in common with the Main Reef Leader, and particularly with the Main Reef Leader on the far Eastern Rand, as, for example, in the character of the sections shown by it and in maintaining a uniform horizon, marked by an exceedingly even plane of deposition. From the relative position of their central and marginal portions, however, it appears probable that the point from which the material of the South Reef was discharged was a few miles further west than in the case of the Main Reef Leader, or that the direction of the distributing current was somewhat different, either of which suppositions would be easily accounted for under the conditions of deposition which have been suggested.

Conglomerates of the Lower Witwatersrand System.—Although the most important groups of conglomerates from a mining point of view have now been considered at some length, the story would scarcely be complete without some reference to the conglomerates met with at still lower horizons in the Witwatersrand System, especially as in several cases these are more nearly akin to such beds as the Main Reef Leader than are the more conspicuous but more variable pebble-beds of the Upper Division of the System. Moreover, the more varied deposits with which they are associated throw much light on the conditions leading up to the deposition of the Main Reef group.

In reviewing the main features of the Upper Witwatersrand System, we saw that, regarded in a general way, it presented from above downwards, a gradually decreasing coarseness in the character of the deposits, a feature well seen in the successive groups of conglomerates. The gradual tendency towards finer and finer grained material as we pass downwards in the system is continued in a similar manner below the horizon of the Main Reef. The general tendency is shown by the increasing frequency with which the quartzites are interrupted by groups of beds of argillaceous origin, until in the lowest parts of the system these became the dominant type.

The Northern Reefs.—A thick series of beds (Jeppestown Series) whose deposition preceded that of the strata more immediately associated with the Main Reef Group, consists, in the Central Rand, of over 3,500 ft., mainly of fine-grained argillaceous types which, however, usually contain sufficient admixture of non-quartzose material to render them very susceptible to weathering, so that they very rarely appear as outcrops. In this series the alternating finer and coarser types usually grade imperceptibly into one another, and the whole group represents a long succession of muds and silts with occasional variation to coarser types. No distinct conglomerates are known in this group, but in the next series below—the *Government Reef Series*,—pebbly bands and distinct conglomerate beds are fairly well represented. The whole *Government Reef Series*, some 6,000 ft. in thickness, includes

several large groups of quartzitic beds frequently coarse and gritty, and otherwise similar in general type to those occurring both above and below the Main Reef. They form almost continuous outcrops along the greater part of the Rand. Right on the foot of one of these quartzites occurs the band of conglomerate known as the *Government Reef*. It lies directly upon the uppermost bed of a series of shales nearly 500 ft. in thickness, and is itself succeeded by about the same thickness of quartzites. It usually consists of a single bed of conglomerate, composed of pebbles similar in general type to those of the Main Reef. In the Western portions of the Rand, where it attains its largest development, the Government Reef rarely exceeds 3 ft. in thickness. Its western termination is unknown, but going eastward it gradually diminishes, until in the Eastern Rand it is usually but two or three inches thick, or may be reduced to a few scattered pebbles along the foot of the quartzites. Yet this conglomerate is practically continuous over a distance along the strike of at least forty miles, and most probably has a corresponding extension on the dip. Lying as it does along the junction plane of some 500 ft. of quartzites, with a similar thickness of slaty rocks, there can be no doubt whatever as to the identity of the bed and of its persistence at the same horizon. Such persistence would not be so surprising in a basal conglomerate which had been gradually laid down along a junction with an older system of rocks. Here it forms but one member of a very big succession of entirely conformable beds. In its "slate footwall" and in many other respects it closely resembles the Main Reef Leader, especially the form in which that reef is developed in the Far East Rand, and like it, points to the sudden accession of gravelly material to a wide area in which for a long antecedent period only the finest sediments had been deposited.

Another conglomerate having very similar relationships occurs at the foot of the group of quartzites next but one below the Government Reef, and is known locally as the "Coronation Reef."* In this case, however, the reef may be separated in places by a few inches of sandy material from the "slates" below. Like the Government Reef, however, it occurs in connection with a sudden change to a coarser type of sedimentation which resulted in the deposition of over 500 ft. of sandy and gritty beds. Further, conglomerates are met with at horizons still lower in the Government Reef Series. One such horizon is characterized by the presence of one or more conglomerate bands, called by various names, of which the best known is the "Promise Reef." All the conglomerates of the Government Reef Series are auriferous to a greater or less degree. The Government Reef itself is stated to assay in places 4 dwt. or so over its width, and the reefs at the Promise horizon frequently give exceedingly good pannings from samples taken over a very small thickness. It was chiefly the results obtained in such a way from the Promise Reef that led at one time to the expenditure of nearly half a million sterling on the exploration of this and other conglomerates of the Lower Witwatersrand System, known collectively as the "Northern Reefs." These enterprises received additional support on account of the erroneous opinion which prevailed that the reef worked at the successful Rietfontein Mine was a member of the same group.

Other members of the Lower Witwatersrand System.—Practically no well marked or continuous conglomerates are known below the horizon of the Promise Reef. Slaty rocks, mainly of highly ferruginous types form the predominant members of the *Hospital Hill Series*. Three well-marked quartzite groups which occur are of much finer texture than is the rule at higher horizons, and the fineness of the grain becomes more and more marked as we pass downwards in the succession. Between the *Hospital Hill Quartzites* and those known as the *Orange Grove Quartzites*, which form the lowest member of the whole Witwatersrand System, we have a continuous succession of "slates" or "shales" some 2,500 ft. in thickness interrupted only at about equally distant intervals by two thin quartzite bands which are worthy of notice here as further examples of the wonderful persistency shown by individual beds in the Witwatersrand System. These two beds of coarser sediment are known respectively as the "*Ripple-marked Quartzite*" and the "*Speckled Bed*," the latter perhaps being the more remarkable.

This bed is a felspathic quartzite or arkose, the weathering out of the felspar giving the outcrop a peculiar speckled

* Not to be correlated with a reef of the same name in the Heidelberg District, which attracted much attention some years ago.

appearance, on account of which the rock received its name. It consists practically of a single massive bed, any indication of bedding-planes within the rock itself being rare. Its thickness is very constant, never exceeding 6 ft. and rarely falling below 4 ft. In spite of its small thickness, the bed is persistent along the outcrop for a distance of 25 miles, and from the fact that it shows no signs of any marked diminution in thickness at the points beyond which it becomes impossible to trace it, the bed probably extends much further than the known outcrop. This solitary bed of coarser sediment occurs in the midst of the long succession of very fine grained argillaceous and ferruginous slaty beds already mentioned. In the Central Rand upwards of 700 ft. of such fine grained beds separate the Speckled Bed from the next succeeding occurrence of quartzite, while below it are 1,500 ft. of slaty beds, interrupted only about the middle by the "Ripple-marked Quartzite," which does not exceed 16 ft. in thickness and yet shows a similar persistence to the "Speckled Bed." The occurrence of these two solitary beds of quartzites each separated by 700 ft. of fine grained or argillaceous rocks from the nearest similar beds, and though but a few feet thick, persistent over twenty miles or more, points to the existence of somewhat peculiar conditions of sedimentation, for the occurrence of such a stratum as the "Speckled Bed" in the midst of so large a group of argillaceous deposits, can hardly be explained by any oscillation of level, nor does this coarser bed show any signs of having been gradually pushed forward in the course of deposition over the underlying muddy sediments. On the other hand, its peculiar and homogeneous lithological character, its constant thickness and its want of any indications of bedding, particularly anything like current bedding, point to its having been laid down rapidly and continuously as a single bed over an area probably embracing some hundreds of square miles. It will be seen then that the persistence attributed to the Main Reef Leader is not without parallels within the Witwatersrand formation, and that the whole character of the lower part of the System is consistent with the wonderful persistence of many of its individual members. Both the "Speckled Bed" and the "Ripple-marked Quartzite" appear to represent the results of sudden incursions of coarser sediment into areas normally far beyond their reach, resulting from some unusual accession of strength or change of direction in the currents to which the deposition of the whole series was due.

Persistence of individual beds in the Witwatersrand System.—Attention is drawn to these examples of coarser deposits suddenly following the deposition of a considerable thickness of much finer material, and showing a remarkable degree of persistence, because they appear to be closely analagous in character to such beds as the Government Reef and the Main Reef Leader, and to owe their origin to similar causes which prevailed at intervals throughout the deposition of the greater part of the Witwatersrand System, and were operative from the earliest stages. For the Orange Grove Quartzites, which form the lowest member of the whole System, also show a remarkable persistence over great distances, and are found in very similar form at the base of the Lower Witwatersrand rocks in widely separated districts in the Transvaal. The progressive diminution in coarseness of texture shown by the quartzites of the Witwatersrand System towards its base is well maintained in the Orange Grove Quartzites, which are amongst the finest grained in the System. *A true basal conglomerate* resting upon the older granitic rocks, and forming the base of the Orange Grove Quartzites, is met with in a few localities, and the extremely rapid variations in thickness and character shown by it are in striking contrast to the persistence and uniformity of character which distinguish those conglomerates which were deposited later in the history of the formation.

Summary of the principal sedimentary features of the Witwatersrand System.—Having now considered some of the main features of the Witwatersrand System, including the characters of the principal conglomerate zones, from their most marked and extensive developments in the latest phases of the formation to the much more limited occurrences of the earlier stages, it may be profitable to briefly summarise the principal points in the reverse order; that is, in accordance with their natural sequence in time. Such a review of the whole succession would show that apart from the minor oscillations between coarser and finer materials there is a more or less gradual and constant increase in coarseness of sediment from the base to the top of the formation, whether we take the average shown by the larger sub-divisions, or whether we separately consider the various

types of sediments. Thus, if we take the finer grained argillaceous or shaly members, we find that, proceeding upwards in the system, there is an ever-increasing tendency in the successive groups of beds of this type towards a coarser texture, and towards a more frequent admixture of the really fine-grained deposits with others of a much more sandy character. Many of the bands of so-called "slates" or "shales" include a large proportion of fairly coarse-grained sandy or flaggy types. This is particularly the case with the large group of slaty or shaly beds, referred to the Deppestown Series, whose deposition immediately preceded that of the beds associated with the Main Reef, and is also notable in connection with the so-called Kimberley Slates, which form the highest group of argillaceous beds.

Similarly if we consider the many bands of quartzites, we find a gradual but steady increase in the coarseness of texture from below upwards. Thus the Orange Grove Quartzites at the base of the system are the finest in grain of all the well-marked quartzite groups. The next in succession, known as the Hospital Hill Quartzites, are somewhat coarser, with a greater tendency to contain occasional gritty bands, and so on throughout the whole formation, until in the Kimberley and Elsburg beds, near the top of the system, we have almost exclusively coarser and more gritty types, which are developed to a larger extent than in any earlier portion of the System.

Finally, the various beds and groups of conglomerates exhibit very clearly an increasing coarseness of material, as we pass upwards in the system, and at the same time the thickness and number of the beds occurring in the various conglomerate zones show a marked increase. Thus, with the exception of very local occurrences of a basal conglomerate, no pebble beds are known to be associated with the lowest (Orange Grove) Quartzites throughout their 50 miles or more of exposed outcrops, and the Hospital Hill Group show only occasional thin bands of gritty material. The first well-marked pebble beds occur at the horizon of the Promise Reef, and are, for the most part, small pebbled and impersistent in character. At the horizon of the "Coronation Reef" a conglomerate band, with pebbles up to 1 in. or so in diameter, is more persistently found. The first really well developed and continuous conglomerate bed is that known as the "Government Reef," which is persistent for a distance along the strike of some 50 miles. Where most strongly developed it attains a thickness of 3 ft. or 4 ft., and its pebbles, with an average of 1 in., frequently reach 1½ in. in diameter. There is, however, only one well-marked bed of pebbles, although this may be accompanied in places by some smaller bands. Proceeding upwards in the succession, the next well-marked and persistent conglomerate horizon is that of the Main Reef Group, in which we have at least three very persistent conglomerate beds, accompanied by many lesser bands, occurring through a succession of strata up to 200 ft. in thickness. In this case the pebbles are usually about one inch in average diameter, considerably larger individuals up to two inches or so being frequently met with. Above the Main Reef horizon, the Bird, Kimberley and Elsburg groups of reefs show a progressive increase, both in the number of conglomerate beds in each group and in the thickness over which they are distributed. At the same time, the average and maximum sizes of the individual pebbles show a corresponding progressive increase from the Bird Reefs, where the pebbles, on the whole, are usually smaller than, or similar in size to those of the Main Reef Group, through the much larger pebbles characteristic of many of the Kimberley Reefs, to the well-known Elsburg Reefs, with their abundance of big pebbles which, in the most conspicuous bands, frequently attain a diameter of from 6 to 10 inches.

Changing conditions of deposition in the Rand area during the Witwatersrand period.

There is thus seen a gradually increasing coarseness of texture and a correspondingly increasing tendency towards the occurrence of conglomerates throughout the Witwatersrand System, from the bottom toward the top. These conditions, together with many other features of the formation, appear to indicate that during the long period occupied by its deposition the land surface from which the sedimentary materials were derived, occupied a position relatively to the locality of the Rand, which was progressively nearer to it. To allow of the deposition of some 20,000 ft. of sediments, there must necessarily have been a long period of subsidence in the Rand area, and the additional inference is frequently drawn that the sea in this region must therefore have been encroaching on a receding land. The gradually increasing coarseness of all

the sediments, when viewed apart from minor oscillations, to which attention has just been drawn, points, however, rather to the opposite conclusion. It seems highly probable that during the deposition of the Witwatersrand System there was a gradual encroachment of the land from the north or north-west upon the sea, and that although the locality corresponding to the present Witwatersrand and the area to the South of it were, during the deposition of the Upper Witwatersrand beds, intermittently subsiding, an area not far distant to the north or north-west, was undergoing a corresponding degree of elevation. The evidences of the probable denudation of certain of the Lower Witwatersrand beds in the later stages of the history of the formation, of the overlap of the upper division of the system on to the lower, and of the progressively increasing coarseness in general character of the Witwatersrand sediments, all point to this conclusion. The character of the floor on which the earliest Witwatersrand beds were laid down, and the widespread extent and wonderfully uniform lithological character and thickness of the beds in the lower part of the system, which in many examples, like the Orange Grove Quartzites and the well-known Hospital Hill Slates, maintain their individuality over hundreds of miles, show that these sediments were distributed over a very wide and uniform area, and it seems likely that they represent the discharge from one or more continental rivers of considerable magnitude, a view which is supported by the occurrence of individual beds of great extent, and of peculiar lithological character, like the Speckled Bed described above. At first the area corresponding to the present Rand lay at a considerable distance from such point of discharge and received only the finer sediments, but with the continued elevation of the land to the north and north-west from which the sediments were derived, and its consequent encroachment on the subsiding area, the Rand locality came more within the range of inshore, or more probably deltaic conditions. These appear to have prevailed in the Rand area particularly during the middle period of sedimentation, which included the deposition of many of the more persistent individual conglomerates, including some, at least, of those of the Main Reef Group, and particularly the Main Reef Leader. Later still the Rand area was very close to an actual coast line along which sediments of the Elsburg type accumulated.

It may here be emphasized that the existence of a pronounced and presumably a rocky coast-line in close proximity to the present Rand, along which the conglomerates were accumulated as beach deposits, which has been generally assumed by previous writers,[11] is rendered highly improbable by the nature and mode of distribution of the conglomerates of the Main Reef Group, especially the Leader. Many of the features shown by this and other beds associated with it seem to indicate that a contemporaneous land surface showing any degree of relief, and especially one composed of older rocks, could only have existed at a distance of very many miles from the locality at present occupied by the outcrop of the Main Reef zone, and was separated from it by a wide and almost level area over which the widespread sediments of the time were distributed conformably with the thick series of previously deposited beds.

II.—Origin and Distribution of the Gold.

In the preceding portion of this paper considerable attention has been given to the sedimentary features of the conglomerates, their probable mode of origin and their relationships to the other portions of the Witwatersrand System. This has been done because, in my opinion, these features have never received adequate consideration in connection with the question of the origin of the gold, attention having been largely focussed on the evidence available from within the conglomerates themselves, studied more or less in the form of petrological specimens or as mineral "lodes." It should be stated at the outset, that before making a detailed study of the whole system and having an opportunity of becoming personally acquainted with the conglomerates over the whole extent of the Rand, no reason was seen to reject the "infiltration theory" which was then, and is still, probably the most widely favoured of the various explanations suggested for the origin of the Rand gold. It would perhaps be more correct to say that my ideas had received a very distinct bias in favour of infiltrationist views. Closer acquaintance with many aspects of the conglomerates, however, soon raised so many questions which appeared to be answerable only when the gold was regarded as originally of

(11) *e.g.*: Gregory, *loc. cit.*, p. 79.

placer or detrital origin that I found it necessary for a long time to regard the matter as an open question upon which a decision could only be arrived at after much closer personal investigation of the evidence available.

As a result of such further study, I am strongly of opinion that the chief reason why the origin of the gold has so long remained a debatable question, especially with those who must rely for their data on the descriptions given by others, lies mainly in the fact that such descriptions have been largely based upon studies made in those portions of the Rand which were earliest opened up, namely, the Central and Western portions, and also that in descriptions of the conglomerates of the Main Reef Group no very clear distinction has been made between its various members. Had I been acquainted only with the Main Reef and with the Central and Western portions of the Rand, I might never have materially modified my inclination towards the infiltration theory. An extended study, however, of the Main Reef Leader, and of the conditions in the Eastern portion of the Rand area, appeared to place the problem in a very different light, and has led me to a strong belief in the detrital origin of the gold.

Relationship of the occurrence of gold to sedimentary features. From the point of view of the geologist, one of the most striking points in connection with the distribution of the Rand gold is the remarkable correspondence everywhere apparent between its occurrence and purely sedimentary features in the containing rocks. It will be convenient to deal with this correspondence first from the more general point of view of distribution of gold through the Witwatersrand System, and then more in particular with regard to its distribution within individual "reefs" and groups of conglomerates.

Distribution of gold throughout the Witwatersrand System. The occurrence of gold in the Witwatersrand System is far more widespread than is generally recognised. The outstanding position occupied by certain reefs of the Main Reef Group from a mining point of view is apt to give the impression that the distribution of gold is much more limited than it really is. In the first part of this paper a number of conglomerate groups have been briefly described, whose distribution throughout the Witwatersrand System is shown in the section in Fig. 1. Some of these, like the Elsburg and Kimberley groups, may include in any particular section a hundred or more individual "reefs" or conglomerate bands. After examining many detailed sections, in which assays have been made of many of these individuals, it would require some boldness to assert that any of the conglomerate bands throughout the system did not in some portion or other contain an appreciable quantity of gold, and quite encouraging assay results are frequently met with in groups like the Kimberley, which have furnished little in the way of actual mining propositions. Nearly all the known conglomerates in the Lower Witwatersrand System also occasionally show encouraging quantities of gold, and samples which include a very small thickness, frequently give very high assays. In one instance, at least, which has come under my notice, several thousand pounds worth of gold was very profitably obtained from a very small patch of "reef" only a pebble or so in thickness, and the extent to which these indications encouraged extensive exploitation of these Lower Witwatersrand reefs has been already referred to. Above the horizon of the Main Reef, the Livingstone, Bird and Kimberley groups have all furnished reefs which have been actually mined to a considerable extent, and which may assume greater importance at some later stage in the history of the field, when the more attractive propositions now available have ceased to absorb the available resources in capital and labour. In some portions of the groups just referred to, the difficulty appears to be not so much to find a reef which shows encouraging values, as one which maintains its character over a considerable lateral extent.

It might perhaps be said of some of the reefs in the Lower Witwatersrand System that it is the absence of any large body of conglomerate rather than the nature of the values found in that which does occur, which has prevented such reefs from becoming important from a mining point of view, whereas in the Upper Division of the System it is the distribution of the gold which is present, through such large bodies and such very numerous bands, that has led to the same result; for it seems quite possible from what we know of them that had the gold distributed through the Kimberley or the Bird reefs been confined to a more limited zone as in the Main Reef

Group, that we might have had something equally important from a mining standpoint. It is not so much the actual amount of gold present in the Main Reef Group which appears exceptional as its concentration within certain particularly well-defined and continuous beds, and, as will be presently shown, this concentration coincides with, and is probably the result of, special conditions of sedimentation.

From the above it will be seen that gold is of very general distribution throughout the whole Witwatersrand System, wherever conglomerates occur, and as far as we know, it is practically limited to such beds or to the quartzites in their immediate neighbourhood, and, further, there appears to be a very close correspondence between the sedimentary character of the various groups of conglomerates and the extent to which gold is found associated with them.

Distribution of the gold within the conglomerate beds.—Similarly it we study the distribution of the gold within the limits of the individual conglomerate beds themselves a close relationship is usually seen between the occurrence of gold and purely sedimentary features.

Perhaps the most striking example of that is seen in connection with the Main Reef Leader in the far Eastern Rand. The mode of distribution of the conglomerate in the Nigel neighbourhood in well defined "patches" or "shoots" has been already described. The occurrence of gold in any notable quantity is practically limited to these patches of conglomerate, and in them gold is rarely absent, and this applies not only to the larger patches, but also to the numerous smaller ones which occur in the intervening spaces as shown in Fig. 4. The explanation given by the infiltrationists of the presence of gold in the conglomerate as due to their acting as channels for the mineralizing solutions, finds many difficulties of application in this case—for even granting the disputed point that the conglomerates offer easy channels as compared with the associated beds, it is difficult to see how this principle can apply in the far Eastern Rand. For the patches are separated in many instances by wide intervals occupied by beds for which no greater suitability for the passage of solutions can be claimed than for the general bulk of the formation. They are practically isolated from each other by intervening stretches of the same quartzites which are supposed in other cases to have restricted the mineralizing solutions to definite channels offered by the conglomerates. In a building, a number of spacious halls would be of little use for the passage of a moving crowd if they were practically cut off from one another or connected only by narrow and difficult passages. Where the conditions are such as are met with in the Central Rand or are similar to those described for the Van Ryn, where the "shoots" are connected by intermediate areas also occupied by conglomerate, this intermediate material can always be requisitioned on the infiltrationist view to supply the necessary channels of transmission, but under the conditions prevailing on the Nigel area the infiltrationist idea of the conglomerates as widespread channels of communication does not appear in any way to meet the case. The irregular disposition of values in the Main Reef, particularly with regard to their vertical distribution, corresponds with the impersistence of the various bands in that reef and with the irregular distribution of such larger pebbles as may occur, and this is in contrast to the much greater localization of values frequently evident in the Main Reef Leader corresponding to the grading of the pebbles in that bed, and especially to the frequently strongly marked preference of the gold for the foot of the reef, particularly when the foot is characterized by the presence of numerous pebbles above the average size. The great frequency with which unusual values are associated with large pebbles, particularly on the foot of the Leader, is remarkable, and it is especially worthy of note that this association is not limited to bands of larger pebbles extending over some considerable distance, but is equally or perhaps more characteristic of isolated groups of larger pebbles, and even of single individuals. I have frequently examined portions of the foot of the Leader which included such nests of large pebbles or single individuals, and in reef of moderate quality have rarely failed to find that specimens obtained from such a situation exhibited 'visible' gold, which as is well known, is not of very general occurrence in the conglomerates. On the assumption that the gold is an original constituent this is a most natural and easily explained arrangement for which we find innumerable parallels in typical placer deposits. On the infiltration hypothesis, however, it is difficult to find any feasible explanation of this association, particularly in the case

of the large pebbles which occur in nests or groups or as isolated individuals. Some definite explanation appears to be due from the infiltrationists as to how an auriferous solution traversing a sheet of conglomerate, hundreds of square miles in extent, should pick out small groups and single individual large pebbles in this extraordinary way. One could understand how a continuous tract of such larger pebbles might possibly allow larger quantities of the gold bearing solution to pass and thus permit of a heavier deposit of gold in such an area, but when, as is often the case, such larger pebbles occur within patches of conglomerate entirely isolated from the main body, any such influence on deposition appears very difficult of satisfactory explanation. Still more striking perhaps are the conditions presented by the well known "Bastard Reef" of the Glencairn, May Consolidated and neighbouring mines in the near East. Typically, this "Bastard" consists of pebbles of Main Reef type sparsely scattered through a dark fine grained matrix, which originally must have been a mixture of muddy and very fine sandy material, similar to that constituting the "Black Bar." In places the Bastard can be seen to grade into the Main Reef, in others into the Black Bar, and there can be little doubt that the Bastard originated by the mixture of Main Reef pebbles with finer material of the nature of the Black Bar, probably in many cases along the margins and the beds of channels cut through the pre-existing pebbly deposits of the Main Reef horizon. The Black Bar in itself is practically everywhere devoid of any notable amount of gold, yet in such admixtures with Main Reef material it acquires values such as might be expected from this mode of origin, and it has been largely mined. Its value appears to be proportionate to the quantity and grade of Main Reef material incorporated with it. Typical "Bastard" with its widely scattered pebbles in a compact matrix, appears an extraordinarily unsuitable medium for the passage of mineralizing solutions required by the infiltrationist theory, but, on the other hand, the distribution and character of the values found in it present no difficulty on the supposition that the Main Reef gravels, as appears probable on so many other grounds, already contained gold before their admixture with the Black Bar material. The above are but a few among many points in connection with the occurrence and distribution of the Rand gold which appear to receive a much simpler explanation on the placer view than in any other way, and it seems likely that these will be considerably augmented as the exploration of the Far Eastern Rand progresses. It is, however, not possible to deal more fully with this question within the limits of the present paper, except in so far as is necessary in considering some of the objections which have been raised to the placer theory.

Objections which have been raised to the placer theory of the origin of the Rand gold.—An objection which has frequently been raised to a placer origin for the Rand gold is that the deposits with which it is associated are not similar to those in which the majority of known placer deposits occur, and especially those which have been most conspicuous as gold producers. This objection appears to be largely the result of ordinary stream-placers having been mainly used as a basis of comparison on the one hand, and on the other of attention having been largely directed to a consideration of such admittedly exceptional "reefs" as the Main Reef Leader and the South Reef, thus of leaving out of account the hundreds of other conglomerate bands which occur throughout the Witwatersrand formation, and which probably afford a much safer basis for comparison. In the character of the sedimentation and in the quantity and distribution of the contained gold, such conglomerates as those of the Bird and Kimberley groups appear to differ in no essential features from many well-known placer deposits, and to these might be added many members of the Main Reef Group, including the Main Reef itself, and numerous impersistent bands of conglomerate frequently found above and below it. For while it is true that these deposits have little in common with the more ordinary type of stream-placer, they show a very close correspondence with certain placer deposits of coastal origin. The description of the Alaskan "gravel-plain placers" given by Brooks and others, abound with points in which these deposits show close similarity to the Rand conglomerates and their associated beds.

Comparison with the coastal-plain placers of Nome, Alaska.—The coastal gravel-plains of Alaska appear to have been formed by material of the nature of small delta deposits brought down by local streams, and

more widely distributed by coastal currents and other agencies. If allowance be made for the vast difference in the scale of deposition in the two cases, and for the fact that the Nome deposits have been laid down close to a coast line of older rocks on a comparatively uneven floor, whereas the Rand conglomerates were distributed at a distance from the actual shore line over a widespread level expanse composed of previously deposited conformable beds, the similarities between the two occurrences are many and close. There are few features of the more general types of Rand conglomerates that do not find a parallel in the Nome coastal gravel-plains and in the beaches which have been locally formed in them from time to time. Of the various types of sedimentation exhibited by the Rand conglomerates, such beds as those below the Main Reef (Main Reef Footwall Quartzites), with their impersistent bands of conglomerate, sometimes lying on a bed of clayey character, may be compared to the gravels of the coastal-plain deposits, with which also such groups as the Bird and Kimberley Reefs have much in common. The Main Reef itself appears frequently to correspond more nearly in type, but on a very much larger scale, with the Nome "beach-placers," and its origin may be, as suggested for these, due to concentration of gravel-plain deposits by wave action, with richer patches determined by the action of local streams crossing the surface of the beach.

When, however, we come to such uniform and extensive sheets of conglomerate as the Main Reef Leader and the South Reef of the Rand, which in the upper Witwatersrand System are the only two known occurrences of the kind amongst hundreds of others for which parallels can be found in the Nome deposits it is not surprising that an analogue is not so easily discovered. It is, however, not difficult to see how in the Nome area an exceptional discharge from one of the rivers which traverse the Nome coastal-plain could give rise on a small scale to a deposit of essentially the same character as the Main Reef Leader, and the production of such a bed would be much more likely to occur under the widely spread uniform conditions assumed to prevail in the Rand area when the Leader was laid down and as a result of the redistribution of very much larger quantities of material than would be possible at Nome.

It appears, then, that there is nothing in the sedimentary characters of the Rand gold-bearing conglomerates which is incompatible with the idea that the gold was an original constituent of the beds. Nor does there appear to be much in support of the further objections which have been made to the acceptance of a detrital origin for the Rand gold that the values found in the conglomerates as a whole are of a different order to those characteristic of placer deposits, and that the Rand gold is too fine in character to be of placer origin, the rare occurrence of anything like a nugget being quoted in support of the same opinion. With regard to the former objection, the tendency to make comparisons with the exceptional values found in certain reefs and in special portions of the Rand rather than with the average must be remembered, for if the extensive low grade areas of the Rand, and also reefs outside the Main Reef Group, were more often taken into account as a basis of comparison, greater similarity would be certainly found. Where concentration of the Nome coastal-plain gravels has taken place, as in some of the beaches which have been formed in them by wave action, values of a high order are also met with. In one of these beach-deposits values as high as 500 dollars to the pan are recorded, and from one area 100 ft. × 15 ft. in extent more than 330,000 dollars was taken, 90 coming from the bottom 3 inches of the pay-streak.[12] With regard to the second objection, the fineness of the gold in the Nome deposits is frequently referred to in the descriptions given of them. Brooks[13] states that anything like a nugget up to the value of one dollar is very exceptional, and that the average grains as saved by panning run from 70 to 80 to the cent., but that much of the fine gold is lost in separation, thus increasing the average size of the colors that are saved. He further mentions that beach gold from Randolph, Oreg., averages 110 colors to the cent., that from the Silver Mine, Denmark, Oreg., about 600 to the cent., and the river-bar gold from Snake River, Idaho, from 900 to 1,000 to the cent.

It will thus be seen that there is nothing remarkable in the absence from the Rand conglomerates of coarse gold, or of nuggets, for considering that such beds as the Main Reef Leader were probably laid down

(12) "Geology of the Nome and Grand Central Quadrangle, Alaska," F. H. Moffit. *U.S. Bulletin*, 533, 1913, p. 117.

(13) A. H. Brooks, "The Gold Placers of Parts of Seward Peninsula, Alaska." *U.S. Bulletin*, 328, 1908, pp. 153–154.

many times further from the actual shoreline than those of Nome the gold in them might naturally be expected to be of still finer character. It is perhaps worthy of note that in the Nome gravels, as in the reefs of the Main Reef Group, the pebbles are all of comparatively small size, individuals of more than an inch in diameter being rare. The Nome deposits are regarded as the redistributed material of small deltas, and the small range in the size of their pebbles and the rarity of larger constituents is no doubt due to the elimination of larger pebbles and boulders which this mode of origin entails, a process which must have been operative to a very much greater extent under the conditions governing the deposition of the Main Reef conglomerates.

In connection with the question of the fine character of the Rand gold and the absence of anything of the nature of nuggets, it is interesting to consider what would be the result if the Rand deposits as they exist to-day could be subjected to rapid denudation, removal and redeposition.

The pebbles from the existing conglomerates would probably form part of new beds possibly of a similar character, and with them might conceivably be associated a large part of the existing gold, and this would presumably be in the same finely divided state in which it now exists. Anything in the nature of a nugget would be of extremist rarity; even if the new deposits were laid down at no great distance. Would therefore future generations of geologists be justified in saying that the gold could not be of detrital origin because of its uniform fineness? Moreover, under such circumstances, large quantities of new quartz-pebbles formed from the debris of the innumerable small veins occurring on the Rand would be added to those derived from the present conglomerates, yet few of this multitude of new pebbles would be in the least likely to contain any gold, since the quartz reefs from which they would be derived are rarely found to show any values. Yet the absence of gold in these pebbles might be used as an argument against the detrital origin of the gold then associated with them.

The Infiltration Hypothesis. Difficulty of application in the Far East Rand. The conception of the conglomerate beds, on the infiltration theory, as easy routes for the passage of ascending gold-bearing solutions, by which the conglomerates were converted into something bearing a close analogy to quartz-veins, probably owes its origin, and still more probably the continued advocacy it has received, to the study of sections, either in nature or on paper, in which these reefs appear in a vertical or highly inclined position as is the case with sections taken from most outcrop mines in the Central and extreme Western portions of the Rand. As usually represented in such sections, the conglomerate beds suggest analogy with a highly inclined fissure, filled in this case by a pebble-bed whose interstices afford an easy passage for ascending solutions. The suggestion is so strong, and the explanation of the deposits thus afforded appears so simple and attractive, that it is embraced with readiness, and it is one which in the earlier stages of my acquaintance with the Rand appealed to me as forcibly as it has done to others. In sections and other illustrations of the Rand reefs it is of course usually quite impossible to represent them in full detail, or on a correct relative scale to the remainder of the section, and as a result such sections are usually very diagrammatic in character. This will be readily understood by reference to the figures in such well-known accounts as those by Hatch and Chalmers,* and by Truscott. In viewing such necessarily diagrammatic figures one is apt to forget that the conspicuous spaces which appear between the pebbles as represented are really very fully occupied by a matrix which may be practically as little pervious to the passage of solutions as the beds outside the limits of the conglomerate are usually presumed to be. In sections illustrating the more general relationships of the conglomerates, as for example those given with Hatch's map of the Southern Transvaal and similar figures, the band of conglomerate representing the Main Reef is necessarily shown much larger than it really is. In the example just mentioned, at the Main Reef horizon there is usually represented a band of conglomerate about 200 ft. in thickness taken on the scale of the map, whereas, of course, 20 ft. for the aggregate thickness of all the pebble bands in this zone would be a very liberal measurement, and this 20 ft. would not be in a single body, but distributed in several layers through a thickness of quartzite ranging up to 200 ft. or more. If now instead of the Central and Western Rand with sections like these, so suggestive of

* e.g. H. & C. *Gold Mines of the Rand.* Figs. 44, 50, &c.

easily traversed channels leading downwards in the direction of whatever extraneous source of gold seemed most desirable as the origin of the mineralization, our first acquaintance with the gold-bearing conglomerates had been, say, in the central portion of the wide and comparatively shallow basin of the Far East Rand, where the conglomerate beds lie almost horizontally and can be followed laterally through mines and borehole sections for many miles in all directions, the inclination to view them as comparable to ordinary lodes would probably have been very much less. The suggestion that they themselves had supplied the channels by which their auriferous contents had arrived at their present position would not have been so obvious nor indeed so plausible, especially when in addition it were remembered that in the Far East the conglomerate frequently occurs in isolated patches and not as a continuous reef. We should then probably have been less inclined to accept the conglomerates themselves as channels for auriferous solutions, and should have sought other sources of supply. These might have been looked for in the form of "verticals" bringing up the solutions from below to be distributed far and wide along the slightly inclined conglomerate beds. It is, however, a very significant fact, which tells strongly against such extraneous origin for the gold, that in the great extent of country which can be said to have been subjected to close examination, and which has been traversed in all directions by a network of mine-workings, the total length of which must run into hundreds of miles and is increasing every day, nothing in the nature of such channels of supply has yet been discovered. And when we consider that over very large areas in the Far East Rand the "reef" lies directly on a "slate" footwall from which it is very carefully stripped, and in which definite channels of supply could hardly escape being fairly conspicuous, the failure to detect any such channels is the more remarkable, especially as the richest patches which one might expect to be most closely connected with any possible "verticals" are among those which have been most completely explored, worked and otherwise subjected to the closest scrutiny. And if we waive the necessity for evidence for the existence of channels of supply we are still confronted with some very awkward questions, for we have either to assert that a certain special horizon a few inches thick, out of several thousand feet of associated strata has some special quality which leads to its selection by the gold-bearing solutions or we have to explain why numerous other bands of pebbles apparently offering far easier channels of circulation have been passed over or only favoured to a much less degree. If we invoke the aid of the "slates" underlying the Main Reef Leader in the Eastern Rand as a determining influence we must offer some explanation as to why similar bodies occurring at no great distance both above and below the Main Reef horizon, and also frequently associated with extensive bands of conglomerates, did not equally affect the deposition of the gold. Some influence might more readily have been conceded to the slates had the conspicuous deposit of gold been along their *under* side, but as it occurs *above* them it is not easy to imagine the source or the course of the solution which deposited it, and it would be extremely interesting if some advocate of the infiltrationist hypothesis would give us some sort of diagrammatic representation, drawn approximately to scale, of the Far Eastern Rand, showing the probable route of the gold-bearing solutions from an outside source to its present wide distribution over practically the whole of the conglomerate sheet which forms the principal reef in that district, including the isolated patches, large and small, which are characteristic of the Nigel area. In constructing such a chart or section it would have to be borne in mind that on a liberal allowance the conglomerate band forming the most favoured horizon for the deposition of the gold would bear to the remainder of the Witwatersrand succession about the same proportion and relationship as a single leaf in a book of 6,000 pages does to the remainder of the volume. When the favoured conglomerate sheet affords so much evidence of having resulted from a continuous act of deposition as does the Main Reef Leader on the Eastern Rand, the limitation of placer gold to the bed so formed is a possible and natural consequence of the mode of deposition. On the other hand, for circulating solutions to follow a single horizon so faithfully over hundreds of square miles is a much more difficult operation to conceive. In fact, when we come to deal with conditions such as are met with in the Far Eastern Rand, the theory of infiltration from any outside source of supply demands so extended a series of purely

hypothetical steps uninterrupted by any solid ground of observation and of demonstrable fact, that any one may be pardoned who comes to regard the further pursuit of an explanation in this direction as demanding a greater exercise of the imagination than is usually allowed in geological speculation. If the section represented in Fig. 1 be imagined to be reduced somewhat in thickness to correspond to the decrease in thickness of the formation in the Eastern Rand, and then extended laterally about six times the width of the page, corresponding to a distance of 25 miles, or about 130,000 ft., it would then represent on an approximately natural scale a section across the basin of the Eastern Rand from North to South. Assuming the Reef in that area to average 3 ft. in thickness, it would be represented in the section by a line less than one-thousandth of an inch thick, which would lie immediately above the band representing the "foot-wall slate" of the East Rand. On the infiltration theory we have now to suppose that solutions from some extraneous source travelling upwards through the many great bands of slaty rocks represented in the section, select a particular one which is not known to possess any special characters and is not even the uppermost, and travelling laterally, follow faithfully the upper surface of that particular body of slate over the whole of the section, representing in nature an area of hundreds of square miles. As an alternative we may suppose that the solutions, entering from above, pass downwards through a multitude of conglomerate bands in the Elsburg and Kimberley groups, which, to all appearances, offer greater facilities for lateral movement, pass also through the big group of Kimberley slates and again unerringly pick out for the deposition of the major portion of their gold a single bed of conglomerate, which over a large portion of the area is broken up into isolated patches with no apparent lateral connection. When viewed on a proper scale the difficulties in the way of such a course of events will appear to most enquirers so great as to render this mode of introducing the gold into the Reef of the Eastern Rand still more difficult of application than it is in the Central area.

Possible influence of dykes.—Before leaving the question of the introduction of the gold into the conglomerates along definite channels of supply, the possible action of dykes in this direction may be briefly considered, since they have frequently been suggested as the possible source of mineralization. In connection with visits to nearly all the principal mines of the Rand I have made a point of enquiring whether any instance had been found of a distinct connection between a dyke and the general distribution of gold in the conglomerates. Up to the present I have not been able to find a single case in which such a connection was clearly shown. In almost every instance the experienced mining men of whom my enquiry has been made, have been quite definite in stating that they were unable to point out any example of clear connection between dykes and the values shown by the reef, except for extremely limited and local effects which belong to secondary movements of the gold such as will be referred to later. In one or two cases which have been suggested as possible instances, the probability of a different explanation being more applicable has been recognised and pointed out to me.

In this connection the Far Eastern Rand with its simpler conditions, again appears to afford us the best field for judging of the possibility of dykes being concerned in the introduction of gold. And, again, the Nigel Mine in particular appears to offer us a better means of testing this hypothesis than is available anywhere else. As already pointed out, the completeness with which this mine has been explored and worked makes it a particularly useful field of investigation. The very marked distribution of the conglomerate in well defined patches to which the gold is entirely confined has been described; a practically complete plan of these patches appears in Fig. 3, and with them are shown the courses of all the principal dykes. It will be seen that these dykes traverse both auriferous patches and barren intervals in a manner which might have been specially devised to afford an answer to the question of their possible connection with the gold content of the reef. Yet no case is known to the officials on the mine in which any such connection was evident. The dykes passed through rich and poor conglomerate and barren ground indifferently. It will be noted that in some cases the course of a dyke may coincide for some distance with the margin of a patch of conglomerate. In such a case before the conditions were fully known it might have been supposed that the dyke was the cause of the difference between

the conditions met with on either side of it. This would have seemed still more possible if the patch of better ground had been surrounded by an area of inferior reef as is usually the case nearer to the Central Rand. In several such cases, however, when at first a connection has been supposed to exist between a dyke and the richer or poorer reef found on either side of it, more extended working of the area has shown that the dyke when followed further had in other portions of its course richer or poorer ground on *both* sides of it, and that there was no real connection between the dyke and the gold distribution.

The coincidence of a dyke with a fault-plane along which richer and poorer ground have been brought together by lateral movement is, of course, in some instances the means of bringing about an apparent connection between the dyke and the character of the ground in its vicinity.

Against the experience of the great majority of mining engineers that it is exceedingly difficult to trace any connection between the numerous dykes which traverse all parts of the Rand and the distribution of the gold, we have the advocacy of such a connection by Mr. Horwood, who cites as especially notable examples, dykes at Randfontein and Rietfontein. The assays from the dykes given in the appendices to Mr. Horwood's paper do not appear very encouraging, especially when one compares them with the values shown by the conglomerate which they are supposed to have supplied with gold. The derivation of the gold in the dykes from the bankets near or through which they pass would appear to many a more probable deduction. Moreover, evidence is forthcoming which to most enquirers would appear to require a good deal of explanation if the dykes are the source of the gold. A number of sections from the Randfontein area have recently come under my notice, of boreholes which passed through a large number of conglomerate bands, all of which were assayed. The evidence from all the examples is similar, and a few details of one may be given as typical. This hole intersected a numerous series of reefs lying about midway between the Randfontein Leader and the Monarch Reef (both well-known gold-bearing horizons). With the reef a number of dykes were also intersected. In one portion of the hole a dyke 250 ft. thick was met with, having a reef immediately above it with a smaller dyke above that again—yet this reef, sandwiched between two dykes and, from the description, eminently suitable for the deposition of gold, gave only a trace. In another portion of the section a dyke occurs which is 90 ft. thick with 65 in. of reef lying directly on it, which also gave only a trace of gold, while 30 in. of reef directly below the dyke gave the same result. Yet in a third portion of the section, a reef only 6 in. thick which is 200 ft. from the nearest dyke, gave 2 dwt., and one of 7 in. far distant from any dyke gave 3·5 dwt.

In endeavouring to supply a universal source for the gold and associated minerals, not only of the Rand bankets but also of the Black Reef and Dolomite in far distant parts of the Transvaal, Mr. Horwood seeks to connect all these, through the agency of dykes, with the period of volcanic activity associated with the Ventersdorp Amygdaloid.

The suggestion, as far as I am aware, is entirely novel, and that it has been made so late, among the multitude of alternatives, is probably due to the vast amount of field evidence which appears to be so directly opposed to it, and which has probably eliminated it very early from the minds of most observers as a possible explanation of the origin of the Rand gold. Some of the views on which Mr. Horwood appears to base his suggestion, as, for example, that the shales normally interbedded with the Black Reef quartzites in the Randfontein district are to be interpreted as intrusive sheets of Ventersdorp Amygdaloid,[14] certainly require revision, as they appear to be directly contrary to field evidence of the clearest kind. It is also a noteworthy fact in this connection that although so vast a mass of the Ventersdorp Amygdaloid lies immediately to the South of the Rand and once overlaid that district, the Elsburg group of conglomerates which is nearest to the igneous mass, is probably the poorest in gold of all those associated with the Witwatersrand System.

Secondary Changes in the Conglomerates.—One of the main reasons for the divergencies of opinion which exist as to the origin of the Rand gold, is undoubtedly the condition in which the gold now exists and its association with many minerals regarded as characteristic of deposits of the nature of veins or lodes.

These considerations attain greater importance when studies of the reef are large-

(14) *Trans. Geol. Soc. S.A.*, Vol. XII., 1912, pp. 77–80.

ly directed to the examination of specimens from a mineralogical or petrographical standpoint, rather than to the geological and stratigraphical characteristics of the beds. And in the case of many who have not been able to make an extended personal acquaintance with the Witwatersrand, the former class of evidence is likely to have the more weight because the papers in which it has been embodied are perhaps much more exact and scientific than those dealing with the latter aspects of the question, these being mostly of early date or confined to a very limited portion of the system. For that reason stratigraphical features have been given greater prominence in the present paper, and it is not proposed to add to the excellent descriptions of the mineralogical characters of the conglomerates which are available,[15] except to enquire to what extent the conclusions which have been drawn from mineralogical features are opposed to the possible placer origin of the gold.

Prominent among the arguments which have been used against the detrital origin of the Rand gold are those connected with the fine state of division in which the gold occurs, its crystalline character, and the absence of waterworn scales or fragments and of nuggets.

If the conglomerates originated as suggested in the earlier portion of this paper, either as simple deltaic deposits, as in the case of the Main Reef Leader or in modified form, redistributed by coastal currents, then the fineness of the gold and the absence of anything of a nuggety nature is only natural and in agreement with such deposits as those of Nome, and many other marine placers where the conditions are not even so favourable for the elimination of the coarser gold as in the Rand deposits, and instead of the crystalline character of the gold being an unexpected phenomenon, surely it would have been a matter for surprise had the original gold escaped solution and re-crystallization when we reflect that the Rand conglomerates are almost certainly of Pre-Cambrian age, that they have undoubtedly more than once occupied a position many thousands of feet below the surface, and have been subjected to all the vicissitudes which this submergence entails, and further that apart from such a simple change as that involved in the re-crystallization of the gold, the conglomerates in common with the whole formation abound in evidences of profound and prolonged metamorphic action. In connection with the probable solution of the minerals originally present in the conglomerates and their re-precipitation in new forms in the same bed, and in close proximity to their original positions, it is somewhat remarkable that reference has not been frequently made to some very interesting observations on certain deposits in the old Roman baths connected with the thermal springs of Bourbonne-les-Bains, recorded by Daubrée, *Geologie experimental*, pp. 71-90). The thermal waters have a temperature of about 150° F., and contain alkaline chlorides and sulphates. In carrying on certain operations in connection with these baths in 1874, there were found embedded in the mud and sand below the old Roman works and near the point of issue of the spring, a large collection of articles, including rings, pins, statuettes, frames, etc., together with some thousands of silver, bronze and pewter coins. Four gold coins were also found. One of the lowest layers of the deposits, which consisted of rock fragments and of coarse sandy material, contained hundreds of coins which showed all stages of solution by the thermal waters, in some cases an imprint of the coin alone remaining. The material derived from the solution of the metals had been re-deposited in the enclosing sandy matrix, which was thus for the most part cemented together by various minerals, which frequently exhibited well developed crystal forms. In some cases these new minerals had bound together the sandy matrix immediately surrounding an enclosed coin, so as to form a flattened spheroidal mass.

Among the new minerals formed by the solution of the bronze, silver, lead, iron, etc., were black powdery oxide of copper, well-defined crystals of red copper, grey copper and copper-pyrites, comparable to the minerals in the lodes of Redruth, also well defined crystals of galena. Crystals of pyrite were also found deposited upon fragments of sandstone and upon worked flints associated with the other ancient objects.

A feature of these deposits, which is specially notable in the present connection, is that although the circumstances were

(15) "The Petrography of the Witwatersrand Conglomerates, etc.," Hatch and Corstorphine. *Trans. Geol. Soc. S.A.*, Vol. XII., 1904, pp. 140–145.
"Notes on the Auriferous Conglomerates of the Witwatersrand," R. B. Young. *Trans. Geol. Soc. S.A.*, Vol. X., 1907, pp. 17–30.
"Further Notes on the Auriferous Conglomerates, etc.," R. B. Young. *Trans. Geol. Soc. S.A.*, Vol. XII., 1909, pp. 82–101.

apparently very favourable to the removal of the minerals formed as a result of the solution of the coins and other objects, these minerals were largely re-deposited in the *immediate vicinity*, and in some cases actually in contact with the coins from which the minerals in solution had been derived. Under the very much slower and more restricted circulation which might be expected to have occurred in the conglomerate beds of the Rand, the re-deposition of minerals in close proximity to their place of origin might be much more reasonably expected. If such changes as those described can be brought about by such insignificant agencies in comparatively so brief a time, the re-crystallization of the Rand gold seems scarcely a matter for surprise. The association of the gold on the Rand with pyrite is frequently quoted in favour of the theory of the introduction of both these minerals into the conglomerates from external sources. While it is no doubt true that some of the pyrite has been so introduced, an adequate study of its distribution in the reefs, and especially in some of the quartzites associated with them, would probably convince most observers that much of the pyrite represents a modification of some of the original constituents of the rock. For frequently this distribution of the pyrite in relation to bedding-planes, current bedding and other sedimentary features of the conglomerates, and especially of the associated quartzites, as, for example, in the Main Reef Leader, the South Reef and the "pyritic bands" associated with them, and in particular with the Main Reef, is precisely similar to that of iron oxides, such as ilmenite and magnetite in sediments of recent origin. The natural inference, as suggested by Gregory and others, is that much of the pyrite we see so distributed represents original oxides of iron. From the chemical point of view there seems no difficulty in the way of accepting such a natural interpretation. The conversion of oxides into sulphides has been artificially effected by several experimenters and by different methods.[16] Clarke mentions that Doelter[17] prepared pyrite by heating haematite, magnetite or siderite with hydrogen-sulphide and water for 72 hours to 80° or 90°. If so little time and such a moderate temperature suffices for the conversion of magnetite into pyrite, the assumption that such a change has taken place in rocks which are most probably Pre-Cambrian in age and which have certainly for long periods been subjected to the conditions prevailing thousands of feet below the surface, appears quite a modest one, and surely demands far less indulgence than the supposition that such pyrite has been introduced from outside sources and has disposed itself entirely in conformity with the usual mode of distribution of such constituents as the oxides of iron in sedimentary rocks.

In this connection, it may be recalled that iron oxides, and also pyrite itself, are among the most usual and abundant companions of placer gold. Pyrite figures constantly as a constituent of placer deposits in descriptions of the Nome gravels and of the Tertiary gold-bearing gravels of California.

The evidence for the derivation of much of the pyrite associated with the Rand gold from constituents originally present in the conglomerates is very strong. There can be no doubt, however, that considerable quantities of pyrite have also been introduced at some period subsequent to the deposition of the conglomerates. The existence of two different generations has been emphasised by Young,[18] and indeed it seems very probable that in more than one phase in the history of the conglomerates pyrite was added to that derived from the original constituents of the rocks. The accession of some of this pyrite to the reefs was probably associated with the formation of numerous mineral veins met with in the Witwatersrand formation which are of comparatively late origin.

In these veins, in addition to the coarse vein-quartz, which usually forms the bulk of the filling, galena, zinc-blende, copper-pyrites and other minerals usually associated with vein deposits are frequently met with. Quartz-veins occur very abundantly in various parts of the Rand, and their outcrops strew the surface with abundant quartz-debris. In the earlier days of the goldfields these veins frequently attracted the attention of prospectors, and a large number of them were opened up and tested. Some of the very earliest mining on the Rand was done by Struben on a vein of this character prior to the recognition of the importance of the conglomerates as a gold-bearing formation. The greater number of the veins show very little indication of con-

(16) F. W. Clarke, "Data of Geo-chemistry," U.S. Geo. Survey, *Bull.*, 491, 1911, p. 30C.

(17) *loc cit.*, p. 317.

(18) *Trans. Geol. Soc., S.A.*, Vol. XII., 1909, p. 95.

taining much of interest: the great majority appear to be entirely barren, and they have long ceased to attract the smallest attention from mining men. It may, therefore, be said that the numerous quartz-veins which occur in the Witwatersrand rocks on the Rand are characterized as a whole by marked poverty in minerals of economic value, including gold. Remarkable exceptions, however, are frequently found where such quartz veins intersect the gold-bearing conglomerates. In these cases the veins in the immediate neighbourhood of the conglomerate may carry gold in phenomenally large quantities, and some of the most remarkable gold specimens from the Rand have been obtained from such occurrences. In one of these, which I had an opportunity of examining in detail before it was removed in the course of mining, the relationship of the vein to its surroundings, the secondary nature of the deposit as a whole, and the derivation of the gold from the adjoining conglomerate were exceedingly clear. The occurrence was met with on the New Goch Mine at a point, where, in close proximity to a fault, the reef had been thrown into a very perfect fold about 50 ft. in width. In the neighbourhood of this fold the rock was traversed by large numbers of small irregular veins and lenticular masses of coarse quartz, which had filled the small fissures which originated as a result of the folding. It was in one of these which intersected the foot of the Main Reef Leader and the upper portion of the underlying "Black Bar" that the coarse gold was found. The relationship of the quartz to the Leader will be best understood from Fig. 6. The gold occurred enclosed in the hungry white quartz as shown in the figure. Some of these patches of coarse gold attained a diameter of a third of an inch or more. They were entirely confined to the immediate neighbourhood of the foot of the Leader, the more distant portions of the quartz mass being barren, as were also numerous other similar masses which occurred in association with it. The Leader in this particular locality showed very good values, which, as so frequently is the case, were associated more particularly with the actual foot of the reef along its junction with the underlying Black Bar. The obvious secondary nature of the deposit of coarse gold, its deposition within a few inches of its most probable source, and the absence of even small quantities of gold from the same body of quartz at any greater distance are all points of the greatest interest in connection with the question of the re-crystallization of the gold in the conglomerates, and its re-deposition after solution within a very short distance of its original position. Processes closely similar to those which took place locally in the formation of these

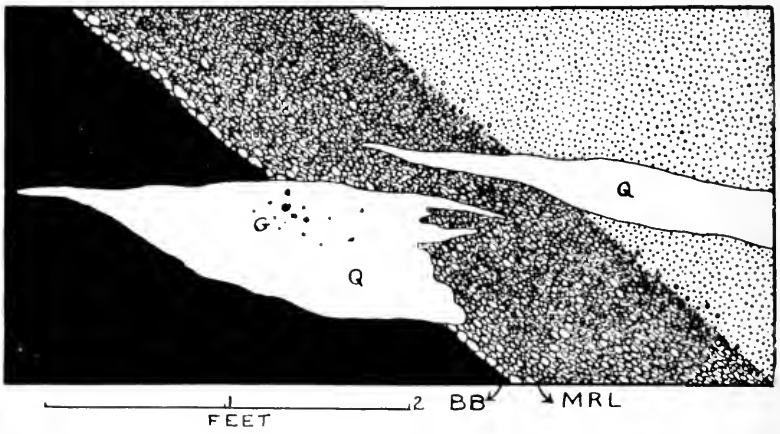


FIG. 6.—Irregular masses of vein-quartz (Q) crossing the Main Reef Leader (MRL) and the Black Bar (BB), and showing coarse gold (G) limited to the neighbourhood of the foot of the Leader. (New Goch Mine.)

small veins, would, on a more extended scale, account for the changes necessary to convert the original placer deposits into their present condition.

Similar occurrences of coarse gold associated with veins and masses of quartz are known from many mines. An example has been brought to my notice within the past few days from the Robinson Deep. In this case a quartz-vein of a more continuous character than those mentioned from the New Goch is found in one locality, where it traverses the South Reef, to contain an abundance of very coarse gold which in some cases occupies a conspicuous proportion of the surface of the specimen. The quartz in this case frequently shows clearly its individual crystals, with occasional small drusy cavities. The coarse gold occurs between the quartz crystals, and is moulded upon them in such a way as itself to appear very coarsely crystalline. Galena is also present in considerable quantities with other minerals not yet determined. Followed above and below the conglomerate of the South Reef, the vein ceases to carry gold, which appears to be limited to its intersection with the auriferous conglomerate. It is the coarse masses of gold found in similar veins, or in some cases perhaps recovered from mortar-boxes after the passage of such vein material through the mill, that are frequently spoken of as "nuggets" occurring in the Rand conglomerates. Descriptions of similar vein deposits and of their contained minerals, in connection with the conglomerates, without any clear indication being given of their secondary and entirely different character have no doubt led many, who have not been able personally to examine the Rand deposits, to assume a much closer connection between such later mineral deposits and the original contents of the conglomerates than really exists, and as a consequence to regard the occasional association of the gold with typical vein minerals as evidence against its possible placer origin. The great age of the Rand conglomerates and the unlimited opportunities for modification and additions to the original deposits which have certainly been available, should, however, never be lost sight of in endeavouring to arrive at a conclusion with regard to their original nature and contents, and the prominence frequently given to such widely separated and comparatively rare occurrences as those just described must not be allowed to obscure the evidence afforded by the main mass of the conglomerates over practically the whole of the field. In this connection it is noteworthy that Young, who has made the most extended studies of the petrological and mineralogical characters of the conglomerates, although strongly inclined to regard the gold as of outside origin, was evidently impressed by the fact that its distribution is frequently similar to that in placer deposits. This, however, he attributes to the distribution of the gold and pyrite having been influenced by that of some heavy mineral of placer origin already present in the conglomerate.[19]

Especially interesting in connection with the possible modifications of an ancient placer deposit, including the re-crystallization of the gold and its association with pyrite, are the auriferous conglomerates of the Black Hills of Dakota first described by W. B. Devereux,[20] and afterwards examined and mapped by members of the United States Geological Survey, including S. F. Emmons and J. D. Irving.[21] We have thus the advantage of independent observations by a practical mining engineer and by experienced geologists specially connected with the examination of ore deposits, and the agreement shown on all essential points makes the data obtained from the study of the Black Hill deposits extremely valuable for purposes of comparison with the auriferous conglomerates of the Witwatersrand, with which they have many points in common. The description of the Black Hill conglomerates should be read by all interested in the question of the Rand gold. It may here be stated that these deposits form part of a series of sedimentary rocks, which, on account of their contained marine fossils, can be definitely referred to the Potsdam formation, which is of Cambrian age. The basal beds of the series are usually conglomerates which attain in places a thickness of about 100 ft., and which rest unconformably upon an ancient floor of schists (Algonkian). The schists are traversed by the famous Homestake vein, the source of the gold in the conglomerates. It should be noted, therefore, that the gold-bearing deposits are thus basal conglomerates, and in this respect differ from the principal gold-bearing conglomerates of the Rand, including those of

(19) *Trans. Geol. Soc. S.A.*, Vol. XII., 1909, p. 98.

(20) "The Occurrence of Gold in the Potsdam Formation, Black Hill, Dakota." *Trans. Am. Inst. Min. Eng.*, Vol. X., 1882, pp. 465-473.

(21) "Economic Resources of the Northern Black Hills," *U.S. Geol. Survey* Prof. Paper, No. 26, 1904.

the Main Reef Group which were deposited as part of a continuous conformable sequence of great thickness, and were composed of material, including the gold, which had travelled far from its source. In the Black Hills conglomerate, on the other hand, the gold was derived from a closely adjacent quartz-vein. Its coarser character and the occurrence of small nuggets, of which the largest recorded attained a weight of 3 dwt., are thus accounted for. The Black Hills conglomerate is cemented together by material which in the gold-bearing portions is largely pyrite or iron-oxide, while the non-gold-bearing portions are cemented by a quartzitic matrix. Irving and Devereux agree that besides gold which still retains its detrital character, there is a portion which has been dissolved and re-deposited, and this is found in a finely divided condition. The values obtained from the Black Hills conglomerate are interesting, and are of the same order as those found in many cases in the Rand reefs, while the occurrence of very rich gold immediately below large pebbles, recalls many similar occurrences met with in the Main Reef Leader. From the above-mentioned descriptions of the Black Hill conglomerates, certain points of great interest bearing on the Rand deposits come out very clearly, *e.g.* (1) by far the greater portion of the gold is undoubtedly of detrital origin, (2) there has been extensive solution and re-precipitation of the detrital gold, (3) the re-precipitated gold was always met with in a finely divided condition. The improbability of the solution and re-precipitation of gold in the Rand conglomerates and its finely divided condition, have been regarded as among the strongest arguments against the placer theory of its origin, yet here, in one of the only occurrences known which are of similar age, these same conditions are conspicuously displayed and most clearly established. A further noteworthy feature of the Black Hills conglomerate is that the gold is everywhere closely associated with pyrite, or with iron-oxides, derived from pyrite, and that where the gold is most abundant, so also pyrite or its derivitives is extremely plentiful. On the Rand where a similar association of large quantities of pyrite with rich gold is frequently observable, the deduction has been constantly drawn that the pyrite was intimately connected with the introduction of gold into the conglomerate. In the Black Hills example, the detrital origin of the gold is so clear that none of the observers who have studied it suggests that the pyrite has anything to do with the introduction of the greater portion of the gold.[22] Hence the tendency to attribute to pyrite an important part in the *introduction* of gold into the Rand conglomerates, because it is frequently found in large quantities in the richer areas is not supported by the very definite facts supplied by the Black Hills occurrence, where such a relationship is obviously absent.

Both Devereux and Irving regard the abundant iron minerals of the Black Hills conglomerate as having been introduced after the deposition of the rock. In reference both to these deposits and to those of the Rand, the question naturally arises as to why the close association of the richer gold with abundant ferruginous material should not be largely due to the same sedimentary causes as bring about a similar association in so many well-known placer deposits, without, however, necessarily excluding a subsequent introduction of additional pyritic material, which has certainly taken place, not only in connection with these gold-bearing beds, but also in very many other sedimentary rocks, where its presence has attracted less attention.

Summary.—It remains to summarize the main points dealt with in the preceding pages, and to consider briefly the ways in which the deductions which have been made bear upon economic questions connected with the conglomerate, particularly in the Eastern Rand, a district which in the near future, and for some time to come, appears destined to attract a very large share of the attention of mining men as being the most important goldfield at present awaiting development.

The following points appear to be worthy of special attention from those interested in the question of the origin of the Rand gold, and in the future of the field :—

(1.) Gold occurs much more widely in the Witwatersrand System than is frequently assumed to be the case, being found in practically all the well-marked conglomerates, which are distributed throughout the system as shown in figure 1.

(2.) Whereas certain of the conglomerates, like those of the Elsburg Group, conform more or less to the generally-held idea that the pebble-beds were originally laid down

(22) Irving considers that the introduction of a small but indeterminable quantity of gold with pyrite subsequently to the original deposition of the conglomerate seems probable but hardly susceptible of proof

as ordinary marine shore-deposits, there are other types including the most conspicuous gold-bearing beds such as the Main Reef Leader and South Reef, which are of a very different character, and which are similar to certain persistent conglomerates in the Lower Witwatersrand System, *e.g.*, the Government Reef. The characteristic features of these special types are their astonishing persistence as individual beds over very large areas, and in some cases the way in which they abruptly succeed a considerable series of fine-grained sediments.

(3.) Such characteristic features appear to be due to these particular pebble-beds having been laid down under conditions similar to those obtaining in an extensive deltaic area either upon its outer fringe or within the delta itself, and as a result of episodes of exceptional activity in sedimentation such as are known to have occurred in existing deltas.

(4.) The character of the whole succession of strata of the Lower Witwatersrand System whose deposition preceded that of the Main Reef Group is consistent with the explanation here suggested, there being evidence that the shore-line of the land surface from which the sediments were derived was at first at some considerable distance from the present locality of the Rand, but occupied successively nearer positions until, at the time of the deposition of the Elsburg beds, it was in close proximity to it.

(5.) The two most important gold-bearing conglomerate beds of the Rand—the Main Reef Leader and the South Reef—are characterized by the comparative uniformity of the material of which they are composed, and by the continuity of the whole deposit. These characters are such as would result from the rapid re-distribution of large quantities of gravelly material already graded and accumulated in the lower portions of a great continental river and in the upper portions of its delta. The pre-eminence of the Main Reef Group from a mining point of view is not exclusively due to an outstanding content of gold as compared, say, with the Bird or Kimberley Reefs, but to a large extent is a result of the degree of concentration of the gold within such beds as the Main Reef Leader. Had the Leader, after its deposition, been subjected to the action of waves and coastal currents the result would have been a deposit more like the Main Reef or like portions of the Kimberley and Bird Series in which the distribution of the gold is very similar to that found in many recent marine placers. The concentration of the gold in such beds as the Main Reef Leader and South Reef is closely connected with their sedimentary features, and is such as might be expected to result from the conditions of deposition, and there is a similar correspondence in other groups and in their individual pebble-bands, between the distribution of gold and sedimentary features.

(6.) The conditions found in the Far Eastern Rand, including the presence of only one principal reef and the possibility of following it in many different directions afford much better opportunities for the investigation of the problem of the origin of the gold than the more complicated conditions of the Central Rand. In the former area the close connection between the occurrence of gold and of conglomerate, and between the richer deposits and certain types of conglomerate, even when these occur in isolated patches entirely separated from the main body, is so universal and striking as to render it exceedingly difficult to conceive how such associations could be due to any other circumstance than a common cause. The continuity shown by the Main Reef Leader in the Far Eastern Rand and the occurrence of considerable quantities of gold in that particular bed over an area of hundreds of square miles which, in this wide extent of horizontal or but slightly inclined beds, is more than ever difficult to explain on the infiltration theory, receives a natural and adequate explanation when considered in connection with the stratigraphical features of the area and the probable mode of origin of the conglomerate bed.

(7.) Difficulties in the way of accepting a placer origin for the gold have arisen from confining attention largely to a few particular reefs which are exceptional in character as compared with a much larger number of associated conglomerates, and from making comparisons between these and stream-placers or ordinary shore-deposits with which they have as little in common in sedimentary features as in regard to their gold contents. When a comparison is made between the more ordinary members of the Rand conglomerates and such deposits as those of Nome, a very great similarity is apparent and the exceptional characters of such auriferous beds as the Main Reef Leader are not surprising now that fuller data concerning its stratigaphical relationships and the conditions leading up to its deposition are available.

(8.) The solution and re-precipitation of the gold which among those who regard it as of

detrital origin is generally accepted as a step in the evolution of the conglomerates as we now find them has been shown to have similarly happened in the Black Hills occurrence, one of the only known auriferous conglomerates of an age comparable to that of the Rand deposits. In this case, as well as in the secondary quartz-veins of the New Goch Mine, and also in the example given from Bourbonne-les-Bains, a very notable feature is the re-deposition of the gold and other minerals in the immediate neighbourhood of the spot in which they had been originally deposited.

It may further be noted that the features presented by the Rand conglomerates with regard to the distribution of the gold are no more exceptional than are the sedimentary features of the conglomerates themselves when compared with other known occurrences, and further, that whereas the evidence in favour of the gold being an original constituent of the conglomerates is *direct* and becomes more and more complete with the fuller investigation of the sedimentary characters of the formation, the arguments for an extraneous origin are highly speculative and are frequently of such a nature that though apparently satisfactory when applied to one portion of the field, they are entirely inadequate to explain the conditions met with elsewhere. The theoretical and highly speculative nature of much of the evidence adduced in favour of the infiltration hypothesis will be noticeable to anyone examining the fullest and most recent exposition of it by Horwood, and the fact that it is thought necessary to bring in so many subsidiary lines of argument indicates the absence of such direct evidence as would be considered in itself of a sufficiently convincing character.

The question of the nature of the association of gold with the Rand conglomerates is, of course, of the very greatest importance, both from the point of view of the student of ore-deposits and of those interested in economic and practical mining matters. Whether the conglomerates are to be regarded as "fossil-placers" or as "lodes" has much to do not only with the future possibilities of the Rand but also with that of conglomerate occurrences in other localities within this country and in other parts of the world.

To my mind, after an extensive study of the whole formation, the evidence in favour of the former view is convincing, and is increasing continually with the extension of our opportunities for the collection of information. The even balance in the weight of evidence in favour of opposing theories which obtained a few years ago has been materially disturbed by the data now available from the Far East Rand, and from a more extended study of the whole Witwatersrand System, and it seems likely that we must be prepared to admit a new and interesting variety into the family of known types of placer deposits.

If the Rand conglomerates are really placer deposits then *the question as to whether mineral lodes decrease in value in depth has no application to them, since they belong to an entirely different class of occurrences.* The character and value of the conglomerates will be related to sedimentary causes, and though they may be subject to the usual variations of sedimentary deposits, they are not likely to exhibit the pronounced vagaries common in mineral veins, and moreover, as our knowledge of the district increases, we shall be able to predict with a less degree of uncertainty future possibilities.

The much debated question of the possible impoverishment of the Rand ores in depth is one beset with many difficulties. It is almost impossible to give due weight to all the considerations which enter into that problem, or even to obtain the data regarding the earlier operations in this goldfield, which are necessary for a solution. In particular, it will never be possible to correctly estimate the extent to which highly selective mining was practised in the "early days" or to calculate the influence on subsequent results which is undoubtedly felt even to-day. That the question of impoverishment with depth has remained so long in doubt shows, however, that the evidence in either direction is not of a very decisive character. Whatever the final results may be, it will probably be fully recognised at some later time that such diminution in value as very possibly occurs is not at all a function of *depth as such* but one of direction and of distance relatively to a certain source and depends upon circumstances connected with the flow not of subterranean mineralising solutions, but of surface streams.

In connection with this matter, Rickard refers with ridicule[23] to the expectation of Rand engineers that profitable banket would extend to 18,000 feet on the dip. It is only fair to assume that this attitude was adopted without knowledge of the weighty information we possess with regard to such a chain of mines as the Van Ryn, Kleinfon-

tein, Van Ryn Deep, Brakpan, and Springs. Here we have already the essentials of the expectations referred to more than realised.

Further, from a geological point of view, the Nigel has practically the same claim as Modderfontein or Kleinfontein to be regarded as the outcrop of the Springs reef, and with very slight changes in the geological history of the district, the Nigel might very easily have been the *only* outcrop of the whole of the reef mined on the Far East Rand. This is an aspect of the matter which is frequently overlooked and one which suggests a number of very interesting speculations well worth consideration by those interested in the question under discussion.

In connecting the possible extension of gold-bearing conglomerates below the Witwatersrand basin wtih the destruction of auriferous quartz-veins *in the same area*[24], Gregory, appears to attribute to conglomerates like the Main Reef Leader and South Reef, relationships similar to those of a basal conglomerate. Such considerations might apply to a deposit at or near the base of the system like that sometimes present below the Orange Grove Quartzites. (See Fig. 1.) When, however, it is remembered that many thousands of feet of conformable beds separate the Main Reef Group from the older rocks at the base of the Witwatersrand System it will be seen that the character and distribution of the quartz veins in these rocks anywhere below the present basin of the Rand can have little to do with the contents of the conglomerates at the Main Reef horizon.

In a field like the Rand, where there exist enormous quantities of auriferous material, whose payability or unpayability depends upon slight reductions in the cost of mining or of treatment, the question of the extent to which natural causes contribute to the cheapness of working is a very interesting one. In this connection, the degree to which the gold distributed through certain beds has been concentrated either laterally or vertically, is noteworthy. In the Central Rand the comparative degree of separation or approximation of the different reefs is often a very important factor, as, for example, where two thin leaders are near enough together to be mined in the same stope, or where a thin but comparatively rich reef, as the Main Reef Leader may be in places, which cannot itself be profitably mined, lies sufficiently close to such thicker but poorer bodies as the Main Reef or the Bastard usually are, to enable the combination to be advantageously dealt with. In the far Eastern Rand, where usually a single reef has alone to be considered, the question of the *lateral* distribution of the gold becomes of greater importance, and it is a fortunate feature, throughout the greater portion of the area with which we are acquainted, that the conglomerate of the principal reef shows a marked tendency to concentration laterally into well-defined patches as described above. The corresponding concentration of the gold distributed over any particular area, which has so far been found to be usual, will considerably simplify the work of the miner in the Far East Rand. Another fortunate circumstance, probably only fully appreciated by those directly concerned in actual mining, is the presence of the slate footwall. Exploratory work directed to finding reefs, temporarily or permanently lost by faulting or other causes, has probably formed a much bigger item in the cost of mining in the Central Rand than is generally recognised, and might form an interesting, if not exhilarating, subject of research in statistics. In the Far East Rand, with a flat dip and a single reef, absent altogether in places, such exploratory work might have been a still more formidable item were it not for the unusually good "marker" afforded by the slate footwall in that area—another advantage of the close relationship everywhere existing between the distribution of gold and sedimentary features in the Far East Rand. In the ealier stages of exploration and development on the West Rand the scanty and hurried attention bestowed on geological considerations resulted in useless expenditure, the extent of which has probably never been even remotely realized. The East may possibly be more fortunate.

Of an abundance of points in connection with the Rand conglomerates, which are of the greatest interest both from a purely scientific and from a practical mining point of view, no mention has been made in the present paper, which has nevertheless extended beyond the length originally intended. In connection with some of these points, a large amount of information has already been collected, others remain for investigation. As stated at the outset, no attempt has been made to deal with the formidable body of literature which has already accumulated around the subject of the Rand conglomerates. The necessity for limiting

(23) "The Persistence of Ore in Depth." *Bull.* 122, *I.M.M.*, Nov., 1914, p. 10.

(24) J. W. Gregory. *loc. cit.*, p. 38.

consideration to the line of investigation which has fallen more especially within the scope of recent survey work also prevents adequate reference to the extensive researches which have been carried out by previous writers in connection with aspects of the conglomerates not specially discussed in the present paper.

The President: In proposing a very hearty vote of thanks to Dr. Mellor for his exceedingly valuable paper, I wish to congratulate the Society on having secured the reading of this paper before it. The paper is being read here and also before the Institution of Mining and Metallurgy, thanks to the working agreement we have with that Institution.

I also take the opportunity of welcoming the members of the Geological Society here to-night, and hope they will contribute to the discussion on this paper.

No doubt Dr. Mellor has had exceptional opportunities of investigating the subject but, at the same time, I think there is not the slightest doubt that he has brought very conspicuous ability to bear on the subject. Once more, I have much pleasure in proposing a very hearty vote of thanks to Dr. Mellor.

Prof. G. H. Stanley *(Past-President):* I am sure it is a very long time indeed since the Society has had a paper of such value; one which will certainly become a classic. I must say, speaking merely as an amateur geologist, that I have always been unable to accept the infiltration theory. There seemed to be so many points about the occurrence of the gold that could not be satisfactorily explained in that manner. That theory seems to be supported rather by examinations of microscopic slides than by consideration of the geologist's field as a whole. Dr. Mellor has undoubtedly tackled the problem in the right way, and I think his conclusions are sound they may be challenged, but I do not think they will be successfully challenged.

I have much pleasure in seconding the vote of thanks

Mr. E. M. Weston *(Member of Council):* I have much pleasure in supporting the vote of thanks to Dr. Mellor for his lucid and comprehensive paper. Later on I hope to have something else to say on this subject. I rose to-night to show the members a large pebble from the oxidized stopes of the Wemmer mine, which shows gold in the quartz of the pebble itself. This was one of several obtained by a friend of mine from the mine long ago. It has been urged against the placer theory that such gold-bearing reef quartz pebbles do not occur. It will be noted that the gold is not leaf or paint gold and is evidently original.

Mr. D. Wilkinson *(Member):* There is one subject which Dr. Mellor has not referred to in his paper. As you are all aware, the Main Reef Series from Randfontein to Springs is intersected by a most extraordinary reticulation of dikes, and, so far as I know, there is not a mine in which dikes have not been intersected. There has been a great deal of discussion as to the sources from which these dikes were injected into the strata. The influence of these dikes and their apparent influence upon the value of the reefs near the intersection is often quoted as evidence in support of the infiltration theory.

Dr. Mellor has had the greatest opportunity of any man on these fields of observing and studying these dikes in detail. I am curious to know whether he is of the opinion that the strata of the Lower Witwatersrand Series have been intersected to the same extent by dikes as the Main Reef Series. I am well aware that it is not an easy matter to always trace the occurrence of a dike on the surface, and, hence, I am only asking Dr. Mellor for his general impressions on this subject.

Dr. E. T. Mellor: First of all, I would like to thank the meeting for the way in which my paper has been received. I also wish to do myself and my subject a little service, in that, while I have so many mining men before me, I would ask that when you meet with occurrences which you consider are extraordinary or interesting, you will be kind enough to draw my attention to them.

I have visited, of course, a great many of the mines, but you know what a tremendous task it is to examine even a very small proportion of the existing workings.

I have been fortunate in having always the extremely willing and generous help of the mine officials in directing my attention to what they considered interesting, and also in going to considerable trouble to enable me to get, with the least expenditure of time and energy, at those portions of the mine which I suggested would be most interesting to me; and, without that very cordial co-operation I should not have been able to get so far with the task in hand because, as you all realise, it is a very big undertaking for a single observer.

With regard to Mr. D. Wilkinson's question, I have in a portion of my paper, which time did not allow of my reading to-night, dealt, though not at very great length, with the question of dikes, and you will notice that one of the diagrams before you has special reference to that subject.

In answer to the first portion of Mr. Wilkinson's question, I will only say briefly that although there are undoubtedly many instances in which local concentration and segregation of gold has taken place owing to the agency of dikes — some of the most striking occurrences of coarse gold being probably connected with such a cause — I have not yet met any clear case of dikes having influenced the main distribution of the gold.

With regard to the second portion of Mr. Wilkinson's question, as to whether many dikes occur in the Lower Witwatersrand system, it must be remembered that although shown to be so numerous in mine workings in the upper part of the system, they are comparatively rarely met with at the surface as they usually weather much more readily than the quartzites which they traverse. They are even less likely to be seen at the surface in the Lower Witwatersrand System, as their weathered outcrops would be very inconspicuous among the ferruginous slaty rocks which form a large part of the section. In underground workings they would probably be equally numerous if not more so than in the Upper Witwatersrand System. In the case of a few very large dikes which intersect both the upper and lower portions of the system no connection with the occurrence of gold can be made out.

The President: I have brought the four diamonds found on the Simmer Deep to show Dr. Mellor, and, perhaps, the other members of the Geological Society would like to examine them.

I will not ask any further discussion on this paper to-night, as we have other items to get through, and the hour is getting late.

CYANIDE CONSUMPTION ON THE WITWATERSRAND.

By H. A. White (Vice-President).

THE PREVENTION OF HYDROLYSIS IN CYANIDE SOLUTIONS.

By H. M. Leslie (Member).

(Printed in Journal, September, 1915.)

DISCUSSION.

Mr. Thos. B. Stevens *(Corresponding Member of Council):* At the present time any paper which deals with the economy of imported stores is of great importance to all metallurgists working within the British Empire, and it may be helpful to compare the figures for cyanide consumption on the Rand given by Mr. H. A. White, with those obtained under somewhat similar conditions, but with a different method of treatment in a Western Australian plant.

In the plant referred to the ore treated is similar to Rand ore, in that it contains 2·5% of pyrite, and no other minerals which are usually regarded as causing losses of cyanide, the assay value is slightly higher. The method of treatment is by grinding in cyanide solution to pass a 150-mesh screen, agitation by means of air in 0·04% KCN solution for eight hours, and separation of the gold bearing solution by a Cassel type vacuum filter in which separate washes of spent solution and water are given. No spent solution is run to waste with the residue with the exception of that remaining as moisture in the filter cakes. Precipitation of the gold from the solution is by means of zinc shavings. For the past three months the average consumpton of sodium cyanide (129 %) has been 0·38 lb., and the consumption of zinc has been 0·23 lb., as compared with 0·4 lb. of cyanide (of unstated strength, although presumably sodium), and 0·32 lb. of zinc, which is stated by Mr. White in his opening paragraph as being the average for the Rand. In the Western Australian plant the cyanide discharged in the residue water in the form of cyanides, ferro-cyanides and sulpho-cyanides amounts to 0·16 lb. of sodium cyanide per ton of dry slime, as compared with 0·25, 0·15 and 0·25 lb. given by Mr. White in Table A for the loss in slime treatment at Simmer Deep, Rose Deep and Geduld.

Only a rough comparison is possible, but it does not appear that as far as cyanide consumption is concerned the one method of treatment could be more suitable than the other. Mr. White's figures deal only with the decantation process of slime treatment, and I assume that where Butters plants have been installed the ore continues to be

crushed in water and collected by settlement so that the loss of cyanide remains the same. Am I right in this assumption? I should also like to ask Mr. White if, apart from all other considerations but that of cyanide consumption, he considers that the milling of Rand ore in cyanide solution with subsequent vacuum filtration of the slime would reduce the cyanide consumption. The advocates of crushing in cyanide solution years ago made a great point of the saving of cyanide effected, but my own experience does not confirm this, as apart from the dissolving of the gold I have found that in order to obtain good precipitation on the zinc a much higher cyanide strength is required than when the solutions are frequently renewed by dilution as when crushing in water; in fact I have visited many plants where a high cyanide strength has been in use solely for the purpose of obtaining good precipitation. Mr. White states that a saving of cyanide could be effected, but at the cost of counterbalancing gold loss Might I ask if in this instance the gold loss would be due to less gold being dissolved, or less gold being precipitated from the solutions, or both; that is to say, would the extra loss be in dissolved or undissolved gold?

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Potash from Kelp. "An exhaustive examination of the kelps found on the coast of the United States has given the following average figures for the various kinds: *Macrocystis pyrifera*, K_2O, 12·59%; N, 1·57%; I, 0·23%... *Nereocystis luetkeana*, K_2O, 20·1%; N, 1·9%; I, 0·13%... *Maria fistulosa*, K_2O, 9·1%; N, 2·6%; I, trace. The available beds of the Pacific coast and the annual harvest of kelp, with the resultant quantities of potassium chloride, are estimated as follows:—

Region.	Area sq. miles	Fresh Kelp.	KCl.
		Tons.	Tons.
Cedros Island to San Diego ...	91·4	16,979,800	649,000
San Diego to Point Conception ...	97·9	18,195,300	696,000
Point Conception to Cape Flattery	36·2	4,377,400	167,000
Puget Sound ...	5·0	520,000	20,000
South-East Alaska	14·6	15,666,000	598,000
Western Alaska...	17·9	3,367,000	136,000
Total ...	390·0	59,105,500	2,266,000

The amount of potassium chloride that it is possible to produce thus is five times the total import of potash salts (calculated to chloride) from Germany. It is thought that the cost of handling, drying, grinding, storing, and loading, at San Diego and neighbourhood, should not much exceed $1 per ton of dried kelp; cutting and collecting is estimated at $1.83 and general expenses at $1, giving a total cost of $3.83. The total fertilizer value of dry kelp is calculated to be $15.75 per ton, of which $9 represents the potash value. The margin of profit thus shown should stimulate the utilisation of the large local resources. It is suggested that it may be possible later to replace the plants in some of the beds with varieties, such as *Pelagophycus porra* (egg kelp), which contain higher percentages of potash; this variety is found in scattered groups along the coast of Lower California."—F. K. Cameron, *U.S. Commerce Rept.*, No. 143, June 19, 1915.—*Journ. Soc. Chem. Industry*, August 31, 1915. p. 869. (J. A. W.)

Potash Deposits of Spain.—"Numerous borings have been made in the region where the potash salts abound in the provinces of Barcelona and Lerida. The results have been particularly favourable, but it will require the employment of capital and enterprise to make the potash from these deposits a profitable product. In the various borings near the town of Sarria, potash salts were found at depths between 121 and 197 feet and others at 420 ft. At 886 ft., the greatest depth attained, important quantities of potassium compounds were found to rest on a stratum of salt not yet pierced. In the area tested by borings, comprising some 2,690,000 sq. ft., there is approximately 2,550,000 tons of carnallite and 1,150,000 tons of sylvite.

From the general characteristics of the region it is considered probable that there are further deposits in the neighbourhood of those already tested. In a stream running by the salt works of Cardona there is a large percentage of potash in solution, and vast quantities of potash have already been allowed to go to waste in the salt mines that might have been profitably used. On account of the exceptionally irregular geological formation of this particular part of the country near Barcelona, it is difficult to make exact valuations of the amount of potash salt that can be mined."—U.S. Cons. Rept., July, 1915.- *Journ. Soc. Chem. Industry*, August 31, 1915, p. 869. (J. A. W.)

METALLURGY.

The Effect of Mineralised Waters in Gold Cyanide Plants.—"At the Sons of Gwalia mine (W. Australia), the best practical conditions for using the local water supply were ascertained. The 'fresh water' was obtained from shallow wells and the 'salt water' from the drainage of underground workings. The salt water contained $CaSO_4$ 3·584, $MgCl_2$ 6·317, $MgSO_4$ 2·760 and NaCl 12·972 gm. per litre. Until recently salt water only was used in the plant, the sand from the battery being treated with 0·3 and the slime with 0·07% cyanide solution. The use of protective alkali was not economically possible, as 12 lb. of pure lime was required per ton of salt water; but by adding 2 lb. per ton of ore the cyanide consumption was reduced owing to the protective action of magnesium hydroxide. The addition of a small quantity of fresh water caused the deposition of considerable magnesium hydroxide and calcium sulphate on the zinc shavings, due to the dilution of the sodium

chloride. Experiments made with fresh water proved that it was far more satisfactory, and that the bicarbonates consumed more cyanide than the other cyanicides present: this was counteracted by adding a sufficiency of lime."—T. B. Stevens and W. S. Bradley.—*Inst. Min. and Met.*, April 15, 1915.—*Journ. Soc. Chem. Ind.*, May 15, 1915, p. 495.—(J. A. W.)

The Milling of Vanadium Ore.—"At the plant of the Primos Chemical Company, Vanadium, Colo., the ore containing from one to three per cent. vanadium is crushed with rock salt and iron sulphide (1,500 lb. ore, 90 lb. rock salt and 30 lb. iron sulphide constitute a 'charge'). The material is crushed to pass a 30 mesh screen and roasted in McDougal and Wedge oil fired furnaces. The vanadium is thus obtain as sodium vanadate. The iron sulphide helps in keeping the required temperature in the roasters.

The roasted ore is cooled on conveyor belts, and discharged into a launder carrying a weak solution of sodium vanadate: the mixture being carried to the leaching vats of a capacity of 100 tons each. The strong solution draining off from the tank which is being filled is pumped to storage tanks ready for precipitation. Weak solution obtained by leaching the material in the filled vats with hot water from the roasters is pumped to the launder to meet the one from the roasters. The strong solution carries from 6 to 12 gm. vanadium per litre, while the weak solution, which is diluted when necessary carries from 1·5 to 5 gm. per litre. The time for leaching a vat is about 36 hours, the vat being discharged when the solution draining away contains less than 0·1 gm. vanadium per litre. The test is as follows:—To 25 cc. solution 5 cc. conc. sulphuric acid and then 5 cc. hydrogen peroxide are added: 0·01 gm. vanadium per litre or more gives a deep cherry red, while approximately 0·1 gm. per litre gives a light straw yellow.

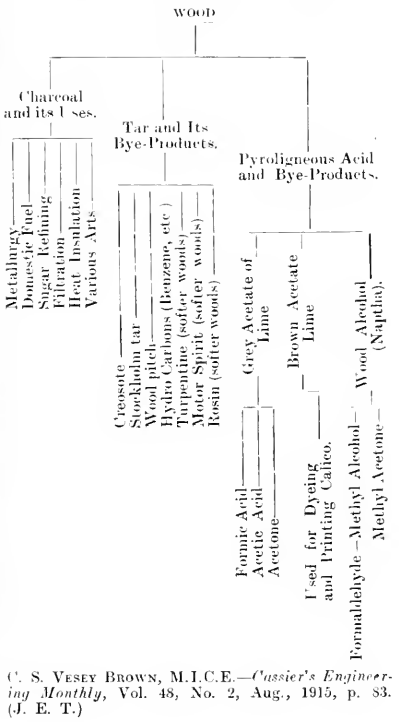
The solution in the storage vats is cooled by means of air jets since the most complete precipitation is obtained from cold solutions. From the storage vats the solution is transferred to the precipitation vats of 50 tons capacity each. The precipitating reagent is a solution of iron sulphate (one ton iron sulphate in 2·5 to 4 tons water. In practice ·5 to 2·7 lb. iron sulphate are required to precipitate 1 lb. vanadium. During the addition of the iron sulphate the contents of the vat are agitated by means of air. After precipitation the mixture is agitated for 4 to 6 hours to prevent the precipitation of lime. The precipitate is then collected in filter presses, where it is washed. It is then transferred to a Ruggles-Coles dryer, from which it is discharged to be bagged and shipped to the refinery."—A. F. Hallet, *Colorado School of Mines Magazine*, July, 1915 (Vol. 5), No. 7), p. 131.—(H. R. A.)

Treatment of Arsenical-Antimonial Gold Ore.—"From sulphide ore containing Fe 8·3, Sb 0·21, As 0·96, S 5·62, MgO 2·74, Al_2O_3 2·02, CaO 3·36%, and gold to the value of 53s. per ton (the remainder being insoluble matter, water, etc.), the maximum extraction obtained by ordinary methods (amalgamation, cyanidation, concentration, etc.) was about 50%. In the method adopted, the finely-divided (practically 150-mesh) ore is roasted sufficiently to liberate the gold from the sulphides, then mixed with oxidized ore, and the mixture agitated with a solution of sodium cyanide and filtered under reduced pressure. The ore not being dead-roasted, lead acetate is added to the solution to remove soluble sulphides, and the gold is finally precipitated by means of zinc dust. The average extraction exceeds 84%, the consumption of sodium cyanide being about 1 lb., and the total cost 9s. per ton. The loss of gold during roasting rarely exceeds 3%."—K. B. Moor and H. R. Edmands, *J. Chamber of Mines*, W. Australia, Feb., 1915.—*Journ. Soc. Chem. Ind.*, Sept. 15, 1915, p. 908. (J. A. W.)

MISCELLANEOUS.

Wood Charcoal.—"In this article the author considers the production of wood charcoal by destructive distillation and illustrates various forms of retorts. He particularly draws attention to the products of distillation from various woods, and considers that land with timber at present looked on as "waste" land might easily be turned into a source of considerable revenue. The uses to which the products are put are shown in the following table:—

Tabulated List of Products of Distillation.



C. S. Vesey Brown, M.I.C.E.—*Cassier's Engineering Monthly*, Vol. 48, No. 2, Aug., 1915, p. 83. (J. E. T.)

PREVENTION OF POROSITY OF CEMENT. "An effective treatment by which the porosity of cement used in constructional work can be stopped and the free lime neutralized is said to be the painting of the surface with a solution of ¾ lb. of zinc sulphate ($ZnSO_4 7 H_2O$) in a gallon of water, reports *Engineering News*. A reaction between the zinc sulphate and the free lime occurs as deeply as the solution penetrates, and by it the insoluble neutral salts, calcium sulphate and zinc hydroxide are precipitated into the pores. This priming coat should be given some 96 hours to dry, the surface then being brushed and painted with two coats of a good cement paint."—*Eng. and Min. Journal*, April 24, 1915, p. 733. (G. H. S.)

CHINESE ANTIMONY.—"War demand for antimony has increased the price of Chinese antimony ores from $170 to $700 per ton, according to Huan-Hi Liang, a member of the Chinese Honorary Commission now in this country. Prior to the war the company of which he is president was smelting about 3000 tons of antimony per year, but decided expansion has recently taken place, a combination having been formed with the small producers of crude antimony. About 10,000 persons are now employed in the antimony industry in China." *Mining and Engineering World*, July 3, 1915, p. 183. (H. R. A.)

WATTLE BARK IN NATAL. "A wattle bark extract factory is to be erected at Merebank, Natal, in which the green bark will be subjected to a series of mechanical pressings by means of heavy bronze rollers, hot water and tan liquor washes being applied to the pressed bark between each set of rollers. The extraction is effected within two minutes, and no darkening in colour is possible as occurs in the process of leaching in pits.

The amount of the bark exported from Natal in 1914 was 58,132 tons, valued at £286,399, as compared with 65,042 tons, valued at £309,268, in 1913." *Board of Trade Journ.*, June 10, 1915 *Journ. Soc. Chem. Ind.*, 15th July, 1915, p. 726. (J. W.)

SULPHUR-SAND CEMENT. "Sulphur-sand cement is composed of 1 part sulphur and 7½ parts quartz sand ground to pass at least a 60 mesh screen. The mixture is heated to about 150° C. when it flows nicely and is sufficiently above the melting point of sulphur, 111° C., to prevent sudden chilling. The fact that sulphur begins to thicken above 156° C. and becomes so viscid that it will not flow at 186° C. must be borne in mind or there will be difficulty in working the cement. This is possibly the most satisfactory general cement available for low temperature work. It is readily handled and remarkably strong, has a tenacious bond, is free from cracking and inert to most solutions. It will be found valuable for large scale work, as well as in the laboratory. I believe it was developed by Charles S. Bradley for use in his copper leaching process; at any rate, it was learned of through him." E. J. HALL. *Eng. and Min. Journal*, June 19, 1915, p. 1071. (G. H. S.)

Abstracts of Patent Applications.

(C.) 66/15. Wilhelm Mauss. Improvements in dressing ores and the like. 17.3.15.

This application, which is not accompanied by drawings, claims the separation of the specifically lighter colloid from heavier crystalline particles in ore or slime pulp "by force of the nature of gravity," such as the centrifugal force employed in the separators described in the applicant's previous Patents Nos. 76/13, 150/14 and 369/14.

(C.) 69/15. Harold Guthridge Lovell. Improvements in "damp blasts" or water spraying devices for use in mines. 20.3.15.

This application has reference to a damp blast or water spraying device for use in mines, the general principle employed being in the nature of an ordinary atomiser whereby a stream of air, impinging across the jet of water, atomises the jet, and at the same time has a suction effect on the water supply.

(C.) 305/15. Charles John Williams (1), The Willfley Company, Limited (2). Apparatus for sampling streams of wet ores and the like. 15.12.15.

This application relates to an automatic pulp sampler. The apparatus consists of a pivoted water vat counter-balancing a sampling scoop, which, on its descent, cuts the pulp stream at right angles and receives and diverts the entire pulp flow over a period of time regulated by the speed of water discharge from the vat. While the scoop is held in the stream, the sample is diverted into a special box and channel fitted underneath the feed launder, whence it flows to a collecting box.

(C.) 307/15. James Kerr Fulton. Improvements in and in connection with hose clamps or means for securing hose and other similar pipes on their connections, couplings, and the like. 17.12.15.

This application relates to a hose clamp consisting of a loop shaped wire frame to act as a clamp, and with a special tool for straining the wire round the hose and turning over the end to fasten same.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

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ROSENDORFF, R., *l/o* Johannesburg; Cardozo Hotel, Lourenco Marques.

SHARPLEY, H., *to* Piggs Peak Development Co., Ltd., Piggs Peak, via Barberton.

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UNIVERSITY OF TORONTO

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Vol. XVI. MARCH 1916. No. 8.

Proceedings
AT
Ordinary General Meeting,
March 18, 1916.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, March 18th, 1916, Mr. J. E. Thomas (President) in the chair. There were also present:—

22 Members:—Mr. H. A. White, Prof. J. A. Wilkinson, Messrs. H. S. Meyer, C. Toombs, F. Wartenweiler, F. W. Watson, John Watson, E. M. Weston, A. Whitby, Prof. G. H. Stanley, J. R. Williams (Members of Council), R. A. Cooper, H. S. Kenney, A. King, S. Nettleton, S. Newton, E. A. Osterloh, W. S. V. Price, C. B. Simpson, A. L. Spoor, A. Thomas and J. H. Veasey.

12 Associates and Students:—Messrs. J. Davidson, O. A. Gerber, J. Gibson, W. Human, W. J. R. Hunter, J. Ingram, R. W. Irwin, D. W. Pugh, H. Rusden, T. L. Thorne, H. Ward and J. A. Woodburn

6 Visitors and Fred Rowland (Secretary).

A DEVICE FOR REGULATING THE UNDERFLOW OF CLASSIFICATION CONES.

Mr. E. H. Johnson *(Past-President):* The accompanying drawing illustrates a modification of the "Dowsett" patent for this purpose as applied to classification cones of the diaphragm type. The original, as designed by Mr. Dowsett, consisted of a wooden box projecting at right angles to the vertical axis of the cone, and was actuated by the sudden fluid pressure acting on a canvas diaphragm which lifts a cone valve closing the orifice. The re-opening is effected by a fixed weight against which the diaphragm has to operate.

Experience with this form showed uncertainty of action owing to the dewatered sand having to travel horizontally for a short distance and also to the diaphragm having to operate against a fixed weight.

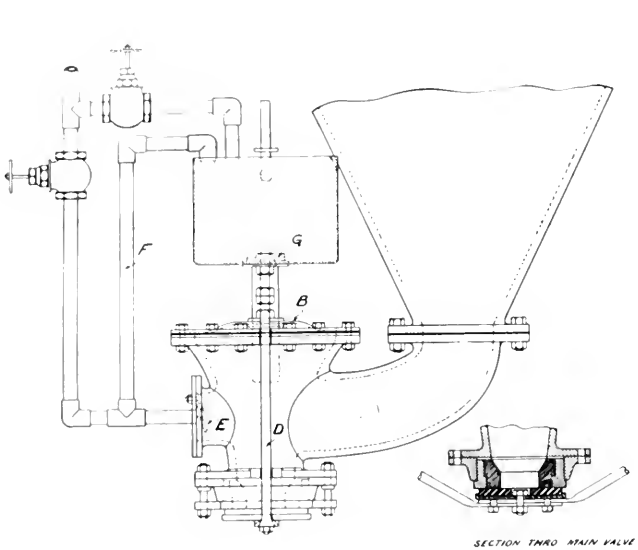
To overcome this uncertainty of action a cast-iron chamber was designed enabling the sand to maintain a course of easy flow and for the fixed weight a small vat was substituted, which automatically filled with water when the valve was closed, and emptied itself when the valve opened, thereby giving a maximum weight for the purpose of opening and a minimum resistance to the operation of the diaphragm when closing.

The results obtained with these modifications have fully justified the change, and an automatic cut-off has been evolved which is thoroughly reliable in action and has brought down supervision and labour to a minimum while maintaining a classification of the greatest uniformity. The moisture content of the underflow varies between 28 and 29 in a slime free sand, while the slime overflow, producing 55 of the total pulp as slime, never exceeds 2 of +200 mesh product. The sand grading averages 66% −90 of which approximately 22% is −200 mesh.

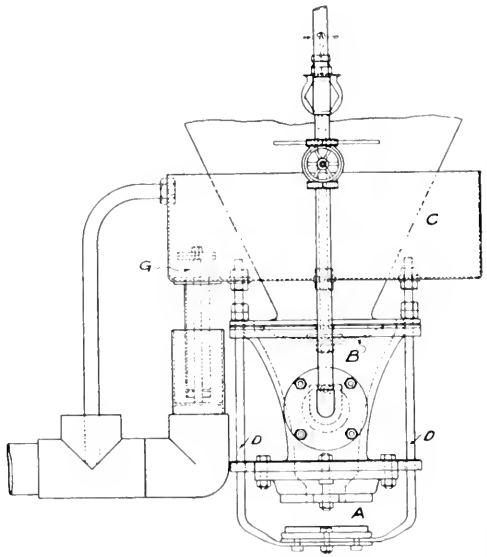
The action of the canvas diaphragm appears to depend on the relatively smaller area of the nozzle outflow compared with the larger opening in the apex of the cone causing a sudden upward distention of the diaphragm when a "break-away" occurs. When the thickened sand re-collects in the apex of the cone the pressure on the diaphragm decreases and the weight of the vat of water causes the valve to re-open.

These modifications are due to Mr. Lloyd, Cyanide Manager of the Cason Section of the East Rand Proprietary Mines, and his mechanical assistant.

The President: In thanking Mr. E. H. Johnson for his interesting description of the modification of the "Dowsett" regulator, I



VIEW SHEWING MAIN VALVE SHUT



VIEW SHEWING MAIN VALVE OPEN

Automatic Regulator for De-watering Cones, East Rand Proprietary Mines, Ltd., 3 in. 1 ft. Under normal conditions the pulp passes through the main valve "A." When a break occurs in the cone the sudden water pressure lifts the canvas diaphragm "B" together with the vat "C" and the stirrup "D" thus closing the valve "A" and the valve "E" which diverts the feedwater into vat "C" through pipe "F" the pulp settling in the cone meanwhile. The vat "C" fills with water, the valve "G" being closed. When full the vat "C" falls depressing the diaphragm "B" and opening the main valve "A," the last part of this fall opens waste valve "G" and empties vat; the valve "E" opens and admits a steady flow of water into the chamber, until the next break occurs.

might remark that all these improvements tend towards the attainment of our ideal, viz., efficiency. It would, I think, add to the value of this contribution if Mr. Johnson would give us some idea of the cost of installing these improved regulators.

A NOTE ON TUBE-MILL LINERS.

Mr. A. J. Herald *(Member):* In the reply to his paper on "The Use of Scoop Discharges in Tube-Mills,"* Mr. W. R. Dowling included the results obtained from a tube-mill lining made from shoe shank blocks. It was also mentioned that due to erosion the concrete of that liner became too weak to support the weight of the shoe shanks, and that to overcome this weakness another liner of shoe shank blocks had been installed with a mild steel bar five inches deep and three-quarters of an inch thick between each row of shank blocks. The latter was installed in the outlet section (one third of the total length) of a tube-mill; it has been in constant use for ten months, and in all probability will last for at least six months more.

Since much of the wear of tube-mill liners is undoubtedly due to slip of the pebble load, a liner designed to prevent this slipping and at the same time possessing the other necessary qualities of a good tube-mill lining would probably give highly satisfactory results. With this object in view it has been decided to line a section of a tube-mill with white cast iron segments, each segment having a projecting rib. The segments will be 22 inches long, 11 inches wide, and one inch thick; the rib will project three and a half inches from the centre line and along the full length of the segment; it will be two inches thick at the base and two and a half inches thick at the top, the taper of the rib being all on that side against which the pebbles will be lifted. When in position, the projecting ribs of this liner will therefore be nine inches apart, and it is expected that they will have a much better gripping and lifting action on the pebble load than the ordinary tapered bar liner, and that consequently the power consumption will be slightly higher, but on account of the fewer joints and crevices less amalgam will be locked up.

The President: This is another of those excellent little notes that we are only too grateful to get, and we would be grateful for any further contributions of the same sort. In that connection, I might say that, at the last meeting, I mentioned something about welded cam shafts. Mr. Brazier, of the City Deep, rang me up the other day and said that they had adopted a similar procedure for some time, and found it was answering very well, and that they were running at a cost of 0·71s. per day.

*See this *Journal*, Vol. XVI., October, 1915, p. 72.

NOTES ON RARE MINERALS IN MADAGASCAR.

By T. P. Waites.

I. Uranium, niobium, and tantalum minerals.

The centre of Madagascar, in the districts of Antsirabe and Betafo, about one hundred miles south-west of Tananarive, is extraordinarily rich in uranium and niobium minerals, which occur in pegmatite.

Very little, however, was known of the occurrences till the end of 1911, when the district was visited by M. Lacroix of the Académie des Sciences de Paris.

On one claim (2,500 hectares) is found blomstrandite, betafite, samiresite, ampangabeite, brown garnet, monazite, euxenite, amazonite, columbite, tantalite, bismuth, uranite, copper, beryl, amethyst, orangeite, sphene and oxide of manganese. Betafite, samiresite, and ampangabeite (related to pyrochlore, variety hatchettolite), are minerals, which, prior to 1912 were unknown to science, and their average uranium oxide content is, approximately, twenty-two per cent. (see analyses attached). It is only during the past few months, say, only during this year, that any market has been found for these uranium ores. The method of treatment for pitchblende, and for the extraction of its radium content was known, but it was not certain that the same methods would apply in the case of these new minerals. Recently it has been found possible to treat them, and I venture to predict that the district will become the premier producer of uranium, and enable radium to be obtained at a price which will render it much more freely available for medical purposes than it has been hitherto.

Betafite, samiresite, ampangabeite differ from each other in their chemical consti-

tuents, as also they differ from blomstrandite (see analyses attached). They are named from the places at, or near which, they were first found, viz., Betafo, Samiresy, Ampangabe.

The districts around Betafo and Antsirabe are decidedly volcanic. There is no active volcano here, nor in any other part of Madagascar, and authorities differ much in the hypothetical estimation of the geological age of the volcanic activity; but there are craters made up of scoria or volcanic ash, horseshoe shaped, lopsided and broken on the south-east, whence comes the trade wind prevailing for nine months out of the twelve. Others are of lava, and have flooded the country in parts for miles around. Here and there are patches of columnar basalt. There are also many volcanic necks, rounded, fez-shaped, or conical; and east, west, north and south, are hot springs and cool "mineral water" springs, side by side, whose waters also contain uranium salts, and have, from time immemorial, been resorted to by "natives," on account of their medicinal qualities, without the natives knowing the why or wherefore.

The Vavavato range consists of hornblende gneiss. This range has the appearance of having been tilted up by volcanic action all along its eastern side. Gneiss is certainly the prevailing rock in the archaean region, *viz.*, two-thirds of the area of the island.

To turn to the pegmatites, I find that they are composed of orthoclase felspar (a mineral which gives the whole range of Vavavato a pinkish colour), quartz, hornblende, (the hornblende is sometimes very much decomposed, and of a waxy nature in places, white or streaked with red) and what is, locally and generally, termed biotite, or black mica, but is really uranite. The black-brown garnety mineral which carries only four per cent. of uranium oxide, is a negligible quantity commercially, not being rich enough in uranium. Sometimes on breaking up large pieces of this, beautiful betafite crystals are found within.

I have tried to touch lightly on the salient points of this interesting district, its niobotantalotitanates, its pegmatites, its geological formation. All the constituents of the pegmatite are radio-active and uraniferous. Even the felspar carries one per cent. uranium oxide. Further prospecting has shown that uranium ores occur over a very considerable portion of the archaean region, and not in the pegmatite only. Uranium is found from Cape St. Marie, the southern point of the island, to as far north as Ankazobe, about 50 miles north-west of Tananarive, and from the Indian Ocean to the Bongolava range. The niobates are found not only in the form of crystals, but also as veins in the pegmatite, and as veins in the schists and gneisses, the outcrops being traceable for many kilometres. One of them persists for 70 kilometres, and one sample taken from it assayed 2·87 U_3O_8. Parallel with it is another vein carrying thorium, two assays showing 7·8 and 2·78 ThO_2. The analysis of these minerals is difficult, but my experience has been that the uranium content of ores, as shown by assay, and by the electroscope, in expert hands, is nearly identical.

The following are analyses of four different uraniferous crystals:—

	Blomstrandite.	Betafite.	Samiresite.	Ampatogabeite.
Nb_2O_5 ...	23·30	34·80	45·80	34·80
Ta_2O_5 ...	28·50	tr.	3·70	8·90
TiO_2 ...	10·87	18·30	6·70	4·90
SnO_2 ..	0·30	0·30	0·10	—
ThO_2 ...	—	1·30	—	2·50
UO_3 ...	18·10	26·60	21·20	19·40
Bi_2O_3 .	0·49	—	—	—
Al_2O_3 ...	—	2·10	0·74	2·10
Fe_2O_3 ...	—	2·87	—	8·60
FeO ...	1·35	—	1·06	—
(Ce, La, Di)$_2O_3$	2·50	0·60	0·20	0·60
(Y, Er)$_2O_3$ ...	0·30	0·90	—	4·00
MnO ..	0·50	—	—	—
MgO ...	0·20	0·40	—	—
CaO ...	4·00	3·45	—	1·50
PbO ...		—	7·35	—
K_2O ..	—	—	0·30	—
Loss on ignition	9·60	7·60	12·45	12·40
	99·92	99·22	99·60	99·70

These analyses are from Paris. Prof. Lacroix "On the uraniferous niobotantalotitanates (radio-active) of the pegmatites of Madagascar." Séance de l'Académie des Sciences, 22 April, 1912.

II. The Uranium phosphate of Vinaninkarenena near Antsirabe.

The phosphate of uranium is exposed by a deep cutting made by a river flowing south through the lacustrine deposits, about ten kilometres south of Antsirabe. The uranium salts thus exposed have evidently been precipitated on a thin layer of (tourbe) peat, decomposed vegetation of some sort, favourable to the deposition and crystallisation of

these salts, which salts are not found throughout the "clay" but in one thin layer only on the decomposed vegetation. The whole of this lacustrine deposit on the old lake bed at Antsirabe contains this uranium phosphate. As this deposit is about ten miles distant from the mineral springs of to-day, I think it is probably not deposited from the mineral springs, but is a product the result of oxidation, disintegration, and the action of water on the uranium ores in the surrounding hills.

In the Infanja marsh, north of Itasy, are mineral springs, hot and cold; north of this marsh, in an area where uranium oxide is found, is a cold spring of mineral water carrying uranium salts in solution. This spring gushes through the gneiss, not with a constant flow like the mineral springs at Antsirabe, but pulsating, and deposits its uranium in a manner I have not found elsewhere in the island. I put my hand as far as I could into the spring, and found the "pipe" of the spring filled with small rounded stones, in appearance, shape, and colour, like hard peas. These "pea-stones" are evidently accretions of uranium from solution, around small specks of sand or other foreign matter, which have been blown into the spring, in rings of brown, lemon and black colour, the shape being due to the beating or pulsating action of the spring, and the friction of stone against stone, giving all of them a spherical shape, the colour, inside and out, being identical with the uraniferous ores found elsewhere.

I do not give this example as proof that the carnotite of Utah, and the autunite of Antsirabe, are *not* deposits from uranium salts in solution in the waters of mineral springs, but it is proof that uranium in solution does not necessarily crystallise as the bright green, or canary yellow, of the carnotite and autunite. Where the uranium phosphate is found, there are no mineral springs nearer than ten kilometres—there may have been in some past geological age, but there is no evidence in the shape of calcareous sinter deposits such as one sees at Antsirabe, and from which Antsirabe takes its name. These "pea-stones" (they are not the aragonite of the Carlsbad springs) are radio-active and uraniferous.

The district around Intanja marsh is volcanic, probably of comparatively recent date—in any case more recent than the volcanic activity which produced the patches of columnar basalt near Betafo.

Uranium phosphate is found also in the district of Analalava, in the north-west of the island, near to the sea.

The President: I beg to propose a hearty vote of thanks to Mr. Waites for his interesting paper. I trust that any of our members who have been to Madagascar will raise some discussion on this paper and so add to our knowledge of the country. It is also possible that members may know of non-members who have been to Madagascar and are qualified to speak on the subject, and who might care to join in the discussion.

Dr. Moir has sent his apologies for not being able to be present this evening, and Mr. Toombs will read his papers for him.

ANALYSIS OF NIOBIUM-TITANIUM MINERALS, WITH SOME NEW TESTS FOR NIOBIUM, TANTALUM AND TITANIUM.

By James Moir, M.A., D.Sc.

The following modification is an improvement on the known process for separating the constituents of euxenite, aeschynite, pyrochlore, and similar minerals, namely, those which contain titanium in addition to niobium and tantalum.

The agated specimen is fused with caustic potash (not soda), and the melt dissolved in twenty parts of water and immediately filtered through a hardened paper in a Buchner funnel under reduced pressure, finally washing out the insoluble matter with warm dilute potassium hydrate solution. By this method practically all the titanium remains behind "salted out" as K_2TiO_3 along with the true insolubles (the hydrates of iron and manganese, and the yttrium and didymium earths), whereas in Simpson's process (Geological Survey Bulletin, Western Australia, 1906) the solution is acidified, thus bringing the titanium into solution, in which case according to my experience it cannot be separated from niobium—at least within 5 or 10 of the true value. The use of caustic *potash* in the fusion is essen-

tial since *sodium* titanate does not "salt out" satisfactorily and tends to redissolve on attempting to wash it. In addition, sodium *tantalate* is so sparingly soluble compared with potassium tantalate that tantalum may be overlooked from being left in the "insolubles." The alkaline filtrate may contain silicate, tantalate, niobate, stannate and manganate as potassium salts. For identification of Nb and Ta it is then treated with strong *sodium* hydrate solution, when *sodium* tantalate if present crystallises out in needles in the course of a minute, and on decanting the supernatant liquid and allowing to stand for several hours, sodium niobate crystallises out in hexagonal plates. This method does not profess to be quantitative, but it affords a remarkably decisive identification of these troublesome elements.

Reactions of Niobic Acid.—Solution of sodium niobate when treated with acid not in excess and warmed is precipitated as hydrated niobic acid, but it rapidly mixed with excess of cold HCl gives a clear solution, and it is only with such a clear solution that the confirmatory tests for niobium can be got. The precipitated niobic acid is also fairly soluble in hot strong hydrochloric acid (concentrated acid plus half its bulk of water), and this solution gives most of the subjoined tests. Curiously enough $HNbO_3$ is much less soluble in concentrated HCl than in the above mixture.

The following are some of the known tests for niobium, including both the ancient and the modern: the clear solution of $HNbO_3$ in HCl is used.

1. *Tannin Test.*—Pure niobic acid gives a reddish chocolate precipitate, whilst pure tantalic acid gives a sulphur-yellow precipitate, the usual colour with crude niobates containing tantalum is russet-brown.

2. *Ferrocyanide Test.*—Complete absence of iron is naturally necessary. $HNbO_3$ gives a red-brown precipitate (like that of uranyl salts). $HTaO_3$ gives nothing in the cold and a yellow precipitate on warming.

3. *Nascent-hydrogen Test.*—Addition of zinc to the acid solution of $HNbO_3$ gives a brown colour if the acid is strong and the action vigorous; if the acid is weak an intermediate olive coloration is observed. If both titanium and niobium are present the colour is indigo blue, probably a combination of the violet of titanium with the olive colour of niobium. If tungsten is present a deep indigo-coloured *precipitate* (not coloration) is produced.

4. *Meinbach's Tin Test.*—The solution of $HNbO_3$ in *strong* (2:1) HCl is boiled with tin *for some time* when a deep sapphire-blue coloration is obtained, fading on standing and regenerated by boiling. This is the most sensitive test for niobium. Note the remarkable difference in the behaviour of zinc and tin in niobium solutions.

5. *Pennington's Sulphocyanide Test.*—An alkaline niobate solution treated first with sulphocyanide and then acidified gives a yellow coloration: then on adding zinc, the solution becomes greenish and finally red-brown (J. Amer. Ch. Soc., 1896, p.51). This also is a sensitive and characteristic test.

The following are some new reactions observed by the writer.

1) $NaNbO_3$ treated with ammonium oxalate and then with acid and zinc gives a deep-brown coloration, not altered by hydrogen peroxide or by sulphocyanide. On prolonged reduction the colour fades to yellow. Titanium solutions treated in this way give a dull yellow colour changing to orange-red with hydrogen peroxide.

2) $NaNbO_3$ treated with Rochelle-salt or other tartrate and then with acid and zinc gives a permanent brown solution.

3) Potassium hydrosulphite (from zinc dust and potassium metabisulphite) added to an acid solution of niobic acid gives a curdy yellow precipitate, oxidising on standing to white niobic acid. Titanium solutions with this reagent give a red-orange colour very similar to that given by the H_2O_2 test for Ti. The violet colour of trivalent Ti could not be observed as an intermediate stage.

4) Pyrogallol (and other polyhydroxybenzenes and allied substances such as gallic acid) give an orange precipitate with niobic acid, but mineral acid must not be present in excess; this is analogous to the known tannin test, but the colour is more characteristic and less liable to be confused with that from titanium. Tantalum gives pale yellow precipitates with these reagents.

5. *Reactions of Niobium and Tantalum in Concentrated Sulphuric Acid.*—If a few milligrams of $NaNbO_3$ or $NaTaO_3$ are added to concentrated sulphuric acid, and the latter gently *boiled* for a minute, a clear solution is obtained. Tantalic acid dissolves comparatively slowly and generally leaves a faint opalescence. The solution on cooling is treated with various phenolic bodies, and gives characteristic colour-reactions in the case of niobium, but only yellow colours in the case

of tantalum. The test is analogous to the thymol test for titanium and gives a somewhat similar range of colours (Lenher and Crawford, J. Am. Ch. Soc., 1913, p. 138, and Report of 1912 International Congress). Niobium may be confirmed *in presence of titanium* by subsequently diluting with four parts of water and adding zinc, when a further series of reduction-colours is given by niobium, whereas titanium gives a barely visible violet coloration, and tantalum gives no reduction-colour. Thus thymol, hydroquinone, pyrogallol and pyrocatechol give blood-red colorations with niobium in concentrated H_2SO_4, which on dilution and treatment with zinc give inky-greenish reduction colours, whilst phenol at first gives an orange, which on dilution and reduction gives a dirty-violet, and morphine gives a pink going yellowish-green on reduction.

Tests for Titanium.—The most sensitive is Lenher and Crawford's, mentioned above, viz., a solution of thymol in sulphuric acid added to the titanium compound dissolved either in concentrated H_2SO_4, or by fusion with $KHSO_4$ diluting with concentrated H_2SO_4. The coloration is garnet-red and the test is 20 times as sensitive as the hydrogen-peroxide test.

The writer has noticed that the violet reduction-colour of titanium is much intensified by adding sulphocyanide after reducing with zinc and hydrochloric acid. Under these circumstances the colour is as intense as that given by the H_2O_2 test, and is not given by any other metal.

SOME NEW METHODS OF TESTING FOR MOLYBDENUM.

By James Moir, M.A., D.Sc.

It is well-known that a fugitive deep-blue coloration is obtained as the first stage in the reduction of molybdic acid by nascent hydrogen. Chapman and Law (*The Analyst*, 1907, p. 251), by using platinised zinc with a sulphuric acid solution of MoO_3 succeeded in obtaining this coloration in a comparatively stable form, and by titration with permanganate showed that it corresponds to Mo_2O_5 (or possibly Mo_3O_8). On further reduction the solution passes through green and olive to chocolate-brown, when the titration shows that the state of oxidation is Mo_2O_3. This chocolate coloration is the one ordinarily obtained when zinc and fairly strong acid are employed, whereas tungsten under these circumstances always gives blue precipitates, not colorations. From the intensity of the deep-blue coloration the author expected that a specially-sensitive test for molybdenum could be devised if conditions could be hit on under which any too extensive reduction to Mo_2O_3 could be prevented. For example if the molybdic acid is present in a solution only faintly acid with *mineral* acid, and this is treated with a *few drops* only of highly dilute stannous chloride, a deep-blue coloration of some stability is obtained; too much acid or too much $SnCl_2$ causes the test to fail, as the brown colour of full reduction then obtained is not sensitive. The use of acetic acid instead of mineral acid in this test is of no advantage, as in this case the addition of dilute $SnCl_2$ produces only a yellow turbidity turning brown, and not at all sensitive.

Hydrazine, N_2H_4, forms in the writer's experience the best reagent for developing the blue colour. A solution containing a trace of alkali molybdate when acidified with acetic acid and treated with a little hydrazine sulphate and boiled, rapidly turns deep-blue and retains this colour on boiling. Hydroquinone may be used in place of hydrazine with much the same result. An analogous but more remarkable reaction, is that obtained when a slightly acid solution of MoO_3 is treated with potassium iodide (in some excess) and boiled for some time: iodine is slowly liberated and the solution turns blue. The use of *phenyl*hydrazine for reducing molybdenum has been described by Spiegel and Maas (*Berichte*, 1903, p. 513), but the reddish coloration obtained is different and seems to contain phenylhydrazine. Simple hydrazine is an improvement.

The best known sensitive test for molybdenum is that in which the acid solution is treated with sulphocyanide and a tiny piece of zinc added, when a crimson coloration is obtained in a few seconds. If iron is also present the solution becomes blood-red on adding the sulphocyanide, but on adding the

zinc this becomes colourless through reduction (to the ferrous condition) and after a few seconds becomes crimson if Mo is present. If the solution is strongly acid the coloration verges to red and is less sensitive; if nearly neutral the coloration resembles that of permanganate. It is probably due to $Mo(SCN)_3$. Another modification of the test for Mo in presence of Fe is to add stannous chloride to the acid solution until the yellow colour just disappears and then add sulphocyanide.

Another well-known reaction for MoO_3 in the absence of iron consists in adding potassium ferrocyanide. *Mineral* acid must be present, and a russet-brown precipitate is obtained which still contains hexavalent molybdenum.

In the presence of acetic acid (not mineral acid) tannin (or gallic acid) gives a similar reaction, which has been known for some time. The writer has improved this by substituting pyrogallol or pyrocatechol for tannin. Either of these when added to a molybdic acid solution previously treated with sodium acetate gives a very sensitive orange coloration.

The facts contained in these papers were communicated to the S.A. Association of Analytical Chemists in May, 1915.

The President:—These two papers of Dr. Moir's are most interesting. Perhaps some of our chemical friends may have something to say about them.

CYANIDE CONSUMPTION ON THE WITWATERSRAND.

By H. A. WHITE (Vice-President).

(Printed in Journal, September, 1915.)

REPLY TO DISCUSSION.

Mr. H. A. White *(Vice-President)*: I have to thank those gentlemen who took part in the discussion for the kind remarks made, and to express gratification that the opportunity for putting new experimental results on record was made use of. Dr. Caldecott's correction of the clerical error made puts the value of cyanide used yearly in South Africa at half a million pounds, and, no doubt, this will be an inducement to the establishment of a local industry for its production.

With regard to the suggested experiments on the reaction responsible for the strong zinc sulphide coating on our zinc shavings, I may say that repeated and various experiments show that decomposition of sulphocyanide even in presence of strong cyanide or alkaline solutions and a vigorous evolution of hydrogen cannot be demonstrated. A warning may be given that the zinc now being imported itself contains a faint trace of sulphide. The presence of this well-known sulphide coating can however be traced with fair certainty to the presence of thiosulphates in our solutions, though the reactions involved are slow and incomplete.

In response to the invitation given, I would express as a result of the available evidence my conviction that the present cyanide consumption can be allocated as follows:—

	Per cent
Loss by dilution	30
Loss by cyanicides	25
Loss as HCN gas	25
Loss as NH_3 gas and N compounds	20

A more detailed analysis of loss is given in Tables A and B, on p. 28, of the original paper.

Mr. H. M. Leslie is correct in emphasizing the cyanide regenerated in experiments G. 16 and G. 17, as this would make the closed test slightly better than the open test with adequate alkalinity, but against this must be set the large scale test at Modder B, where only a trace of HCN could be regenerated.

Mr. John Watson's results are the more valuable as he has given the method he uses for determining protective alkali, and has stated the fact that the tests were made in the laboratory. His results show far smaller loss than Mr. Leslie's (due to presence of zinc) and much higher than those from Geduld, where however more natural conditions were maintained. I would urge that all future experiments of this nature should observe the following conditions:

(1) Works solutions should be used as a basis.

(2) Exposure should be in the open air over a cyanide sump.

(3) Alkalinity should be determined without the addition of ferrocyanide.

Under these conditions the losses of cyanide shown will, in most cases, be very much less than the usual reported laboratory result.

Mr. A. Whitby gives some very interesting results on the effect of cyanide and alkali concentrations on the consumption of

zinc. The fact that this side issue was not investigated (or rather reported) does not reflect on the adequacy of the "tests made on behalf of the Chamber of Mines" as he supposes. I can, however, only briefly indicate why the whole of this admirable contribution is beside the mark aimed at. Not only is the presence of zinc quite without influence on the determination of total cyanide, but any increased or decreased action of the solutions on the zinc shaving is in practice met by varying the number of compartments used in the zinc extractors so that for the same value of solution the amount of zinc consumed by direct action (and inevitable production of nascent hydrogen, the effective agent) would vary very slightly in spite of considerable modification in temperature, alkalinity, cyanide strength, etc. Mr. Whitby asks "Why, when lime deposits pursue us from launder to sump should we call it soda?" The answer is obvious if we recollect that our figures do not refer to deposits but to solutions, *e.g.*,

$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4$$
$$\text{and } Ca(HO)_2 + Na_2CO_3 = CaCO_3 + 2NaHO$$
(deposit) (solution)

Mr. Thos. B. Stevens' communication is very welcome and helpful, and the replies to his questions are as follows:—

The figures given refer principally to cyanide of sodium of which two kinds were in use. One contained about 120 , and the other about 130 KCN, which latter is now in sole use.

The ore here is still crushed in water in all cases but one (Meyer and Charlton). I do not think that the milling of Rand ores in cyanide solution would in actual practice materially reduce the cyanide consumption, as we have to offset the "dilution loss" by the increased cyanicide loss due to the absence of the preliminary alkaline wash treatment which the milling in water practically involves.

The counterbalancing loss in gold experienced when special trials were made with weaker solutions was almost entirely due to ineffective precipitation in the zinc extractors and was thus in the form of dissolved gold. If the cyanide had been cut down very much further still no doubt the point where decreased solution of gold comes in would have been reached:—in present practice that can only happen as a rare and unfortunate accident.

THE CONGLOMERATES OF THE WITWATERSRAND.

By E. T. Mellor, D.Sc. (Lond.), M.Inst.M.M.

(*Printed in Journal, February, 1916.*)

DISCUSSION.

The President: The Secretary will read an addition to Dr. Mellor's paper, dealing with the occurrence of diamonds on the Witwatersrand, and referring to the diamonds exhibited by myself at a previous meeting. Dr. Mellor has asked me to express his regret that he is unable to be present to-night.

Dr. E. T. Mellor.—*Diamonds from the Simmer Deep.*—The four specimens of diamonds from the banket at the Simmer Deep exhibited by Mr. J. E. Thomas are a very interesting addition to the recorded occurrences of diamonds in the Main Reef conglomerates. These already include specimens from the Wellhuter and the Modderfontein B. Mines[1]. Further examples of diamonds from the conglomerates of the Witwatersrand system are the numerous specimens obtained from the Gold Estates Reef at Klerksdorp.[2]

The four stones exhibited by Mr. Thomas weigh together approximately 1 carat. They are all practically complete crystals. Two are of a pale yellowish green, a third has a somewhat more decided greenish colour, especially in certain lights, while the fourth has only a very faint tinge of yellowish green. All the stones are perfectly clear and transparent. In all, the original sharpness of the edges and the markings on the faces have been considerably modified by wear. As these particular specimens have passed though a number of machines as described by Mr. Thomas the smoothing of the stones might be attributed to these agencies. It is, however, much more probable that the wear is the result of natural processes gone through prior to the deposition of the stones on the reef, as specimens from other localities which have been taken direct from the mortar boxes have been subjected to comparatively little attrition after being freed from the rock yet show similar evidences of wear.

[1] Young. "Notes on Diamonds in the Banket." *Trans. Geol. Soc. S.A.*, Vol. XVI., 1913, pp. 39-40.
[2] Denny. "The Klerksdorp Goldfields." London, 1907, pp. 15, 97-98.

In the Summer Deep specimens a prominent angle of one of the stones has suffered a recent blow which was probably a result of its passage through the mill. In this case the edges formed by numerous cleavages are perfectly sharp and do not seem to have been modified at all by the grinding to which they were almost certainly subjected after the fracture.

In March, 1913, Prof. Young recorded the finding of a single specimen at the Modder B mine. On my visit to the mine two years later I was shown a collection of over a hundred stones. In colours these vary, from practically colourless specimens through pale yellowish green and olive green to dark bottle green. One specimen looked quite black and was opaque. Except in some few cases the stones are complete crystals. They show unmistakable signs of wear and there can be no doubt as to their alluvial origin.

At Kleiksdorp, in 1910, I was shown three small stones from a much larger number found in the Gold Estates Reef. These also had the same waterworn appearance as the Rand stones. Two were almost black in appearance, but showed a dark dull green coloration when held up to the light. The smallest was of a clear light yellowish green.

The similarity of coloration shown by all the stones so far found in the conglomerates suggests that this feature may possibly have some connection with their history after leaving their original source.

The occurrence of diamonds, iridosmine, and other rare minerals with the conglomerates appears to strengthen considerably the arguments for the alluvial origin of the gold. It shows that the currents which laid down the conglomerates of the Main Reef zone did actually carry such heavy minerals as are likely to travel in company with alluvial gold. More careful search will probably show that such minerals are of much commoner occurrence than is yet known to be the case, but the evidence already available shows that they have a fairly wide distribution. All these points seem to support the conclusions already drawn from the character and mode of distribution of the more ordinary constituents of the conglomerates, viz., that they were derived from a distant source, had undergone an extended process of selection and concentration, and owe their present distribution to the action of a comparatively uniform current of wide extent.

CHIEF SOURCES OF ACCIDENTS IN THE WITWATERSRAND MINES.

By Chas. E. Hutton, M.Inst.M.M.,
Inspector of Mines (Member).

(*Printed in Journal, November, 1915.*)

DISCUSSION.

Mr W. S. V. Price (*Member*): In reading this paper one notices many things that read like improbabilities, but the fact of their having happened proves that they are not as rare as they should be. Mr. Hutton has called his paper the "Chief Sources of Accidents in Witwatersrand Mines." I have waited for a long time to see if anyone would mention what, to my mind, is one of the chief sources of accidents in these mines. I refer to the almost absolute lack of maintenance of discipline among the natives by their overseers. We know that "authority without responsibility" is a bad state of affairs, but responsibility without authority under existing conditions underground now, is just as bad, and, I am convinced, a fertile source of accidents. I read in last Month's *Reef* that the Ginsberg mine bewailed its misfortune at losing its good position in the "Safety First" competition through a Kaffir going where he had been told not to go, or doing what he had been told not to do, and thereby losing his life. Some years ago I was spoken to by the manager of the mine I was working on owing to some of the boys complaining of my mode of dealing with them. I told him that when boys came to my section they were not there for a rest cure, but to do as much as I could get out of them. The manager agreed, but said that in case of trouble I should report them to the Compound Manager or some such person. This did not appeal to me at all; I considered that the Compound Manager was their "boss" when they were in the compound, but when they were in my section I was boss and they had to do as I told them, or I should have to apply some punishment. I may appear keen on underground discipline, but it may be understood why when I say that I was a ship's officer for a good many years. A good officer on the deep water sailing ship never tells any man to do anything twice. He demands, and gets, absolute and implicit obedience. Lack of discipline is fatal on board a vessel, and many cases of vessels being lost might be traced to it. I contend

that a miner's position is very much like that of a ship's officer. Miners are on their own resources with a gang of from six to 40 or even 50 people under them, and thanks to the system we have in vogue, unless he is prepared to take the law in his own hands, he would have a rough time of it. Of course, it is common knowledge that lots of men are not fit to have authority; they have never had charge of anything in their lives before, and they do not know how to handle men of any colour. All the same, I hold that the man in charge (who is responsible) should have more authority vested in him; as if a boy knows that he will get punished by his boss if he does wrong, it will have a greater deterrent effect on him than some probable visionary accident or future punishment by another person who takes the control from his boss.

The meeting then terminated.

Contributions and Correspondence.

A SIMPLE METHOD OF CALCULATING THE RELATIVE OR PERCENTAGE HUMIDITY IN MINES FROM DRY- AND WET- BULB READINGS.

In a recent number of *The Coal Age* there appears an ingenious formula for calculating humidities of the air from the ordinary hygrometer readings: this is:

$$100 - H = \frac{460}{d}(d - w)^{0.84}$$

in which d and w are the dry- and wet- bulb readings (Fahrenheit) and H is the percentage humidity. This formula requires logarithms, and is for sea-level, so it struck me that a simpler one could be devised to suit the underground conditions of the mines, where neither the temperature nor the humidity varies much.

The following formula is for 27 in. barometer, or about 3,000 ft. above sea-level (but 1,000 ft more or less makes little difference) and holds true for all readings likely to occur in the mines.

$$100 - H = (d - w)\left(\frac{165 - d}{20}\right)$$

Thus, if d = 75 and w = 73, $100 - H = 2 \times 4{\cdot}5$, and H = 91,

and if d = 95 and w = 92, $100 - H = 3 \times 3{\cdot}5$, H = 89·5.

These both agree very closely with the *data* in the Smithsonian tables; therefore the formula must be correct for all dry-bulb readings between 75° and 95° F., especially since the wet-bulb cannot be read with sufficient accuracy to give H within ½.

JAMES MOIR.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ACTION OF NITRIC ACID ON ALUMINIUM.—"Plates of commercial aluminium of 99·7% measuring 100×28×1 mm. and weighing 7 gm. to 8 gm., were kept in closed glass bottles containing nitric acid of various concentrations and at various temperatures. About 9–10 of each plate was immersed in the acid, the rest being exposed to the fumes. After 15 days the plates were carefully brushed, washed, and weighed, and immersed for a second period of 90 days. The accompanying table shows the loss in weight of the plates under

Specific Gravity of Acid.	Temperature Deg. C.	Percent Loss of Weight		Loss of Weight in 15 Days. Gm. per sq. m. surface.
		In 15 Days.	In 105 Days.	
1·52	0	0·54	1·03	15
1·52	5	0·92	1·57	25
1·52	18	1·42	2·58	38
1·52	28	4·27	6·54	116
1·39	8	8·76	16·70	237
1·39	28	31·10	58·50	845
1·13	18	11·78	22·40	320
1·13	28	46·10	82·60	1,250

different conditions. In a second series of experiments the bottles were left open, exposed to the air; the corrosion was considerably greater than in the first series, partly on account of absorption of moisture by the acid. The plates were most deeply corroded at the surface of the liquid. Attempts to protect the metal by coating it with celluloid varnish, cellulose acetate, caoutchouc, and paraffin were unsuccessful; but by heating it for about an hour at 400° C. to 500° C. its resistance to attack by nitric acid was much increased, though this treatment is not recommended for thin aluminium vessels as it impairs the mechanical strength of the metal. Welded aluminium was found to be as resistant to nitric acid as the untreated metal. The rate at which aluminium is attacked by nitric acid is increased by the presence of copper, sodium, or magnesium in the metal, or chlorine or hydrochloric acid in the acid. The author concludes that nitric acid can be safely transported in aluminium vessels provided that the metal is of high purity and the acid is at its maximum concentration (cp. Fr. Pat. 451,812 of 1912; this J., 1913, 699,) but great care is necessary in using aluminium vessels for the preparation of nitrocellulose, as the rate at which they are attacked is much accelerated by hydration of the acid and rise of temperature."—A. TRILLAT, *Bull. Soc. d'Encour.*, 1915, *142*, 547—554; *Journ. Soc. Chem. Ind.*, August 31, 1915, p. 874. (J. A. W.)

DETERMINATION OF TRACES OF CARBON MONOXIDE.—"The author claims priority over Graham and Winmill (this. J. 1914, 944) for the determination of the conditions required for the iodometric esti-

mation of carbon monoxide in the air. The reduction of iodic anhydride by carbon monoxide begins at 30° C. to 35° C., and even at dilutions of 1 in 200,000 and more, oxidation to carbon dioxide is complete at 75° C. If the carbon monoxide is accompanied by a considerable excess (10–20 vols.) of ethylene, acetylene, or hydrogen, it is, however, only partially oxidized, and recourse must be had to other methods of estimation, or the gas may be diluted; at dilutions of 1 in 100,000, the hydrocarbons do not react sensibly with iodic anhydride. The iodine, liberated according to the equation $5CO + I_2O_5 = 5CO_2 + I_2$, may be estimated conveniently by direct weighing, after sublimation on to copper turnings heated to 80° C., or colorimetrically by passing the vapour into carbon bisulphide and comparing with a standard tint. Sulphur dioxide or hydrogen sulphide must be previously removed from the air before it comes into contact with the iodic anhydride by passing through potassium hydroxide solution and over soda-lime."—A. Gautier, *Bull. Soc. Chim.*, 1915, *17*, 256–260. — *Journ. Soc. Chem. Ind.*, Sept. 15, 1915, p. 918. (J. A. W.)

Determination of Lead as Sulphite.—"Lead can be precipitated quantitatively from slightly acid solutions by adding an excess of sodium or ammonium bisulphite or of sulphurous acid, provided that excessive acidity be avoided by addition of ammonia. As an example, 22 cc. of a solution of lead acetate containing 7·832 gm. Pb and 10 gm. of free acetic acid per litre was diluted to about 100 cc. and treated with an excess of 2% sodium bisulphite solution, the mixture being well stirred and allowed to settle for not more than an hour. The precipitate was collected in a Gooch crucible and washed with cold water without allowing the whole of the liquid to drain until the final washing. After drying at 150° C. the lead sulphite weighed 0·239 gm., corresponding to 0·1723 gm. Pb; error, 0·066% gm. The lead was also precipitated with sulphurous acid containing 36·8 gm. SO_2 per litre. The addition of sodium acetate is recommended in this case, and a large excess of sulphurous acid must be avoided. The method was successfully applied to mixtures of lead with copper and zinc; it is inadmissible in presence of the alkaline earth metals or of metals which are reduced by sulphites (see also this J., 1914, 435). — G. S. Jamieson, *Journ. J. Sci.*, 1915, *39*, 157–160. — *Journ. Soc. Chem. Ind.*, Sept. 15, 1915, p. 925. (J. A. W.)

Corrosion due to Occurrence of Chlorine in Coal. "Rapid corrosion of the tubes of an economiser was found to be due to chlorine in the furnace gases, the tubes containing much ferric chloride. The coals used had been derived from the brine districts of the Midlands, and the whole of the chlorine (0·018% to 0·34%) in six samples was present as soluble chloride. In the case of another boiler furnace in which nothing but Welsh coal had been used, there were no signs of pitting in the economiser tubes after three years. Eleven samples of the different coals used in this and neighbouring boilers contained 0·003% to 0·032% Cl." "A Dr Waele, Analyst, 1915, *40*, 146, 156. — *Journ. Soc. Chem. Ind.*, May, 15, 1915, p. 481. (J. A. W.)

METALLURGY.

Cottrell Process of Dust Collection. "Electrical precipitation is the term used to describe the removal of suspended solid or liquid particles from gases by means of a strong electric field maintained between collecting electrodes and discharge electrodes. The process was made commercial by F. G. Cottrell a few years ago, and successful applications since then have been several described.

In the electrical precipitation process, direct currents of exceedingly high voltages are used, say up to 100,000 volts. Since corrosive substances are frequently handled and high temperatures encountered, the maintaining of insulation and connections has been a serious problem in practice. Another difficulty has been the removal of precipitated material. In adapting the method to various purposes a great number of incidental problems have been encountered and solved by the Corporation's engineers.

The great field of application of the process has been in metallurgical and allied industries, where it is used to recover by-products, to eliminate nuisances, and even for refining. Among these metallurgical applications may be mentioned the recovery of the valuable constituents in fumes from silver refinery gases; the saving of hydrated tin chloride from the gases of a detinning plant; the removal of chlorine from the gases of a chemical works by adding hydrated lime dust to the gas stream and collecting the resulting bleaching powder electrically; the treatment of fumes, largely zinc oxide, from crucibles producing zinc-copper alloys; the recovery of sulphuric acid mist; the treatment of copper converter gases, with a recovery of lead; the removal of valuable metals and acids from copper roasting furnace gases; the fractional precipitation out of the gases of a copper-smelting arsenic plant, first of solid particles and then of pure arsenic after cooling, thus saving refining treatment; the removal of volatilized hydrochloric acid from a wire company's exit gases, the cleansing of sulphurous gases from a zinc-sulphide roaster; the cleansing of iron blast furnace gases; and the recovery of values from the waste metals plant of a lead company.

Possibly of more general interest are the fields of application outside of metallurgy. The elimination of carbon particles from furnace smoke of any kind has been found entirely feasible and offers an advantage over the use of mechanical stokers, as removing also the particles of ash. Dust in cement manufacture can be controlled with electrical precipitation. Illuminating gas can be entirely detarred. A particularly interesting application is the collection of powdered milk, sugar, eggs, malt, grape fruit juice, etc., in a plant installed for the Milk Flour Company. These powders are thrown down from a hot gas consisting of evaporated solutions, emulsions, etc., from which the water has been removed by heating. Excellent powdered organic materials are thus recovered cheaply. The preliminary removal of dust from the air used in the treatment insures a food product free from dirt. Another exceedingly interesting installation is one for collecting potash salts volatilized from felspar, at Catasaupua, Pa. This undoubtedly is the first commercial installation in the United States for producing potash salts primarily, the residue being by-product. While the scheme as a whole is still in the experimental stage, the collection of the salts by electrical process is proved.

Under consideration also is the collection of aluminium oxide, powdered sugar, powdered starch, powdered coal, tin oxide for enamel, powdered

sulphur, potash from burning of molasses, potash from cotton seed hulls, black smoke from roundhouses and other places, fertilizer dusts, zinc sulphides, zinc oxides, powdered borax, fumes and dust from calcium carbide furnaces, dust from rotary lime kilns, dust from pyrite burners in sulphuric acid plants, gold and silver from jewellers' sweepings, fumes from linseed oil burning kettles in varnish works, powdered alundum, ammonium chloride, nitric acid, etc.

The utilization of the process in conjunction with ventilation of the railroad tunnels in Baltimore is also under consideration and it may be installed there unless electrification of the railroads is decided upon. Furthermore, since it is so easy to remove dust from air by these processes, they may be useful in connection with the ever-present problem of suitably ventilating subways."—Linn Bradley.—(*Pro. American Inst. of Electrical Engineers.*)—*Engineering Magazine*, June, 1915, p. 427.—(H. A. W.)

Lead Acetate in Silver Extraction.—"The experiments described here were made at the Mexican mill, Virginia City, on the effect of lead acetate and zinc salts on the extraction of silver from ores, and also with the idea of finding some method, other than empirical, for the determination of the amount of lead acetate to be used for a given silver ore.

After laboratory tests were made on the ore the extraction appeared to be slightly better when using lead acetate in new solutions, but slightly poorer when using mill solutions. This was attributed to the zinc accumulating in the mill solutions. As the Mexican mill is a continuous-treatment plant, it was decided to run long mill tests with and without lead acetate, keeping all other conditions as nearly the same as possible. The ore was ground so that 92% would pass a 200-mesh screen. The temperature was kept at 95° F., cyanide at 2·5 lb. KCN, and lime at 0·5 lb. CaO. The accompanying table shows averages for each period.

The zinc was determined daily by direct precipitation as sulphide. The reducing power of the solution was also determined. No connection could be found between the extraction and the reducing power. The extraction of silver was 3·2% better in the second test than in the first, the time of treatment was 18% less in the second than in the first, and the consumption of cyanide was ¼ lb. more in the second than in the first.

Average Results of Mill Tests.

	Test No. 1.	Test No. 2
Length of test, days	92	72
Tons per day	72	85·5
Silver, oz., heads	15·88	17·97
Silver, oz., tails	2·22	2·24
Extraction, per cent.	84·2	87·4
Acetate, lb. per ton	0·9	nil
Zinc, gm. per 100 cc. solution	0·0848	0·0168

During test No. 1 we found lead at head of the zinc sump solution by Clennell's method.* I believe that accumulation of zinc in the first period was due to the use of lead salts and that the amount of zinc in solution should be kept as low as possible, or rather that there should be only sufficient lead or zinc salt in solution to precipitate the alkaline sulphide. An excess amount of metals in cyanide solution will retard the rate of extraction. As alkaline sulphide will precipitate lead, zinc and silver out of cyanide solution, in the order named, a daily estimation of zinc should be made to determine whether the silver is protected or whether it is in excess.

During 1913 the Mexican company took over a lease on the Monte Cristo mine and I did more work along these same lines. Before starting the mill on this ore, two series of laboratory tests were made—one with freshly prepared solution and the other with mill solution from Mexican ore—varying the proportions of solution to ore and the amount of lead acetate. The tests made with new solution showed that a dilution of 1 : 10 gave a maximum extraction without lead acetate, while at a dilution of 1 : 2, which would be more nearly mill-working conditions, the maximum extraction was obtained with 1½ lb. of lead acetate.

Tests made with mill solution at a dilution of 1 : 10 without the use of lead acetate gave maximum extraction, but slightly lower than that obtained with fresh solution. With a dilution of 1 : 2 the maximum extraction required ¼ lb. of lead acetate. Besides, an excessive amount of lead gave poor extraction in all tests. Judging from this information, the amount of zinc salts in solution appears to have some relation to extraction.

Upon starting the mill on this ore I determined not to add any lead salts until the zinc salts were lower and some marked change of extraction of silver occurred.

Daily tests were made for zinc and for alkaline sulphide by the nitroprusside test. The results are shown in the accompanying table.

Mill Tests on Monte Cristo Ore.

	Heads, Ozs. Silver.	Tails,* Oz. Silver.	Acetate, Lb. per Ton Ore.	Zinc, Grams per 100 cc.
1	6·12	2·20	nil	0·0650
2	5·17	1·54	nil	0·0420
3	5·04	1·58	nil	0·0380
4	6·96	1·56	nil	0·0380
5	9·21	1·76	nil	0·0203
6	16·62	1·80	nil	0·0124
7	15·72	2·18	¼	0·0050
8	11·59	3·03	¼	0·0050
9	11·21	3·95	¼	trace
10	11·63	4·60	½	trace
11	13·79	3·04	½	trace
12	13·26	2·65	¾	trace
13	18·83	2·10	1	0·0050
14	10·03	2·09	1¼	0·0168
15	14·36	1·67	1½	0·0168
16	10·46	1·46	1¾	0·0230
17	14·56	1·37	1½	0·0295
18	8·31	1·07	1¼	0·0395
19	5·51	1·24	1	0·0346
20	5·50	1·14	1·0	0·0356
21	5·03	0·80	1·0	0·0380
22	6·15	0·74	1·0	0·0320
23	4·88	0·54	1·0	0·0340

*The tails for a given day are for heads three or four days previous.

The table gives the silver content for heads and tails, the amount of lead used and the amount of zinc in solutions returning to the head of the gold vats.

In testing for alkaline sulphide we were unable to detect any, but from the clean-up made during the period when no zinc was in solution matte was formed on our bars. This was the only time we had matte. I believe that alkaline sulphide must have been in solution at that time. During the

* *Engineering and Mining Journal*, Sept. 9, 1911.

period we had trouble in titrating with silver nitrate. After first adding silver, a dark brown precipitate would form, which would disappear on shaking, and the final precipitate would be dark.

Complex cyanide solutions were made in the laboratory to test for alkaline sulphides by the nitroprusside method. It was found that cyanide solutions containing small quantities of sulphide would react with nitroprusside only in absence of silver or zinc salts."—C. R. MORRIS.—*Eng. and Mining Journal*, July 31, 1915, p. 189. (H. A. W.)

REFINING CYANIDE PRECIPITATE.—"The mixed product that results from the usual clean-up in cyanide plants consists, before any acid treatment and apart from gold and silver contents, of two classes of constituents. The first class includes those substances which are, or could be, rendered soluble with the aid of sulphuric acid, though not necessarily under the conditions which are possible at cyanide plants; and the second class includes those substances which could not be rendered soluble by these same means.

Separation of Impurities. The group of substances that are soluble in sulphuric acid consists principally of zinc metal and compounds, copper metal and compounds, and iron and its compounds. There are, besides, various complex cyanides, ferrocyanides, and other elements which occasionally exist in various combinations and quantities, such as magnesium, aluminium, nickel and tellurium.

The group of substances insoluble in sulphuric acid consists principally of sand, calcium carbonate, calcium sulphate, lead and its compounds, and carbonaceous matter. Of these substances sand is usually present as a result of defective filters or of atmospheric dust. Both of these causes can be guarded against and therefore the amount of sand present can be kept down. Calcium compounds cannot always be so easily kept low. Lead, apart from being a more or less regular constituent of zinc, has often to be used to aid the precipitation work. Carbonaceous matter is also present in zinc to a slight extent. It is evident that if everything which is theoretically soluble in sulphuric acid were actually so, there would remain for smelting very small quantities of impurities after acid treatment. The general analysis of the slimes before and after acid treatment, and also after the drying or calcining that follows the acid treatment, is pretty well known to operators.

Usual Acid Treatment Incomplete.—As a rule there remains after acid treatment a large proportion of substances belonging to the first class. This is evident from the base metals in the bullion produced, also from a rough, pocket-lens examination of the slimes after acid treatment and calcination.

The fact that certain complex cyanides are not affected by the acid treatment is shown by the ammonia given off during the subsequent calcination. It is easily seen by a small-scale experiment that material which has been acid-treated and calcined will usually, by further acid treatment, yield in a soluble condition more of the substances of the first class. This further acid treatment has of course been rendered effective by the calcination (oxidation) which has converted sulphides into sulphates or oxides, and insoluble cyanides into soluble compounds. Beyond this, any metallic zinc that during the first acid treatment might have been protected by lead or copper or the sulphides of these metals, will be oxidized during calcination or otherwise prepared for further acid treatment.

The ordinary acid treatment is generally carried out in a lead-lined vat fitted with a water-seal cover, gas-escape pipe, steam-heat attachment and mechanical agitator, or in some apparatus that answers the same purpose. In any case the efficient agitation is most important, as all the material that is to be acid-treated should be kept well in motion.

As this acid treatment proceeds the increasing amount of fine slimy material formed will tend more and more to protect material from the action of the acid, and therefore efficient agitation, even to the point of attrition, becomes more and more important. When the ordinary acid treatment has been finished so that practically no further action takes place and the slimes are still in the acid-treatment vat, any substances of the first class which remain can be removed by the procedure that I shall outline; and any lead or lead compound that may also be present can be turned into sulphate so that it will not appear in the final bullion bar.

Removal of Residual Impurities.—First of all decant or filter off as much as possible of the well-settled zinc solution, leaving all the slime in the acid-treatment vat; then run in fresh water till the total weight of water is about five times the assumed dry weight of the slime in the vat. While continuously and thoroughly agitating, add separately strong sulphuric acid (whatever strength is available) and then bichromate of potassium previously dissolved in as little hot water as possible. It is better while this treatment is going on to keep the contents of the vat heated. These two chemicals are always required in the proportion of 2½ lb. of actual sulphuric acid (or say 4 lb. of 66% acid) to each pound of solid bichromate of potassium. The actual total amount of acid and bichromate to be used will naturally depend on the quality of the slime in the vat; but each addition of say 10 lb. of sulphuric acid (or 16 lb. of 66% acid) will be followed by 4 lb. of bichromate of potassium dissolved in as little hot water as possible.

Three or four minutes after adding the bichromate a small sample of filtered solution from the vat can be tested for chromic acid by well-known tests or it may be evident from the colour of the filtered solution. When chromic acid is present and persists for a minute or two the acid treatment is finished, but until this persistence of chromic acid in the solution is obtained it is necessary to continue the addition of sulphuric acid and bichromate as described. When treatment is finally finished, the slime, after very thorough water washing, is collected and dried in the usual way and is then ready for smelting.

It has been found that a small-scale trial in a glass flask will show in a few minutes the alteration, if any, when the treatment with acid and bichromate follows the ordinary acid treatment. This treatment with acid and bichromate should not be carried out until the ordinary acid treatment has been worked to a finish, as previously described. Salt or hydrochloric acid should not be used either in the ordinary acid treatment or in the subsequent acid and bichromate treatment. After treatment with acid and bichromate the slime will settle well without any of the usual aids, such as glue and alum.

Some Alternative Practices.—Instead of bichrom-

ate of potassium there could be used the same weight of bichromate of sodium. Similarly, permanganate of potassium or sodium in calculated equivalent quantity could be used, but it is not so satisfactory for several reasons. If the treatment with acid and bichromate is carried too far the only trouble is that silver will begin to be dissolved, although this can be recovered from the solutions if necessary. The finished slime is heavy and after slight heating, in order to dry it thoroughly, will generally appear as annealed gold, and sometimes will show the gold colour even before heating.

Unless much sand is present no flux other than borax glass is required; it is well, however, always to add a little nitre to oxidize any small amounts of carbonaceous matter that may be present. The finished slime must be smelted in a clay crucible. It must not be put in a plumbago crucible and should not be in contact by stirring or otherwise with plumbago or any reducing agent during or before smelting; otherwise a mixed lead matte may be formed. The final bullion should be practically free from zinc, lead and copper. The extra cost for acid, bichromate and labour in this work will be more than compensated for by the decreased cost of fluxes in the smelting; by the fact that no light, dusty slime will be handled; by the saving of time, labour, and fuel in the smelting; and by decrease in the amount of slag produced. Practically all the bullion will be obtained at once in the bars."—H. T. Durant.—*Eng. and Min. Journal*, Sept. 25, 1915, p. 523. (H. A. W.)

Gold Precipitation on Paper.—"Apart from any other inherent advantages which electrolytic precipitation may possess over the chemical systems, the former method merits attention at the present time on account of the high price of zinc. Hitherto electrolytic precipitation appears to have received the cold shoulder, doubtless because of the many extravagant claims made on its behalf which have not been realized in practice.

Weakness of former electrolytic processes.—In my experience, electrolytic precipitation on sheet-lead cathodes proved highly satisfactory, precipitation being just as complete as when zinc was employed. But electrolytic precipitation on a metallic cathode and chemical precipitation by means of either filiform zinc, zinc dust or aluminium all involve, subsequent to precipitation, separation of the precipitated precious metal from the lead, zinc or aluminium, as the case may be, before the precious metal is recovered in the form of bullion.

The method of electrolytic precipitation on paper cathodes, herein described, has been evolved with the object of eliminating as far as possible the separation of the precipitate from the precipitant. If precipitation on paper cathodes proves to be satisfactory—and experimental work gives ground for high hopes—then it would appear to be a step in advance of precipitation on a metal, owing to the simple operation required after precipitation to yield the precious metal in the form of bullion.

Before proceeding to a detailed description of the method, it is well to say that I am far from laying claim to any originality in the matter of making a non-conductor, such as paper, a conductor of electricity. The art of making non-conductors conductors has been known for more than half a century. Originality, if originality there be, lies in the manner of application of this known art to the cyanide process, that is, in precipitation on a combustible cathode.

Further, no claim is made that this method has yet emerged beyond the experimental stage, inasmuch as time and opportunity have not permitted an actual clean-up on a full commercial scale. At the same time, all experimental work herein described was carried on with actual mill solutions and in an electrodeposition vat of full commercial size, containing 3,500 sq. ft. of sheet lead cathode surface. Opportunity only permitted of the trial for 30 days of a limited number of paper cathodes. These latter were hung from the negative crossbar, side by side with lead cathodes, and appeared to receive a deposit of precious metal just as readily as the lead cathodes.

Briefly, the process consists in the electrolysis of the clear cyanide solution, and the precipitation of the precious metals on paper cathodes. Whenever it is desired to make a clean-up, incineration and melting of the cathodes will yield the precious metals in the form of bullion.

Preparation of Paper for Electrodeposition.—Any paper of reasonable stoutness and having a smooth surface, such as the pages of the *Engineering and Mining Journal*, answers well. It is cut into strips of the required size, preferably not more than 2 or 3 in. in width. These strips are immersed momentarily in melted beeswax. Before the wax has solidified, the sheets are dipped in flake graphite (not plumbago), which is thus made to adhere very firmly to the paper. When the latter has cooled, more graphite is rubbed on the surface of the paper until the latter appears highly polished. The function of the beeswax is twofold—it not only renders the paper waterproof, but also serves as a bed for the graphite. It is advantageous but not absolutely necessary to dip the sheets, prepared as described in alcohol and again in water, to remove any air adhering to the surface. The cathode is now ready for immersion in the electrodeposition vat. Electric connection is established with the negative main in the following manner: Each cathode strip is clamped at the upper edge between two strips of sheet copper about $\frac{3}{4}$ in. wide, by means of small clips (ordinary tie clips answer very well). A copper hook soldered to one of the copper strips suspends the cathode from the negative crossbar.

Details of the Electrodeposition Vat.—As has already been mentioned, the great majority of cathodes employed were of sheet lead, opportunity not permitting of a trial of more than a limited number of paper cathodes. The vat had a total cathode area of 3,500 sq. ft., giving it a capacity of 2 tons of solution per hr., that is to say, solution flowing through it at this rate was entirely robbed of its precious metal contents. The vat was constructed of 3-in planks and was 25 ft. long, 5 ft. wide, 3 ft. deep, all these being dimensions in the clear.

Anodes were of No. 10 sheet iron, 5 ft. 1 in. long by 2 ft. 8 in. deep, and were fitted into saw cuts in the sides and bottom of the vat. To promote thorough circulation and ionization alternate anodes were raised 2 in. above the level of the solution, the remaining anodes being fitted into the bottom of the vat. Each alternate pair of anodes, therefore, formed a water-tight compartment, consequently the whole of the solution was caused to flow over one anode and under the next. The distance between anodes was 3 in. Anodes were enclosed in cheesecloth to prevent chance of short

circulating. Electrodes were connected in multiple. Anode connections below the level of the solution are gradually eaten away.

The action of the electric current is gradually to oxidize the anodes. In this particular case the anodes were of iron and the anode product consisted mainly of floating flocculent hydrated ferric oxide which accumulated in the anode compartments. The removal of this floating anode product was readily effected by the following device: Above the level of those anodes which projected 2 in. above the level of the solution, the vat had at its outlet end an overflow lip. Below this lip at the normal level of the solution was the normal outlet pipe. By closing a valve on this latter pipe the level of the solution could be raised above the tops of the projecting anodes, causing a direct flow throughout the tank and the rapid and complete removal of the floating anode product over the flow lip into a sump vat. The only labour involved in this operation was the opening and closing of a valve. This operation was found necessary only once in every 24 hours. It is to be noted that the anode product always carries high precious-metal values, which, however, are loosely held, and which rapidly pass into solution again in the sump vat.

A weak current was used, the current density being 0·02 amp per sq. ft. of cathode surface, and the pressure from 3 to 4 volts.

Noteworthy Features of the System.—The following appeared to the writer to be advantageous features: The small amount of attention required; the ease and cheapness with which a clean-up could be made as compared with other methods of precipitation, incineration and melting of the cathodes yielding the precious metals in the form of bullion; the continuous nature of the process. A clean-up entails no stoppage of operations whatever. Cathode sheets may be removed as often as desired and replaced by new ones without switching off the current. The precipitated precious metals adhere firmly to the cathode. There is no danger of mechanical loss from particles dropping off. At first consideration it might be thought that a paper cathode would rot after lengthy immersion in the solution. The reverse of this takes place. As the deposit of previous metal increases, the paper cathode becomes more and more metallic in appearance and properties. After 30 days' immersion, paper cathodes were found to ring, when tapped with a piece of metal. There is a small consumption of electric power and the cost of precipitation is low.—DOUGLAS LAY. *Eng. and Min. Journal*, Aug. 14, 1915, p. 276. (H. A. W.)

Abstracts of Patent Applications.

(C.) 184/15. Botha Helmuth Lindhorst. Improvements in means for actuating shaking chute conveyors. 7.8.15.

This application refers to an arrangement for actuating shaking chutes for stope conveyors. To the ordinary chain suspended chute the applicant proposes to attach a fulcrum carrying a form of bell crank lever, the horizontal arm being the operating handle, while the vertical arm is connected to a chain carried from the hanging wall at a suitable horizontal distance from the bell crank. When pressure is applied to the operating handle the chain attachment to the vertical arm becomes the central fulcrum and the point of attachment to the chute moves backwards taking the chute with it; on the handle being released the chute falls forward again, and the repetition of this action carries forward the rock in the chute.

(C.) 317/15. James Charles Hoskins Vaught. Improvements in Pneumatic Drills. 30.12.15.

This application refers to a rock-drilling machine of the pneumatic hammer type in combination with a support or shell and a reversible or double acting air feed mechanism for the said machine.

(C.) 96/15. William Rogers Sennett (1), Richard Pascoe (2). Improvements in Tube Mill Liners. 29.4.15.

This application relates to tube mill liners in which it is proposed to use old stamp shoes set with the shank end projecting inwardly with filling material between.

(C.) 209/15. William David Williams. Improvements in water spraying devices. 1.9.15.

This application refers to a water spraying device comprising a casting or casing having at one side a water inlet and at right angles to said inlet an air inlet and an outlet opposite the water inlet, the water passage from said water inlet to the said outlet being provided by a piece of tube and a cap adjustably secured in the outlet and constructed to form around the outlet end of the water pipe an adjustable annular air outlet passage.

(C.) 12/16. Gogu Constantinesco (1), Walter Haddon (2). A method and means for transmitting power by wave transmission through liquids. 18.1.16.

This application refers to a method or means for transmitting power for operating reciprocating apparatus such as pumps, drills, hammers and the like, by means of a series of periodic variations of pressure and volume travelling along a single liquid column.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

CULLEN, W., *to* "Westwood," The Avenue, Upper Norwood, London, S.E.

GORDON, W. S., *l. a.* Johannesburg; Bantjes Cons. Mines, Ltd., P.O. Box 2, Florida.

HARDY, J. A., *l. a.* Klerksdorp; New Heriot G.M. Co., Ltd., P.O. Box 10, Cleveland.

HUTCHATT, C., *to* Shamva Mines, Ltd., Shamva, Rhodesia.

JONES, J. A., *to* P.O. Box 6, Millsite.

MARRIOTT, F. A., *to* Golden Kopje Mine, Lomagundi, Rhodesia.

MAXWELL, F. A. G., *l. a.* Randfontein; P.O. Box 1242, Johannesburg.

ROME, A. S., *l. a.* Nigel; Bell Mine, Que Que, Rhodesia.

WELSH, T. S., *l. a.* Germiston; P.O. Box 8, Cleveland.

CRUMPTON, H., *to* Eldorado Mine, Mashonaland, S. Rhodesia.

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Proceedings
AT
Ordinary General Meeting.
April 15, 1916.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, April 15th, 1916, Prof. J. A. Wilkinson (Vice-President) in the Chair. There were also present:—

21 Members: Messrs. J. Gray, Prof. J. S. Cellier, E. H. Croghan, John Watson, E. M. Weston, A. McA. Johnston, Prof. G. H. Stanley (Members of Council), H. R. Adam, W. Bleloch, H. C. Boydell, J. Chilton, H. W. Gill, B. J. Hastings, W. H. Jane, J. A. Jones, P. T. Morrisby, S. Nettleton, S. Newton, E. A. Osterloh and W. S. V. Price.

6 Associates and Students: Messrs. J. Cronin, O. A. Gerber, J. Gibson, J. Ingram, D. W. Pugh and H. Rusden.

10 Visitors, including Mr. H. Foster Bain, Mr. C. B. Kingston, Mr. F. Searls, and Fred Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General Meeting, as printed in the March *Journal*, were confirmed.

MEMBERS FOR ELECTION.

Prof. G. H. Stanley and Mr. A. McA. Johnston were appointed scrutineers, who declared the result of the ballot to be that the following gentlemen had been elected members of the Society:

LUCAS, CLAUDE DAVIS, P.O. Box 1, Mill Site, Randfontein. Mining Engineer.

WAITES, THOMAS PARK, P.O. Box 5705, Johannesburg. Miner.

INDISPOSITION OF PRESIDENT.

The Chairman: I am sure we are all very sorry to hear that the President is absent this evening, owing to illness, but he has our best wishes for a speedy recovery.

MR. RALPH STOKES.

The Chairman: It is with great pleasure that I rise to propose a very hearty vote of congratulation to one of our Corresponding Members of Council. I refer to Mr. Ralph Stokes, who has been promoted from Captain to Major, and Assistant Inspector of Mines on the General Headquarters Staff, for distinguished services in Flanders.

The Secretary: In that connection, I wish to state that, two or three meetings ago, I mentioned that certain gentlemen had given me a couple of guineas to send comforts to Major Stokes and Capt. Gerard Williams, our two Corresponding Council Members, for their men at the front, and, subsequently, another two guineas were received. Three guineas of that is being sent to Major Stokes and one guinea to Capt. Williams for comforts for their men. If any other donations are received, they will be forwarded in the same way.

WELCOME TO MESSRS. FOSTER BAIN, F. SEARLS AND C. B. KINGSTON.

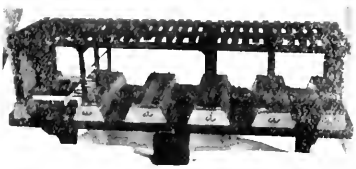
The Chairman: I should like to draw your attention to the fact that we have with us this evening two very distinguished American visitors: Mr. Foster Bain, the Editor of the *London Mining Magazine*, and Mr. Searls, who is doing, I believe, some prospecting work on the East Rand. There is no necessity to dilate further on this, and I would ask you to extend to them a very hearty welcome.

I also wish to welcome an old Member of Council, Mr. C. B. Kingston, who was once resident amongst us. I am sure we are all delighted to see him again looking so well and more in that he has taken the opportunity during a short stay here to pay us a visit, which fact alone proves his continued interest in our work.

CONCRETE SHAFT EQUIPMENT AT THE BANTJES CONSOLIDATED MINES.

By W. W. Lawrie & G. Heddick Smith (Members of Council.)

The concrete shaft equipment in the Central Shaft of the Bantjes Consolidated Mines consists of concrete shaft rail foundations as continuous stringers through the shaft, as shown in the photograph of a model, a.a. Fig. 1.

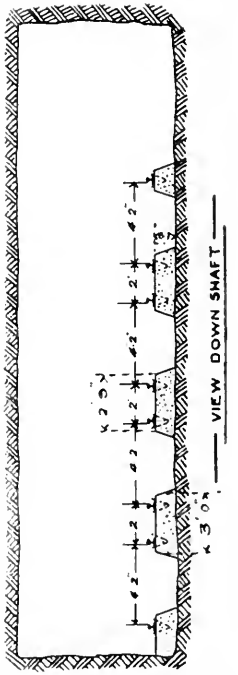


The concrete stringers are a substitute for timber sills in inclined shafts. The shaft rails are laid and held in position on the stringers by holding-down bolts as described later.

The Bantjes Consolidated Mines Central Shaft is an inclined shaft, 33 ft. 7 in. 6 in., dipping at an angle of 34½°. The position of the shaft in respect to the reefs varies, due to dikes and faulting. The distance between the South Reef and the Main Reef Leader is 95 ft., measured at right angles to the angle of dip, which averages 35°. Throughout the whole length of the shaft such shaft pillars have been left as to ensure no danger of movement in the shaft, not only of the hanging-wall but also of the footwall, the latter being an essential point where concrete shaft-rail foundations are used.

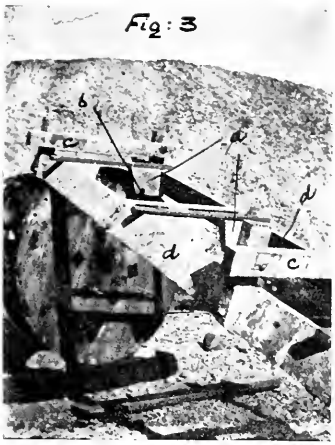
Dimensions of Concrete Stringers. Fig. 2 shows a cross section of the shaft. From this it will be seen that the rail gauge is 1 ft. 2 in., the rails being of the ordinary 60 lb. type, the distance between the outer edges of the rail head and the top edges of the stringers is 1¹ in. (not 5¹ in. as erroneously marked in sketch). The width across the top of the stringers, which carry two sets of rails being 2 ft. 9 in., the width of the outside stringers placed against the side of the shaft varies according to the irregularities in the width of the shaft, making the average width of the outside stringers 1 ft. 6 in. The angle of batter of the sides of the stringers is 65°. The thickness of the stringers varies, due to irregularities of the footwall, the average thickness being 1 ft. 9 in.

Method and Materials of Construction of



the Stringers. To equip a section of the shaft with stringers, work is commenced from the bottom portion of the section to be built, and the concrete built upwards through the shaft. Boxing, as shown on Fig. 3, in lengths of 18 ft., is placed in the required position in the shaft and carefully laid to the correct grade, etc., wood blocks, a.a. Fig. 3, to form the pigeon holes for the rail holding-down bolts are bolted to the boxing. It will be seen that these wooden blocks are slotted at the bottom edge, b. Fig. 3, this is to allow for the iron bars, a. Fig. 1, 3 ft. long by ½ in. diameter, which are built into the concrete

and round which the hook holding-down bolts are placed as shown in Fig. 4. Origi-



nally the holding-down bolts were placed 6 ft. apart. It is now found better to space them 4 ft. apart, thus the wooden blocks forming the pigeon-holes are placed 4 ft. apart on the boxing. When the length of boxing is in position, being held in any position required by means of cross timbers, b.b. Fig. 3, which in practice project beyond the sides of the boxes and are blocked up off the footwall of the shaft.

A mixture of, roughly, six parts of unwashed development rock—quartzite or reef, not dike matter—obtained from the most convenient point and having been passed through a screen of 2 in. square mesh, and one part of Pretoria Portland cement is shovelled into the box after a thorough mixing, with the necessary quantity of water, on a temporary staging, placed as near the box as possible. When the filling of the box is complete the box is lifted off the concrete, the concrete being levelled and smoothed over by hand with a trowel and the corners pointed up. If the section completed is the bottom section, and other sections have to be built above it the box when removed is placed above and against the new concrete, which is allowed twelve hours to set, and the operations repeated.

One white man and nine natives can build one 18 ft section of concrete in one shift, thus for each stringer the cost per foot is:—

Wages.—White... ... 1 at 22s. 6d.
Native ... 9 at 2s. 9d.
Total 47s. 3d. per 18 ft. section.

$$\frac{47\ 3}{18} = 2\ 7\tfrac{1}{2}\text{d. per foot.}$$

Material.—Assuming that stringers 3·0 × 1·75 cross section are being built, then cubic foot of concrete per foot run 3·0 × 1·75 = 5·25 cub. ft. The theoretical mixture being 6 to 1 ∴ $\frac{5\cdot25}{7}$ = 0·75 cub. ft. of cement per foot of stringer, or 13·5 cub. ft. for a section of 18 ft. Allowing 2·1 cub. ft. in one bag of cement, then $\frac{13\cdot5}{2\cdot1}$ = 6·4 bags of cement.

In practice, due to spillage, etc., it is found that on an average 10 bags are required for an 18 ft. length of stringer, thus:—

$$10\ @\ 6\ 6 = 65/- \quad \frac{65}{18} = 3\ 7\text{d. per foot.}$$

The total labour and material costs per foot of single stringer is therefore 6s. 2½d. Thus for equipping the shaft complete with stringers, as shown in Fig. 2, the cost per foot of shaft would be 6s. 2½d. × 4½ (4½ being the number of equivalent 3 ft. × 1·75 stringers), 27s. 11d., say, 28s.

As a comparison, the following is the cost of equipping a similar shaft at the Nourse Mines with timber sills:—

Wages.—Two white men with 8 boys equip 66 ft. of incline per month, it is estimated that three-quarter of their time is employed in laying down and blocking sills, the other quarter being employed in laying rails, etc.

The wages bill is therefore as follows:—

White wages, 1·5 shifts × 25 shifts × 22s. 6d. = £42 3s. 9d.

Native wages, 6 shifts × 25 shifts × 3s. = £22 10s. 0d.

Total ... £64 13s. 9d.

Material.—Timber:—

Sills 32 ft. × 8 in. × 8 in. = 14·2 cub. ft.
Studdles 30 ft. × 6 in. × 8 in. = 10·0 cub. ft.
Blocking 12 ft. × 6 in. × 8in. = 4·0 cub. ft.

Total = 28·2 cub. ft. at 4s. 6d. = £6 6s. 10d. per set.

As the sills are 5 ft. 6 in. centres there are 12 sills in 66 ft., therefore the cost of timber for

66 ft. = 12 × £6 6s. 10d. = £76 2s. 0d.

Framing of sets cost 15s. each, or £9 for the 12.

Hanging bolts cost approximately £1 per set

The total cost for 66 ft. therefore:—

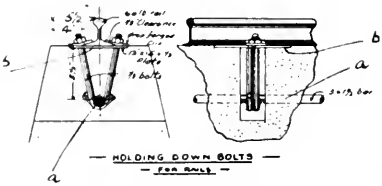
Wages	£64	13	9
Material	88	2	0
Framing sets ...	9	0	0
Total	£161	15	9

or £2 9s. 0d. per ft.

Assuming the cost of bolting-down the rails to the timber sills to be the same as bolting-down the rails on to concrete, the cost of the concrete stringers is 21s. per foot less than the cost of similar timber sills. This is a saving of 43 .

Method of Laying Rails on the Stringers.—The rails are held down on the concrete stringers, as shown in Fig. 4, which is self explanatory. The bedding plates, b.b. Fig.

FIG 4



HOLDING DOWN BOLTS
FOR RAILS

4, bridge across the pigeon-holes and are bedded into the concrete. Strictly speaking the rails only touch the bedding plate, but in practice the space between the concrete and the chair of the rails soon becomes filled up with fine dirt. Originally U bolts were used instead of two-hooks, Fig. 4, for holding down the rails, the latter having been found preferable.

Where required, wooden roller frames are placed in hitches cut in the concrete stringers.

Conclusion.—Provided always that there is no danger of movement of the footwall of a shaft, concrete stringers as shaft rail foundations are preferable to timber sills in incline shafts, with any angle of dip up to 15°, for the following reasons:—

1. A longer life can be obtained from the rails, which can be allowed to wear down with safety to a greater extent on a continuous concrete foundation than they can when laid across timber sills.
2. Skip derailments do much less damage with concrete than with timber.
3. Concrete is a much more permanent foundation than timber, requires no repairs or renewals, and appears to suffer no damage from any cause whatever.
4. In deep shafts considerable time is saved during the equipment of the shaft, there being no long timbers to handle.
5. If more than one pair of skips is to be used in a shaft, the shaft may be equipped for two skips only until such time as other hoisting capacity is required, thus saving the extra cost of equipment at the moment.
6. Less damage to rails from corrosion by acid water.

With regard to the maximum inclination at which concrete stringers can be used in incline shafts, it is thought that by pegging the footwall at inclinations from 15° up to 60° concrete stringers could be safely held in position.

The method of concrete shaft equipment as above described was first introduced at the Bantjes by Mr. Kinkead. No credit is due to the writers for anything appertaining to this concrete work, but having in practice found this type of shaft rail foundation the most satisfactory, this short paper has been written in the hope that their experience may be of some use to others.

Mr. E. M. Weston (*Member of Council*): I believe this kind of shaft equipment was first introduced in the Lake Superior copper mines, and I expect our American friends will know all about it. It seems to me this installation at the Bantjes has been very successful, and the practice might be introduced into quite a number of other mines along the Rand, as the author has suggested with great advantage. I propose a vote of thanks to the authors for their paper.

Mr H. Foster Bain (*Visitor*): I regret that I am unable to give details as to Lake Superior practice in concreting inclined shafts save in regard to some new work at the Creighton Mine of the Canadian Copper Company, which is only in a broad sense in the Lake district. At this mine new equipment has recently been put in with a view to increasing the output and decreasing cost. As the work was done under the direction of Mr. W. W. Mein, long associated with the Rand, and marks the introduction of certain features of Rand practice in America, it may be of some interest locally. The shaft differs from that at the Bantjes in that it has a steeper angle, being

at 55° in place of 45°. The ore is handled in 10 ton skips, planned to permit hoisting at a rope speed of 2,500 ft. per minute. The winding engine is driven by electricity with Ilgner-type control and, incidentally, the steel head frame is the highest at any American mine, rising to 143 ft. in order to accommodate crushing and picking machinery placed in this case at the shaft as on the Rand. In considering the equipment of this shaft a careful study of the use of concrete stringers under the rails was made and experience was taken not only of mine practice but of railway work, both on main lines and in such situations as the New York subway, where enormous traffic is handled at high speeds. From such a study, and having in view the angle of the shaft and the proposed rope speed with the proper tieing in of the rails, it was not felt safe to place the rails directly upon the concrete stringers. Instead steel girders of a special design were set in the concrete at 7 ft. 6 in. intervals as sleepers. These were allowed to project outside the concrete stringers, and 85 lb. rails, 4 ft. 6 in. gauge, were attached to them by convenient fastenings, a bit of belting being placed between the base of the rail and top of the girder. This construction proves very satisfactory. The concrete stringers, 3 ft. 6 in. in width running down the centre of each compartment, which were substituted for individual posts merely because of greater ease in construction, are, of course, anchored to the foot-wall, and rollers for the rope are easily placed on the cross girders. A supplementary feature that may be of interest is that in one compartment an extra line of rails between the usual rails, carries an industrial track. Heavy pieces of machinery can therefore be placed on special trucks in the machine shops and lowered without transfer to the particular level where wanted.

Mr. W. S. V. Price *(Member)*: In one instance where I saw the concrete stringers, I have seen pieces of belting put under the chairs. It gives a better grip and probably prevents corrosion, and it certainly makes for easy running. I have known a skip come off the rails and run down the concrete for 80 ft., but it did very little harm; and it is a very small job to replace 15 ft. or 20 ft. of concrete. I cordially second the vote of thanks.

The vote of thanks was passed unanimously.

CLEAN-UP PRACTICE AT THE SIMMER DEEP.

By W. H. Jane & E. Davey.

(Printed in Journal, October, 1915.)

REPLY TO DISCUSSION.

Mr. W. H. Jane *(Member)*: The authors regret that there has been no discussion on this paper. We feel certain that many members of this Society could have given some useful information. We submitted this paper to the Society in the hope that members would give a brief description of practice on other plants. We are aware that in many mills on the Witwatersrand it is customary to carefully collect all concentrates possible, and treat them in the clean-up room. It would be very interesting to have opinions expressed on this practice. We have an idea that this can only result in increasing the mill extraction at a cost that is not warranted.

On the other hand, it is customary on some plants to send all black sands, after being cleaned of amalgam, into the current tailings. Experience has shown us conclusively that the treatment these sands can receive in the ordinary cyanide works practice, will fail to extract anything but a small percentage of its gold-content. It would be very interesting to know to what extent osmiridium is recoverable in the mill clean-up rooms of the reef. It exists here in small quantities, and costs nothing to recover, as the work entailed in recovering it is a necessary part of the treatment of the steel from tube mills. The authors are pleased to find that the recovery of diamonds and osmiridium is of scientific interest in connection with the geology of the Witwatersrand. With the huge tonnages the mills of the Witwatersrand are handling today, it is necessary to make the mill clean-up rooms of first class importance in the extraction of gold.

The recovery of steel from the tube mill circuit, and its careful classification and treatment, is of great benefit to the following treatment by cyaniding, and very profitable.

In conclusion, the authors hope that millmen have found the paper beneficial.

THE CHIEF SOURCES OF ACCIDENTS IN WITWATERSRAND MINES.

By Chas. E. Hutton, M.Inst.M.M.,
Inspector of Mines (Member).

(Printed in Journal, November, 1915.)

DISCUSSION.

Mr. E. M. Weston *(Member of Council):* I should like to say that Mr. Hutton's paper was subjected to some disadvantage, because it naturally dealt with accidents due to explosives, and we had already rather fully discussed that question in connection with another paper. One can only say that this paper of Mr. Hutton's is most comprehensively and carefully compiled, and there seems to be very little one can add to it at all. In view of the fact that the chief source of accidents, the use of explosives, was discussed a few months earlier, if Mr. Hutton is disappointed with the discussion on his paper, I hope he will remember that the subject was not altogether fresh.

THE CONGLOMERATES OF THE WITWATERSRAND.

By E. T. Mellor, D.Sc. (Lond.),
M.Inst.M.M.

(Printed in Journal, February, 1916.)

DISCUSSION.

Prof. G. H. Stanley *(Past-President)*: In connection with this paper, Mr. Whitby has asked me to exhibit a specimen. I do not know what the exact bearing is on this paper, but it is certainly interesting from the gold occurrence point of view. It is a specimen of reef from the Klerksdorp district, quite an ordinary looking iron stained quartzite. There is nothing extraordinary about its appearance, but, if it is broken, it immediately disengages a quantity of sulphuretted hydrogen gas, the more, apparently, the richer it is in gold.

Mr. H. F. Marriott *(Life Member)*: Dr. Mellor's paper on the Conglomerates of the Witwatersrand is the most comprehensive and instructive yet issued on the subject. The author has had the advantage of collecting and marshalling his facts, and of letting them tell their own story and develop the theory he enunciates therefrom. Earlier students into the mystery of the "Birth of the Witwatersrand" were not so favoured, and had perforce to construct their theories and then secure evidence as it became available.

Dr. Mellor has invited us to say farewell to several beliefs that have earned the respect due to age, not the least of which is the mode of origin of the "interbedded dike," and the doctrine of transference of gold contents from the Main Reef Leader to the South Reef, and vice versa. He has dissipated that dream of the enthusiast in ore deposits, viz., the finding of the key to a system of shoots penetrating to greater depths, a dream none the less attractive for the reason that shoots in the form in which they are usually presumed to exist have never yet materialised in this formation. He has instead given us a rational statement of the condition of the various gold bearing beds as they are now known by actual development, and has drawn an inference of their origin, which it will be difficult to find an equal amount of corroborative detail to controvert.

The favourite dictum of the opportunist—the decrease of value in depth—based on suitably selected instances, can no longer raise unfounded doubts in the mind of the investing community, for the accident of depth is now clearly shown to be subsequent to the deposition of the gold, and the professional pessimist will be forced to turn his attention to increase of temperature, ventilation, increased cost of hoisting, and other economical problems coincident with mining at greater depths, which are in a fair way to being satisfactorily dealt with in the light of experience now being gained in the mines of to-day.

Credit is due to Dr. Mellor that he has seized the opportunity afforded him to rend us aright the tale of the gold bearing beds, for it must be noted that it has been reserved to him to be wise after the event. But our thanks are also due to those pioneers in thought and expression who have kept alive the question of the origin of this phenomenal occurrence, and have thus given the author his cue to the present logical elucidation.

The problem now before us is to find those extensions both laterally and at depth which will bear on his assumptions, and to open up further areas of the deposits containing as great a degree of gold concentration as any of those now left behind. That these richer areas yet undeveloped have every reason to exist is clearly shown in Dr. Mellor's evidence and conclusions. That

we have equal reason to find as rich sections of reef in the future as we have in the past is the obvious corollary of the paper. Whether or not the richer areas will continue in any direction in as close proximity to one another as they have in some of the already developed districts, or whether they will be found to be larger units of area and more widely separated, is a problem for us to solve which provides that element of speculation to the Industry which is the true life of the science of mining. Trend of thought has shaped itself in this direction since the evidence afforded by the work already done has shown that gold has been carried past or extracted from areas which are now in the upper levels of the mines and has been deposited in greater degree of concentration in areas which now form rich deeper levels below their poorer neighbours.

It affords a certain amount of relief to the engineer worried with the occurrence of abnormally poor stretches of banket to realise that these have been denuded of their erstwhile gold contents prior to the consolidation of the deposit, and have provided the valuable constituent which has ensured the reputation and results of the next adjoining richer mines in the vicinity, and in cases where these correspondingly richer areas have not yet been discovered there is the distinct incentive to apply all the resources of science to trace them up.

It has been said time and again that the Rand has arrived at the state of a purely commercial industry, and succeeding events have as often effectively contradicted this view. The present condition of the developments in the Far Eastern Rand is a striking example of the futility of prophesy on insufficient data, and now Dr. Mellor's deductions open up before us a vista of exploration and industry following on intelligent appreciation of the facts now at our disposal, which carries us far beyond even the calculations of future prosperity based on the most optimistic forecasts of recent times.

The steady flow of gold from the Rand has been of most valuable assistance to the Empire in the present struggle for human liberty, and there is every prospect that the output will continue indefinitely to supply the Empire's needs in consolidating a scheme of commercial unity which will bind the various sections of the British Empire in one nation for all time.

Prof. J. Cellier *(Member of Council):* I do not know that what I am going to say bears directly on Dr. Mellor's paper. I have had the pleasure of hearing Dr. Mellor's papers read before both this and the Geological Societies. Mr. Marriott's optimism is certainly very cheering in these days. It appears to me his idea of the gold continuing payable in depth as it has been in the past, must be bred from "Absence makes the heart grow fonder." He has been away from the Rand for a considerable while, and has probably forgotten to look at the latter day returns and compare them with what they have been in the past. In any case, I am not aware that Dr. Mellor controverted or otherwise the fact that values are decreasing in depth.

Mr. E. Weston *(Member of Council):* In going over Dr. Mellor's paper certain points were noted as inviting comment and I have dealt with them in the order they occur in the paper, hence the disconnected nature of this contribution. Regarding the Kimberley Reef, Dr. Mellor does not note that it produced quite a large tonnage of good reef where worked between Boksburg and the East Rand Extension Mine.

Re Bird Reef Series.—This series carries more than a moderate amount of gold on the Western Rand. Under the name of the East Reef it, and probably the Livingstone series, are being worked over a length of 12,000 ft. Stopes of high grade ore have been worked of a width comparable to anything on the Rand, and as many as four payable reefs have been worked in one area. Almost universally the richest gold occurs right against the hanging wall, unlike the case of the Main Reef Leader on the East Rand; but as with it the best gold is found associated with the largest pebbles.

Re Main Reef Leader Footwall on Far East Rand.—Dr. Mellor refers to them as "argillaceous rocks," so that the reef lies almost invariably directly on a "slate footwall." I know this reef in the Rand Collieries, State Mines, Brakpan, and Modder B., Nigel and Van Dyk, and I think this expression should be largely modified There are two foot-walls over this area. First, the typical dark-grey to black shale as "slate" footwall, in which in the case of the Rand Collieries sometimes occur lenses of reef almost identical with the Bastard Reef he describes on the New Primrose and Glencairn Mines. I do not know whether Dr. Mellor ever had the

opportunity of examining a typical occurrence called the West Reef, about 300 ft. S. of No. 1 shaft on the main connection level to No. 2 shaft in the Rand Collieries Mine.

The other foot-wall occurs between the foot-wall of the reef and the slate proper. I should describe it as a green quartzite, in most places having a pronounced fissile structure and in others shading with a greenish white quartzite of more massive structure. Only in the Rand Collieries have I seen well defined gold bearing pebble leaders in it, but the 10 level E. No. 2 shaft State Mines, a well defined pocket of pebbles or channel occurs in it. If our geological enthusiasts and stratigraphists are not content unless they get quartzite under their "main reef" on the East Rand it seems to me they can be easily satisfied by a visit to the mines named.

Mode of Deposition of the Reef in the Eastern Rand.—Will Dr. Mellor kindly tell me how great a vertical depth of strata has probably been denuded from the East Rand area according to his estimate that its original length was 60 miles?

Types of Alluvial Deposits Comparable to the Witwatersrand Beds.—The State of Victoria, in Australia, offers perhaps a wider range of different occurrence of alluvial deposits than any other country. Over a very large area in the neighbourhood of Ballarat occurs a Miocene deposit of cemented gold-bearing wash (pebble beds). This has been very largely denuded, and now occurs only as a capping of isolated hills. It undoubtedly represented a very large deposit, comparable in size to that of the Witwatersrand.

Presence of Gold in Pebbles at Rand Basket.—The specimen I exhibited is apparently not an isolated instance of the occurrence of gold in the quartz of pebbles. It is a significant fact that in the gold-bearing leads of Victoria pebbles showing gold were nearly as great a rarity as they are here. It might be noted in connection with the "shale foot-wall" that in deep-lead mining a soft muddy bottom, say, of decomposed slate is found favourable for gold deposition.

Infiltration Hypothesis.—May I be allowed to use the shelter of Dr. Mellor's ample wings to have another peck at my friend, Mr. T. A. Rickard, who so genially misrepresented my criticism on his paper on the "Persistence of Ore in Depth." To do this, I must quote from my contribution and compare it with what Mr. Rickard and Dr. Mellor say: I wrote "If his placer theory be the true one . . . Mr. Rickard is out of count at once, for in that case unless the line of the deposition of the detritus was from N. to S. (and there is nothing to indicate this) depth *per se* can have no influence on the gold contents of the "reefs." "I did not then know of the evidence presented by Dr. Mellor for a deposition from a NNW. direction. As a matter of fact as indicated by Dr. Mellor any improverishment of the banket beds in depth is due to the smaller thickness of the reefs and the more irregular distribution of the lenses of pebbles. This is only to be expected as the deposit spreads further from its source. I also mentioned that quite possibly in the Central Rand the reefs were narrower in depth. It is, however, most encouraging to find that though the reefs are smaller the gold content is high at the Nigel Mine so many miles on the dip. Mr. Rickard's comment is that "the assertion, that the Rand is unique, is unconvincing to men of wide experience." I also state "there are difficulties in holding the placer theory, but underground experience in the Far Eastern Rand forces one to consider some modification of it. One finds there lying on the slate an isolated patch or pocket of pebbles with no sign that anything leading to it formed a channel for mineralizing solutions, yet there is the rich ore. Poor areas mean pinched reef, or small pebbles, a bare contact or bastard reef. The ore shoots of the Nigel Mine are simply runs of larger pebbles." This is merely a summary of much that Dr. Mellor says.

Possible Influence of Dikes.—I should like again to emphasize what I wrote as to the condition of the Witwatersrand beds, when and if they were mineralized by solutions, as I shall return to this later. I wrote "I am certain the beds were not mineralized in their present position, and they were not mineralized after the enclosing rocks were solidified to quartzites and the pebble beds to conglomerates. In other words, the silicification of the Witwatersrand formation may have accompanied but did not precede mineralization of its ore bodies. It seems to me that in considering problems such as these most extraordinary ideas are held by people who should know better of the penetrating powers of mineralizing solutions in more or less inert, unfractured and solid rocks. It seems to me that a study of ore

bodies formed from ascending waters shows there must be violent fracturing of the rock or the constituents of the rock must be readily attacked by the solutions. It seems also that these conditions did not exist on this field after the rocks were solidified and in their present position. I know portions of the Rietfontein Mine intimately, and Mr. Horwood's theory that the banket beds there were mineralized *in situ* through the agency of the diabase dikes simply amazes me. It is plainly written there, and all over the Rand, just what mineralization the dikes were able to effect. Pyrite in the dikes, and from the dikes, is coarsely cubical and is easily recognised. The vapours and solutions from the dikes have only succeeded in introducing barren quartz veins with perhaps some copper and iron pyrite into the very local fractures, made by the intrusion of the dike or the faulting that preceded it, and also along bedding planes, generally under the reefs. Into the banket beds for a few feet they have introduced galena and zinc blende, also occasionally but for a very limited distance there has been some resolution of gold contents, and in exceptional instances replacements of pebbles, but this is local and limited. They have (at Rietfontein) mineralized and darkened the quartzites and impregnated them with pyrite and sometimes a little gold for a distance of almost 10 ft. or 15 ft. from the contact. It is most plain that the dikes have utterly failed to bring about any general or extensive mineralization of the enclosing rocks, and this is of course exactly what one would expect in these hard and dense formations." Mr. Rickard's comment on this is, "Mr. Horwood's theories may or may not be convincing to Mr. Weston, but unsupported generalizations have little weight against the painstaking research of a real student. Mr. Horwood has contributed to our knowledge. Mr. Weston to our conversation."

It is needless to ask whether Dr. Mellor supports my observations or those of Mr. Horwood and Mr Rickard. I would like to emphasize the point that I do not believe the dikes were responsible for the second generation of pyrite emphasized by Prof. Young.

Secondary Change in the Conglomerates.—Portions of the wash in the deep alluvial deposits of Victoria are full of secondary pyrite, so that it is quite possible a great portion of the original pyrite in the Rand reefs is of similar origin.

Assuming that the gold in the reefs is of detrital origin, I would trace their stages in the history of these beds.

First, the stage when the conglomerates existed as gravel beds overlain by sand and other gravel beds with little or no chemical change going on among the gold and the heavy minerals, like ilmenite, pyrite, iron oxide, etc., that accompanied the gold.

Second, gradual solidification of the beds to the consistency of our present Dwyka conglomerates and the overlying sandstones of the coal measures. These beds have large free spaces between the quartz grains and pebbles and could hold perhaps 20 in volume of water. These beds are gradually covered by others, and the strata subside until the contained water reaches first boiling point and then that of superheated steam with the pressure meanwhile steadily rising. This stage must occur accompanied by active solution, resolution, and reprecipitation of minerals, and the possible introduction of more pyrite from outside sources. When the heat and pressure reach a certain point silica begins to be attached, a period of resilicification sets in, pressure increases and the beds become quartzites and conglomerates, and internal circulation is checked, the water absorbed or driven out, and the beds reach a state we find them in now. I do not think enough stress has been laid on the enormous total quantity of pyrite existing in the quartzites and grits of the upper Witwatersrand. Under the conditions I have described of a saturated deposit transportation of iron minerals would be certain to occur, and there is no need to seek outside sources for any second deposition of pyrite in the beds. The carbon also would come from the distillation of organic matter, which was certain to be present in such a deposit.

Thirdly, the beds are tilted, denuded and fractured and the injection of dikes into the strata bring about the local secondary effects already noted.

There are, however, some points with which the advocates of the placer theory must deal before their case is fully established. They must remember there are two classes of dikes, one basic, and the other acid. Are the acid porphyries older? These contain up to 4 dwt. gold. Are they evidence of an earlier stage of mineralization, which occurred when the beds were shallow and in a permeable form? In the Pretoria series, 15 miles behind the Rand, as at

Schreepoort, near the Magaliesburg, we have true quartz fissure veins or leaders carrying gold. Have these any possible relation to the occurrence of gold in the conglomerates?

The great problem is, however, the existence of the Black Reef gold-bearing conglomerates. Here we have an unconformable and infinitely younger series overlying the Witwatersrand conglomerates, and exhibiting gold-bearing conglomerates very similar in mineralization to them. What is the relation? It it be said that they are simply secondary alluvial deposits formed from the denudation of the Rand banket, how is it they do not contain pebbles made up of lumps of recognisable Rand Banket, as some of the pebbles are large? If on the other hand, the Rand bankets were formed by infiltration, from where came the solutions that mineralized the Black Reef Series? I do hope that Dr. Mellor will have time to fully study this question though, unfortunately, so many of the Black Reef mines are flooded. I do not believe that the final word can be said on the question before this series is properly considered.

Effect of Denudation.—We need to ask what became of the gold denuded from the Rand Reefs, and this gives me the opportunity of again stating that Mr. Rickard has evaded the point I made, that the presence of large alluvial gold deposits does prove that the ore bodies were not formed after the period of denudation, as Mr. Rickard contends, but have been greatly truncated as I contend.

Mr. W. Bleloch *(Member):* It is my intention to contribute to the discussion on Dr. Mellor's paper at the next meeting of the Society in May. Meanwhile, I will just make one or two observations. I will begin by saying it affords me very great pleasure indeed to find in this paper of Dr. Mellor's one explanation of the origin of the Witwatersrand gold, in which I fully agree with him. And I think his paper, as Mr. Marriott has said in his discussion, will go far towards doing away with any fear of the sudden cessation of gold mining on the Rand. After twenty years' experience of prospecting these reefs I am of opinion, that the gold came into the bankets at the same time as the pebbles; it may have been dissolved and re-crystallised out of the solutions, but, as to the origin of the gold, I should say that any practical student of these reefs must come to this conclusion. I never could understand the selective action necessary on the infiltration theory. How it could be contended that gold solutions should pass along the Main Reef Leader and not along the Main Reef—how they could have their gold contents extracted by precipitation along one bed and not along another one close by, was always a stumbling block to me. To my mind Dr. Mellor has very ably dealt with some very important points in the geology of the Witwatersrand system during the past five years, but some of his conclusions I do not agree with, and although these matters are not quite the subject of this present paper, there is one point which Mr. Marriott seems to imply, viz., that Dr. Mellor in his paper thinks that the deeper levels of some of our reefs contain an excess of gold, or gold which has been carried from higher levels down to the deeper levels, and he seems to argue, or imply, that outcrop areas may in this way have been impoverished. I think that is probably wrong. In that area of the Rand which is at present the subject of the keenest observation—the Far East Rand—there are a certain number of mines which are poor; mines round what might be called the fringe of the area; mines such as the Rand Klip, Cloverfield, and some of the mines in the Nigel District. It is quite evident that the percentage of payability of the reef—the Nigel—so far worked in the Nigel District is certainly far lower than the percentage of payability of the reef in the Van Ryn—worked in what is known as the Far East Rand. I would like to say this, we have in our Far East Rand, to-day, ten great mining companies which have well on to thirty million tons of payable ore developed, with a value on an average of about thirty shillings a ton. These ten great companies are all working within what, I venture to say, is the limit of the subcutcrop of the Van Ryn Series proper. We can say, so far as the Van Ryn is concerned, there is not a single failure yet. In that particular area occupied by the reef on which this vast mining industry is now built up, we have four outcrop mines—the New Kleintontein, the Van Ryn, New Modderfontein, and the Modder B. Now, there is no impoverishment of outcrop there. Mr. Marriott's implication is the reverse of what Mr. Rickard said. I do not suppose that there is any finer or better mine in the world than the New Modderfontein. I have not seen any richer ore than is to be found in that particular mine. There the reef outcrops at the surface and goes down at a

moderate angle of dip. So that, as far as Van Ryn Reef is concerned, the outcrop mines are just as good as the deep levels, and I should say that, on the whole, the deep levels will be found to be just as good as the outcrop mines. There is practically no risk of their being otherwise. The establishment, in the course of development, of a definite system of pay ore bearing areas, which is so ably described by Dr. Mellor, is very clear proof—it is only a question of the gold having been deposited in these areas by the natural course of sedimentation, whatever agency it was that deposited the banket deposited the gold. The experience gained in that area shows that approximately the minimum of payable development is 50 , there may be occasionally one or two mines that go 40 for a month or two, but, on the average, I do not think there is any mine that has less than 50 payable ore, some go up to 70 and 80 .

Now, what has possibly caused this idea to be enunciated by Mr. Marriott that round the fringe of this great area there are impoverished areas, may to my mind, be explained in this way. During the course of recent work, I have found very distinct and conclusive evidence that we have two reefs on the Far East Rand, both of which are, as a rule, underlaid by shaley rocks, not always slate; sometimes in the case of the Van Ryn there are bands of quartzite, a matter of 60 ft. thick, betwen the slate and the reef. Even in the case of the Nigel, I have known instances where there are 10 in. or 12 in. of quartzite between the shale and the reef. But what I wish to say most emphatically is that we have two reefs, and these poor mines, on what Mr. Marriott would call the fringe of the area, are, in my opinion, situated on and are working not the Van Ryn Reef but the Nigel and it is to this that we can ascribe the fact that we have poor mines round part of the periphery of the area. As I have said, the percentage of payability on the Nigel Reef, although some very rich areas have been proved, is certainly very much lower than the percentage of payability on the Van Ryn, and I do not agree with Dr. Mellor in his idea that there is a gradual thinning of the Van Ryn Series towards the South East. The Nigel Reef is always thin, more or less, and is so where it is outcropping to the north of the Van Ryn Mine. On the contrary, according to the reading of the geology of that area, of which I have ventured to give a short explanation to-night, the Van Ryn Reef shows no sign of diminishing in thickness in the South-Eastern outcrops in the Nigel District; in fact its thickness is even greater down there than it is in the Benoni-Springs area. The idea seems to rest solely on the identification, erroneous, in my opinion, of the Nigel Reef with the Van Ryn in that district.

The Chairman: In addition to the promise we have received from the last speaker we have also promises from Mr. J. S. Curtis and Mr. Hildick Smith of contributions at the next meeting.

NOTES ON RARE MINERALS IN MADAGASCAR.

By T. P. Waites (Member).

(Printed in Journal, March, 1916.)

DISCUSSION.

The Chairman: Owing to an unfortunate misunderstanding, Mr. Waites was absent from the last meeting, so he could not exhibit his specimens. As we are fortunate enough to have him here to-night, I will call upon him to make a few remarks in connection with the specimens he has to show us, and on his paper.

Mr. T. P. Waites *(Member):* I am very pleased with the reference your Vice-President has just made as to the reason for my absence last month. Although it might appear to you a matter of discourtesy on my part, there was no intention of the sort.

I have brought these specimens with me to-night, which are now reduced to a small compass, on account of so many people nibbling and chipping pieces off them, but, I think, what is left of them, will give the members some idea of their nature. So far as I am concerned, they have ceased to be rare minerals. To give some idea of what I mean by their not being rare, I have laid out here a map of Madagascar to a scale of 1 in 2 millions. This area is about 230,000 square miles, which, I think, is nearly double that of the Transvaal. These minerals are distributed throughout the archæan region, which occupies two-thirds the area of the island. Before my notes were read, I had numerous samples assayed and determined at the Government Laboratories in Tananarive, in Johannesburg, and elsewhere; the samples were also tested for radio-activity by electroscope, electrometer, and radio-photography.

An analysis of greeny-brown waxy crystals by Dr. Moir resulted as follows:—

Uranium trioxide	17·6
Niobium Pentoxide	38·0
Titanium dioxide	15·1
	70·7

Didymium, cerium, and other rare earth oxides form the balance, and a little thorium is probably also present. Iron is completely absent, which is very uncommon in niobate minerals. The mineral was about seven times as radio-active as Swaziland euxenite. The various samples were then exhibited and explained to members by Mr. Waites.) I would like to very roughly and crudely give some idea of some of the pegmatites in the Vavavato hills especially, which rise to an altitude of about 7,000 ft., because it is in these hills that so much of this uranium occurs. (Sketching and illustrating on the blackboard.)

Mr John Watson (*Member of Council*): I should like to ask Mr. Waites if any of these minerals—ores of uranium, etc.—are being worked in Madagascar, or are being treated. Are they, as yet, being worked on a commercial scale?

Mr. T. P. Waites (*Member*): As I stated before, there is no systematic work going on, except on the deposit of uranium phosphate near Antsirabe, where the method of concentration is very crude. The Syndicat Lyonnais has exported considerable quantities of this uranium phosphate. The carnotite of Ejeda is found in cretaceous formation. The deposit of autunite in the Analalava district is found in the cretaceous formation; the autunite of Antsirabe is found in quaternary or recent formation.

Mr John Watson (*Member of Council*): Are the mining laws of the country and their administration good?

Mr. T. P. Waites: No better mining or mineral laws exist anywhere in the world. The surface deposits are not being gathered in a systematic manner. The method of working alluvial gold, if I may use the term "working," and of mining these deposits, in Madagascar, is this: A native goes out and pegs a claim of 2,500 hectares, equivalent to about 2,500 morgen, for £1 a year. He then engages a number of natives, and pays them so much for everything they bring in. With regard to the mining laws, I would again state that no country has more liberal mining laws, nor in any country I have been in would one receive more assistance than one does from the French authorities in Madagascar.

The Chairman: I think we may say that to the investigation of these minerals, we owe the papers which have been written by Dr. Moir, and which appear in the last *Journal*.

The meeting then terminated.

Contributions and Correspondence.

NON-INFLAMMABLE EXPLOSIVES BAGS.

In a recent number of the *Journal* a note appeared by Mr. Pursglove on the subject of non-inflammable bags, and in the table he gives, based on his experiments, a piece of sacking impregnated with zinc sulphate is found to be inflammable.

Some time ago I was asked to do some work of the same kind and for the same purpose. The first step was to consult all available authorities, and for the purpose of this paper I may state that in *Industrial Chemistry*, by Blucher, zinc sulphate is mentioned as a flame proofing agent, while Dr. von Schwartz in "Fire and Explosion Risks" quotes the extensive experiments of P. Lechtin, with forty-four anti-pyrenes, when twelve of these were efficacious and zinc sulphate was seventh on the list. It is, of course, obvious why zinc sulphate was selected by me for experiment—being cheap and abundant around a cyanide plant, and the net result of a considerable amount of work went to show that it is thoroughly efficacious for this particular object, and I am sure Mr. Pursglove will not mind me saying so.

Dr. von Schwartz defines our object thus: "From their nature, all organic substances are combustible, being neither fire- nor flame-proof. They can, however, be endowed with the latter quality, and then be successfully used as a protection against fire since, though they char externally, they do not transmit flame, and therefore the latter is prevented from spreading. The following substances may be rendered flameproof; wood, paper, leather, textiles, fibres, straw, millboard, etc. . . . . A substance or article is fire-proof only when it has been rendered unintlammable throughout its entire mass and not merely on the surface."

For the actual work the filter-press effluent was collected while the acid vat was be-

ing pumped out. The salt was concentrated by evaporation in a shallow iron tray. The sacks were dipped in this and dried, this being repeated till the white crystals were visibly present throughout the fibre. As this dusted off readily the sack was sprayed with a solution of size or glue, re-immersion tending to re-dissolve a lot of the extremely soluble salt. All free acid which is invariably present, must be neutralised before immersing the sacks—I used lime. Experiments were carried out with the sacks actually used, new, and sometimes old sugar pockets. They were fire-proof after treatment. A candle flame would not ignite them, and a red hot cinder from the furnace only charred where it lay—the fire did not spread. The strength of the sack was not impaired, its fire-proof qualities last as long as the sack is likely to, while if the extreme solubility of the salt is kept in mind there should be no more statements that zinc sulphate renders a substance more inflammable.

I have made no experiments with sodium silicate myself, but I may be allowed to quote from Dr. von Schwartz. Referring to calcium sulphate and sodium silicate he says, "These exhibited the following defects: cracking off under the influence of heat, imperfect covering of the surface owing to the crystals failing to melt in the heat of the fire as they should do in order to afford protection against flame; imparting brittleness to the object treated; becoming detached on the surface of the articles being used etc."

ROBERT MURDOCH.

Obituary.

MR. JOHN RUSSELL.

We regret to have to announce the death of one of our members, Mr. John Russell, on Good Friday, after an operation at the Simmer & Jack Hospital. Mr. Russell was born at Northampton, England, and was engaged there in farming. He came to South Africa just after the conclusion of the Boer War and was employed by various commercial firms until joining the mill staff of the Simmer Deep, Ltd., as a learner, on the 1st September, 1908. He joined the Society in June, 1913. He worked his way up through the various grades, and was acting mill shift-boss at the time of his death at the age of 36 years.

The funeral ceremony at the Primrose Cemetery was attended by the President and several members of the Society. The Society desires to extend its sympathy to his widow and family.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

SOUTH AFRICAN SHEEP DIPS.—"An organized protest, headed by the Bradford Chamber of Commerce, was made some time ago against the use of caustic soda and sulphur sheep dips, to the Agricultural Department of the South African Government, on the ground that the use of such dips had a deleterious effect upon the quality of the wool. The use of these products is, however, still officially recommended by the Union Government and a Government report issued recently deals as follows with the objections raised :–

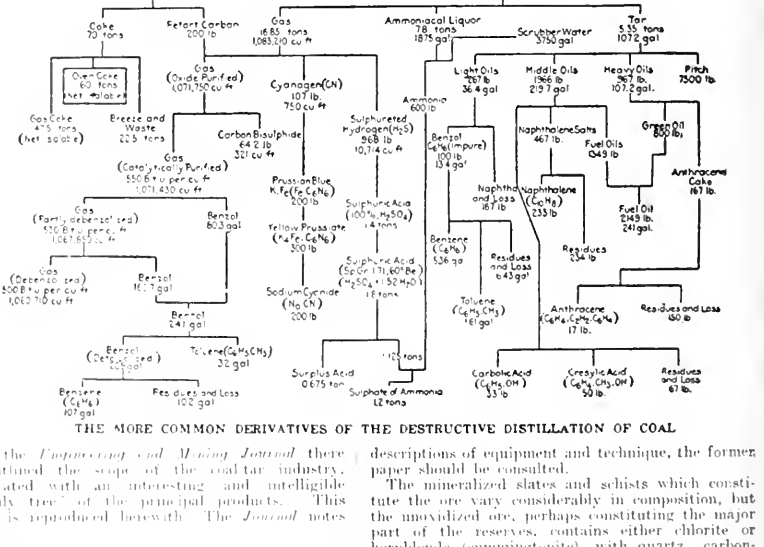
(1) The South African Government is well aware of the nature of sodium sulphide, and of the fact that it is used in removing wool from dead skins; but it is emphatically denied that the caustic soda-sulphur dip, mixed in accordance with official directions, partakes of the nature of a wool solvent, and as proof it may be stated that some of the best prices obtained from South African wools have been made for fleeces which have been dipped in the lime-sulphur or caustic soda-sulphur solutions. Wool buyers continue to purchase, year by year, at top prices, what the Bradford Chamber calls 'dissolved' wool. (2) Wool treated with caustic soda and sulphur in certain proportions will be dissolved, but the Government has made it quite clear that the dipping fluid must be made in proportions which will not damage wool. (3) It is well known that wool is frequently treated with caustic soda or caustic potash in the scouring process, and that if hard water be used for scouring it becomes necessary to soften it artificially by the addition of caustic soda; but the caustic soda or caustic potash would not be used by a manufacturer in proportions which would dissolve the wool. (4) The effect of all alkaline soda salts, when they are used improperly, is to destroy the spinning qualities of the wool, but the manufacturer would find his task an impossible one if he were prohibited from using alkali. The damage caused by the improper use of alkalis is hardly likely to be greater than that caused by the scab mites, for wool from sheep which have suffered from scab is brittle and structureless, and has lost both its spinning and felting qualities. (5) It may be that wool treated with an excess of alkaline matter has a stronger affinity for colouring matters than wool not so treated, but the fact that manufacturers themselves use caustic soda and caustic potash in the course of the manufacture of wool into cloth, is proof conclusive that whilst these substances may be dangerous when applied in a careless manner, they cause no damage to the wool when properly used. Reference to any standard work on the effect of alkalis on wool will show that in some cases caustic soda is used to strengthen the fibre. It all depends on the form in which the solution is made and its degree of alkalinity."—*Chemical Trade Journal*, Aug. 28, 1915.—*Journal of the Society of Chemical Industry*, Sept. 30, 1915, p.977. (J. W.)

By-Products from Coal Tar.—"Increased popular interest has been evidenced in the question of coal tar products since the war cut us off, at least in part and temporarily, from some of the most important of these substances. It is to be hoped that the United States will be able to establish its chemical industries now on a basis that will render it henceforth reasonably independent of foreign supplies. The importance of the coal by-products in the chemical industry and consequently in a wide variety of manufactures cannot be over-emphasized. Dyes, drugs, explosives, fertilizers, wood preservatives, and a host of other products come in whole or in part from coal.

METALLURGY.

Notes on Homestake Metallurgy.—"It is nearly three years since the metallurgy of the Homestake ore was discussed with considerable thoroughness, in a paper read before the Institution of Mining and Metallurgy.

Certain changes have been made in this period which are perhaps of sufficient interest to justify a brief description, and it is chiefly with such details and miscellanies that this paper will deal. In the circumstances, a certain amount of repetition is inevitable, but matter treated of in the former paper will be referred to here only when it is necessary for the sake of coherency. For full



THE MORE COMMON DERIVATIVES OF THE DESTRUCTIVE DISTILLATION OF COAL

In the *Engineering and Mining Journal* there is outlined the scope of the coal tar industry, illustrated with an interesting and intelligible "family tree" of the principal products. This chart is reproduced herewith. The *Journal* notes that

"Tar is an exceedingly complex substance, consisting of a mixture of more than 200 different chemical compounds, and of course varies considerably in quality and quantity per ton of coal distilled, according to the class of coal and the temperature of distillation . . . The primary distillation of one ton of average gas tar will produce approximately . . . Ammoniacal liquor, 5 gallons; crude naphtha, 5·6 gallons; light oils, 26 gallons; creosote oils, 17 gallons; anthracene oils, 38 gallons; pitch, 1 cwt. 1 cwt. 13 cwt. Ed. Com.].

It is stated that during 1915 over a dozen plants were built or nearly completed in this country and Canada for the production of benzol, largely in connection with by-product coke ovens; such is the war demand for this commodity." *Engineering and Mining Journal. The Engineering Magazine*, Feb., 1916, p. 788 (cf. E. C.).

descriptions of equipment and technique, the former paper should be consulted.

The mineralized slates and schists which constitute the ore vary considerably in composition, but the unoxidized ore, perhaps constituting the major part of the reserves, contains either chlorite or hornblende (cummingtonite), with quartz, carbonates of lime, magnesia and iron, and arsenopyrite, pyrite, and pyrrhotite. Ferrous minerals predominate and this fact has been an important factor in determining the metallurgical treatment.

With one or two exceptions the minerals noted are of relatively high specific gravity, and the ore as a whole is exceptionally heavy, many determinations giving an average specific gravity of 3·00.

This high specific gravity presents one decided advantage, when the cost of treatment is compared with operations elsewhere, in that the ton, almost universally the basic unit, represents a volume probably less by 10% than that of many gold ores. On the other hand, this high gravity renders more difficult the discharge of pulp from mortars, its distribution on amalgam tables, and its transportation in launders.

In the same way, as attesting the usual balance

of such variations of different ores, it may be noted that the needle-like fibres of hornblende (cummingtonite), interlacing throughout the mass, render the ore more difficult to crush than would be anticipated, but on the other hand assist in maintaining free leaching in both sand and slime treatment, retaining their characteristic form as far as they can be traced with the microscope.

The metallurgical equipment consists of :

At the South side—3 stamp mills (660 stamps) with 36,356 sq. ft. of amalgam plates; 1 regrinding plant, with independent cone system and 540 sq. ft. of amalgam plates; 4 batteries of cones for classification; 3 clarifying-tank houses; 1 sand plant.

At the North Side—2 stamp mills (360 stamps); 2 tank houses; 2 cone houses; 1 sand plant.

At Deadwood.—1 slime plant, treating the combined slime from south and north sides.

The ore supplied to the north side mills is usually drawn from the upper levels of the mine, and is at least partly oxidized. It is more easily penetrated by cyanide solutions than is the unoxidized ore, and satisfactory extractions are at present made from the sandy portion of the mill tailing without further reduction in tube-mills. In almost all other respects, practice at the two divisions of the works is identical and unless otherwise stated further description will be of operations at the southern branch.

Stamp Milling.—A new mortar has recently replaced the familiar design. In this newer design the old "inside lines," remarkably effective in maintaining a rapid discharge, have been retained. All the front above the screen rest has been cut away, as has also a considerable portion of the back. This has given greater accessibility, with resulting safety to the men when engaged in changing iron and other necessary operations; it permits the use of longer bossheads and simplifies the care of feeders, since an observer at the front of the mortar can determine whether the ore is feeding properly. The slight tendency to splash at the back is corrected by a false back, or apron, of canvas; a small chip tray of 4-mesh screen, resting above the discharge screen, impounds floating particles of wood, which are removed from time to time, washed, and burned. The ash, which assays as high as $300 per ton, is set to accumulate in sufficient quantity and is eventually sold.

The crushing units are not in conformity with modern ideas, and there is no doubt that, were the plants to be built anew, radical changes in design would be made. The Homestake, unlike many younger mines, has developed gradually from comparatively small beginnings. The mills represent the growth of 30 years. Built before the cyanide process was known, they then represented advanced milling practice. It must be conceded that the stamp duty of 4½ tons, with 80% of the tailing passing a 100-mesh screen and 60% passing a 200-mesh screen, is fairly good even when compared with the results reported from many newer installations.

We find that heavier stamps crush more rock, with little or no change in the sizing of the tailing unless more open screens are used; in which case, even without increasing the falling weight, tonnage may be gained at the sacrifice of sizing. To us it appears that compactness of plant, with the saving both in first cost and, later, in labour and supervision, constitutes the leading claim of the heavy stamp for preferment. Indeed, if the gold metallurgist is ready to dispense with amalgamation—and we are by no means prepared to do this—it is more than probable that he will do well to investigate the crushing practice of modern copper mills before he commits himself to the heavy gravity stamp. Alaska, at the moment, holds more of interest than does Africa.

It may be pertinent to note that I have never encountered a Homestake mill tailing, no matter how far advanced in secondary treatment, from which some free gold could not be recovered by laboratory amalgamation tests.

Regrinding.—Operations at the regrinding plant offer little of interest, the practice conforming closely to the usual methods.

The feed to the tube-mills is already so fine (only 25% remains on a 50-mesh screen) that the efficiency of the mills is low. The critical size of sand delivered to the sand plants is about 100 mesh for unoxidized ore, and 80 mesh for oxidized ore. It is advantageous to operate the mills in closed circuit, but it is difficult to do this without permitting sulphide particles which have been sufficiently reduced in size, to remain in the circuit. By introducing a double baffle or trough classifier, containing a hydraulic device, into the slime-overflow end of a Dorr classifier, this difficulty has been in a measure overcome, and two of the mills are at present using this system.

Sand Treatment.—The sand is leached with cyanide solution in vats 44 ft. in diameter by 9 ft. deep, holding 610 tons of sand each. The operations are distinguished by unusual care in the preparation of the material for extraction, rather than in the extraction itself.

Classification and aeration are the two essentials to successful work. The latter, necessary to overcome the tendency of the ferrous compounds in the ore to remove the vital oxygen from the solutions, is achieved by forcing, at intervals, air under slight pressure into the false bottom, below the filter canvas. The sand charge is suitably drained before this air is applied; the pressure is so adjusted that the air is forced into the charge, yet is so low that the column of sand is nowhere broken or disturbed. To secure this immunity thorough classification is essential, and this has always been recognized as a matter of first importance.

The action of the air is interesting. Laboratory tests indicate that, on an average, about 75 cub. ft. of oxygen is absorbed by a ton of ore, before its reducing action is corrected. If this air is not supplied from some extraneous source, the solutions are vitiated and extraction ceases. When air is applied and is followed by a water wash, calcium thiosulphate appears in the effluent solutions; when the air is followed by cyanide, the effluents contain sulphocyanides and free cyanide appears only after some time. After a limited time of leaching, the extractive power of the solution decreases and further aeration of the charge is necessary. Each aeration is attended by the formation of some acid and the consequent destruction of some lime and cyanide. After treatment, the sand shows no sign of oxidation; neither analytical nor microscopic examination can detect differences between charge and residue.

Lime, crushed in a one-stamp mill to pass a 7-mesh screen, is added to the pulp stream as it flows to the vats.

It had long been recognised that a high protec-

tive alkalinity was detrimental, but until within the past two years it had been tacitly accepted that it would be impossible to dispense entirely—or practically so—with protective alkali. Eventually this was done; extractions improved and consumption of cyanide was materially reduced.

The gratifying results are in a measure due to another change in procedure, initiated at about the same time. Solutions of two strengths are used; that stronger in cyanide is used earliest in the treatment of a charge. When these solutions appear as effluents, one portion is precipitated, the other brought to full working strength by adding cyanide and thereafter returned to the extraction of another charge of sand.

This custom is by no means unusual, yet wherever it is used the procedure seems to be to maintain the stronger solution at a fixed content of cyanide and to allow the weaker to vary in strength according as the varying losses in treatment may determine. This is wrong in principle. The critical strength is that below which the weaker solution will no longer dissolve its quota of gold, or, if dissolving it, will no longer freely yield it to precipitation. If the strength is greater than this, we may confidently expect some cyanide to be destroyed without giving compensatory service. In such circumstances, if the maximum working strength is reduced, the weaker solution will be automatically restored to the lowest economic strength; if it falls below this, a temporary increase in the maximum strength will restore it. In other words, cyanide should be added to the strong solution in accord with the determinations of strength made on the weak solutions. Thus, operating with a strong solution of variable strength and a weak solution also variable, but never far from its effective limit, it may be anticipated that little cyanide will be needlessly expended.

Slime Treatment.—The slime plant is equipped with 30 Merrill presses, each of 90 frames, 4 ft. by 6 ft. by 4 in. One press holds 26 tons of slime, and has a capacity of about 70 tons a day.

Treatment is in principle identical with that of the sand, periods of aeration and treatment alternating with sufficient frequency to maintain the activity of the solutions. The alkalinity is carried at a very low point, and the working solutions are of low cyanide strength. Within the restrictions imposed by practice, there seems to be no solution so weak that it will not freely extract gold from the aerated slime. The problem seems to demand the passing of a definite volume of solution through the slime cake, rather than treating with solution of a definite strength or for a definite time. So long as this amount of solution is passed, the extraction is accomplished. Thus the treatment time is a function of the leaching rate.

The minimum working strength is determined by our ability to precipitate the effluent solution, and in developing this minimum much of the treatment is given with very dilute solutions, with the result that the consumption of cyanide has been reduced to an extremely low figure. For instance, a 10-day period, concluding as this is written, gives the following:

Sodium Cyanide (128) consumed per ton of slime treated: In treatment, 0·127; added for precipitation, 0·008; total, 0·135.

"The precipitation of the 'low solution'" (first and final effluents, low in both gold and cyanide) is difficult; doubly so on account of the low alkalinity; and after trial of many expedients to assist it we are not able to improve upon Carter's old system of a drip of strong solution.

This, usually added with the feed of zinc dust at the rate of 1 lb. per hour (or per 25 tons of solution precipitated) constitutes from 5% to 10% of the cyanide used. An excessive amount of zinc dust is also necessary to ensure clean precipitation of these solutions, and this entails further expense, not merely in the first cost of zinc, but in the refining of the larger bulk of low-grade precipitate. Combined, these costs consume about 25% of the profit won through the reduction of cyanide consumption, a satisfactory enough result in a plant where treatment methods have been so well established as is the case here, even though the impression may be conveyed that one branch of the work has lost efficiency.

Unit consumptions per ton of slime treated—Sodium cyanide, 0·161 lb.; zinc dust, 0·118 lb.; lime, 3·84 lb.; hydrochloric acid, 0·393 lb., costing $0.0093 per ton treated; power, 1·15 kw. hr.

Precipitation.—The Merrill system of precipitation with zinc dust is used at all cyanide plants, as is the Merrill filter press of triangular section. This method has been fully discussed in the paper already cited, and the technique remains about as there described. It has been mentioned that the effect of the recent variations in treatment has been adverse to precipitation, since the tonnage of solution requiring precipitatoin has been increased at the same time that the content of the solution in gold and cyanide has been reduced. Beyond the natural effects—increased zinc consumption and lower-grade precipitate—no difficulties have been encountered.

Double cloths are used on the frames, the outer one being taken off when the press is cleaned and replaced by the inner cloth, which is in turn replaced by a new cloth. The outer cloth is burned, the ash going to refining.

Experiments are still in progress to find the lightest and cheapest cloth that can be used for this purpose without unduly increasing the hazard of leakage.

All washing of cloths is eliminated by this system. Against the added cost of cloth may be set the advantage of security from leaks due to rotten or wrinkled cloths and the labour of washing soiled cloths.

Refining.—In its essentials, this remains as described in the earlier paper, except that after acid treatment of the lower-grade precipitate, the briquets are charged to the blast furnace instead of being fused in the cupel. The latter treatment is still the standard for precipitate of higher grade.

Acid treatment is a preliminary to furnace treatment of all precipitates. It may not be necessary, but in my judgment it is sound metallurgy, since it enhances security from stack losses and insures more fluid slags. Moreover, the decreased bulk of product to be smelted with lead decreases the time required for that operation, always more or less hazardous to the health of the workmen.

Oil is now used as a fuel in the cupel furnace, replacing wood. It is an ideal fuel for the purpose, and its use has resulted in an acceleration of nearly 50% in the rate of cupelling."—A. J. Clark. *Canadian Mining Journal*, July 1, 1915, p. 405. (H. A. W.)

CYANIDING APPARATUS.—"A cyaniding vat has the form of an inverted frustum of a cone and contains a partition of similar but flatter form: the inner and outer compartments communicate at the top by a narrow annular space, and at the bottom by a circular channel which contains a screw propeller to cause an upward movement of pulp in the outer, and a downward movement in the inner, compartment. The latter has an apron extending inwards from the upper annular aperture and inclined downwards towards the centre."—F. E. CARMAN, Jamaica, N.Y., Assignor to M. E. Carman, Babylon, N.Y. U.S. Pat. 1,146,783, July 20, 1915. Date of appl., Oct. 22, 1912. Renewed Dec. 22, 1914.—*Journ. Soc. Chem. Ind.*, Sept. 15, 1915, p. 911. (J. A. W.)

MINING.

SAMPLING AN ERRATIC ORE-BODY.—Estimating the value of an ore-body in which the ore is distributed according to no ascertainable rule involves problems which are much in dispute. To those who would discard mine sampling for a mill test it is suggested that the large sample sent to the mill may be entirely unrepresentative. The latter is the critical factor and it is believed that widespread careful hand work gives a more representative sample than does mining for a mill test. Confirmatory results are quoted.

"In your issue of February last there appeared an interesting article by Mr Morton Webber on 'Latent Errors in Mine Sampling.' In this article Mr. Webber classifies the type of erratic ore occurrence, in which the valuable mineral occurs in bunches, under the general heading 'Mines where sampling is no use,' and recommends the mill test as the only reliable method of valuing such mines. To quote: 'The only method of reliable valuation in a deposit of this type would be based on authentic record of past yield, in combination with competently selected representative shipments of the remaining ore.'

With this I cannot altogether agree. Authentic record of past yield is of the utmost value, but I believe that this is much better studied in combination with proper sampling than with mill tests. I do not refer to those tests made to determine metallurgical treatment, or to those made over an extended period of time to confirm sampling results, and which amount to operating the mine under option, but to the mill tests designed to take the place of sampling because considered more reliable—'representative shipments.' These practically amount to large samples, and my contention is that they are no more justified in an erratic mine than in a uniform one; rather less so, if anything, for the chances of error are greater, and the safety resulting from averaging a large number of samples is needed. Before discussing this further, I shall describe briefly the methods used and results obtained at one mine of this type that I had the privilege of sampling and studying in great detail for a period of two years. This mine forms an excellent illustration of this type of deposit, and of some of the principles that affect the accuracy of the results when sampling by the ordinary hammer and moil method.

The formation of the mine in question consisted of highly metamorphosed sediments and igneous rocks, carrying irregularly scattered lenses of gold-bearing quartz ranging from almost microscopic size to several feet in thickness. These schists without quartz would assay from nil to about 3 dwt., averaging less than 1 dwt., and the quartz from nil to several hundred pennyweight. Frequently the gold was in coarse flakes, making most brilliant showings. The quartz stringers, in combination with the richer portion of the schists, formed erratic ore zones separated by comparatively large zones of waste, and showed no consistency, taken either singly or in groups of any size. There was no habit or condition that could be discovered affecting the sudden changes from spectacular ore to practically barren waste. Mr. Webber mentions levels 30 ft. apart giving erratic results. I remember one cross-cut, sampled by a continues channel along each wall, of which, over a distance of 200 ft., one wall averaged one-third of the other. These walls were perhaps 7 ft. apart. The troublesome nature of the ore was evidenced wherever samples were used: in the assay office, duplicates would not check at all unless ground to 200-mesh; in the mill, samples of battery pulp over a month would not check amalgam plus cyanide heads. The amount of underground development was hardly greater than would have been necessary to stope a uniform deposit of the same size, and very different from the network that Mr. Webber states to be necessary for deposits of this type. Levels about 80 ft. apart, and drifts and cross-cuts about 150 ft. and 75 ft., with of course local modifications for all of these, is not a large development proportion. In fact, per ton of ore blocked, it is small. According to my opinion these conditions formed as extreme an example of Mr. Webber's classification and as difficult a problem as one would be likely to run across. And yet this mine was sampled and ore blocked out, and is being sampled to-day. The results are of the utmost value, both to the men financing the property and to the management planning exploration and development work. It had to be sampled. There was no other method of working it intelligently.

The method of sampling did not differ in any essential from that in general use by a large number of engineers, so a brief description of this should suffice to make it clear. As neither the quartz stringers nor the ore zones had any definite trend, samples could have been taken in almost any direction. For ease in sampling all channels were cut horizontally about breast high along the sides of the drifts and cross-cuts. (A distinction was made between drifts and cross-cuts according to their relations to the stopes, but structurally they were the same). Both sides were sampled, as so little was known about the deposit at first that all the information obtainable was needed. These channels were the usual kind cut with a hammer and moil, and yielded about 15 lb. per ft. The rock surface was first cleaned with a scrubbing brush, and the channel, about 3½ in. wide, outlined with chalk. The samples were caught in an ordinary iron meat pan about 12×18×2 in. This was more easily cleaned than a candle box or canvas, and made possible a more detailed examination of the sample at it was cut. In the low-grade or barren schists, samples were taken up to 60 in. long, but in ground which was apparently rich no sample was taken more than 30 in. long. This was in uniform rock. With every change in the character of the ground the sample was bagged and a new one started. In small quartz stringers this would frequently give samples not over 3 in. long. In such cases the

size of the channel was always increased to make the total weight of sample not less than 2 lb. Quartz containing the coarsest gold was not avoided in any way, but everything within the outline taken as mechanically as possible. Any adjustment necessary was made in the office afterward.

The question of high assays was of course always coming up. The first step in their treatment was re-sampling. All places yielding erratic assays were re-sampled in the bottom of the original channel. The decision as to whether an assay was erratic or not did not depend so much upon its tenor as upon that of its neighbours. An assay of 10 dwt. in one place might mean a re-sample and in another one of 50 dwt. be let go. The two or three assays of the same place thus obtained were averaged and the resultant tenor entered in the records with no reduction. At the times of calculating the ore reserves, when studying the relation of each sample to all the others, it occasionally seemed advisable to cut down an assay to a tenor more consistent with the apparent structural importance of the rich spot yielding the sample. In addition to this, an arbitrary high figure of 500 dwt. was adopted, to which all assays above that figure were reduced. There were so few assays higher than 500 dwt. that it was thought they would affect the average out of proportion to the probabilities of their being duplicated in the interior of the blocks. These two causes combined affected a negligible proportion of the assays probably less than one-tenth of 1% — and it is merely to show the methods of work that they are mentioned here.

In connection with the re-sampling an interesting fact was observed: the majority of times the re-sample gave a lower result than the original. I think the reason for this is clear. The occurrences of rich specimen ore were on the average so small that it was probable a large proportion of such an occurrence would be cut away by the offending sample. Similarly, I have no doubt that re-samples of the promising looking ground which had returned low assays would have given on the average higher results than the original. The averaging of the two assays thus introduced a slight factor of safety.

It will be evident from what I have said regarding the size of the rich zones that any attempt to mine them separately would have been absurd. The only feasible project was to take all within the mineralized zone. Therefore, in valuing ore blocked by the sampling results, the ground was considered as a low grade mass, and no attempt was made to segregate tonnages for the small rich zones. This resulted in a method much the same as would have been used for a more uniform deposit. After the few adjustments of the high assays had been made as described above, the ground was divided into areas as indicated by the horizontal workings, and each area valued by dividing the total inch-pennyweights of the periphery, high and low alike, by the total sample length in inches. The blocks between levels were valued by weighting the tenor at each level by the area blocked. Rises in general were used merely to prove continuity of ore, although occasionally their tenor was averaged with that of the levels. The final result was adjusted according to knowledge of the mine. Bore-hole results were of great assistance in this, although used entirely for information as to continuity and not for obtaining the tenor; and as mining progressed the samples from the development were supplemented by mail samples from the stopes. The latter were averaged with the development samples when practicable.

The results of this method were highly gratifying. For a period of two years, the tenor of the total ore mined as indicated by mail samples, when compared to amalgam plus cyanide heads for a quarter of a million tons milled, indicated a sampling factor of 112%. I regret I am not at liberty to give figures in greater detail, but I think these will illustrate my point. It may be contended that these results indicate nothing, because the work was done by the mine staff, who brought their intimate knowledge of the mine to the final interpretation of the results. This was undoubtedly a factor, but it was a factor dependent upon experience and not upon any lessening of the difficulty of the work. Any independent engineer with sufficient experience could have arrived at the same results. In fact, in the early history of the mine, an examination was made by an independent engineer who predicted almost exactly what would occur.

The above example illustrates certain principles that I think can be applied in a greater or less degree when sampling any erratic deposit. Although it is an axiom in mine valuation that every mine must be judged on its merits, according to the experience of the examining engineer, and no hard and fast rules can be laid down, still I think these principles form a good foundation for working, and are certainly of more general application than considering sampling utterly unreliable and not using it for the data that it can give. I shall try to state these principles as I see them, omitting the actual manipulation of the sample, precautions against the increased danger of salting oneself, and details of that nature as being sufficiently taken up in the short description of actual practice above.

How to average high assays is apparently the question that bothers the majority of men most. There is a vast difference between the occasional high assay in ore of essentially uniform tenor, and the condition where the recurrence of very high erratic assays makes up almost the entire value of the ore. In the latter case I belive the result will in general be nearer the truth if high assays are reduced very sparingly, and then only for extremely good reason; assuming of course that there is sufficient development work to allow the law of averages to work. A high assay represents the tenor at a spot sampled. Whether that ore extends to the next development working or not is immaterial; the assumption is that it will be duplicated by other small patches within the blocks. Erratic and variable though the rich intersections may be over short distances or in workings close together, when taken over sufficient length of workings the average will represent, within the limits of error to be decided by the engineer, what the entire mass of ore blocked out will yield. The theory is exactly the same as that which governs uniform deposits, but engineers seem to fear to apply it to the type under discussion. The high assays in the mine I have described were not sufficiently frequent to consider them the usual thing in fact quite distressingly infrequent at times and yet an attempt to cut them down by any of the well vouched for methods would have resulted in indicating on paper merely a large mass of waste. I am not at all sure, from the mine's subsequent history, that even the slightest

reductions I described were justified. When sampling such a deposit, samples should be cut impartially through rich and poor alike; the mine studied structurally, and when the assays are in, all the information correlated before making up the averages. Then if for any reason some assays should be reduced, well and good; but an assay should not be reduced without knowing why it is done, merely on account of some vague idea that it is good practice and playing safe.

Too much discussed methods for increasing the accuracy of the results on such a deposit are decreasing the sampling interval and cutting the ore into small blocks. I do not believe that either of these is exactly the remedy. What is desired in sampling is to obtain sections of the ore-body which are representative of its tenor as a whole. It is obvious that if the development openings are not thus representative, no amount of close sampling will increase the accuracy of the final result; it will merely increase the accuracy of the tenor assigned to those development workings. Therefore the sampling interval need only be small enough to give the tenor of the workings to a degree of accuracy consistent with their representation of the mines as a whole. Decreasing the interval below this point adds nothing. As for cutting the ore into small blocks, this will increase the accuracy of the estimate for each block, and therefore for the entire developed portion of the mine; but is that exactly what we are after? In most cases the maximum amount of development that can be done is more or less a fixed quantity: compressor capacity, shaft or tunnel capacity, and cost limitations all contribute to this. The development that is used to cut already developed ore into smaller blocks is not exploring the mine for extension; and it is the results of this exploration that more than compensate the engineer for the lessened accuracy of each single block. To put it differently: the smaller the blocks, the more nearly accurate is the tenor assigned to the ore thus outlined; but if we grant that this ore is only a small portion of the whole, then the larger the blocks, that is, the more the development is spread out, the more representative of the whole this development will be. There must be enough samples for the law of averages to work for the high assays to represent what will, on the average, be found in the interiors of the blocks. As shown above, obtaining this number by decreasing the sampling interval avails nothing; the samples must be from representative development. Therefore the accuracy of the result when sampling an erratic mine depends most upon the total amount of development in the ore, not the amount per ton, and the blocks need not be smaller than would be required for mining.

The question of valuing by sampling a mine where the values are like 'plums in a pudding,' as it has been so aptly phrased, compared to the mine where values are evenly distributed, is not one of impossibility, but of relative accuracy. The error will be larger than with uniform deposit, but as long as the probable limits of error are known, the results will often be as useful as if they were more accurate. Within these limits of error, by working according to the principles outlined above, ore can be blocked out as in the uniform deposit. However bad conditions may be, they will affect the mill test more than they will sampling. Certainly I know of no method of making a mill test on the mine I have described that would have yielded information at all comparable for accuracy with that obtained by sampling. Two years' actual production gave no indication of what the tenor of the mine as a whole would be; yet the current sampling indicated it after the mill had been running six months. It was over two years after a certain tenor had been predicted by sampling before the mill results began to conform to that tenor. The reason for this is perfectly obvious: the stoping was not done from representative places. *Representative* — that is the crucial point. Unless a man is clairvoyant, there is no method of obtaining representative shipments. And if he could select the proper places, a few holes one way or the other by the machine men would entirely alter the tenor of the test. It comes right down to the old question of a few very large samples versus an enormous number of small ones — a question that has been many times discussed. Obviously, I am an advocate of small samples. It is hard for me to see any advantage in a few large ones that can compensate for the effect of averaging a quantity of small ones taken with machine-like impartiality all over the mine.

It seems to me that often such stress is laid upon great precision in mine sampling and valuation that sight is lost of larger issues that are more vital. Some of these have been outlined above, such as spending time and money on work whose sole object is to gain greater precision in the tenor that is assigned to the ore, rather than spending it on exploratory work that would give much needed information as to the future of the mine. And certainly this striving for great precision has no more serious consequences than when it prevents an engineer from giving any opinion as to the value of a mine, because its character is such that the results cannot be accurate within the limits of error he has assigned himself. The erratic mine is the most difficult problem the examining engineer has to face, but that does not alter the fact that it must, upon occasion, be faced. To reduce high assays sweepingly for no reason except custom, or to refuse to place any reliance upon sampling through an excess of conservatism, is to sidestep the issue. The best opinion obtainable on the value of this type of mine is that honestly given by an experienced engineer without undue conservatism or optimism, after an adequate study of the mine structurally, a thorough sampling, and an analysis and correlation of the assay returns with the structural data obtained, and past yield if obtainable. To this opinion an engineer's clients are entitled." L. A. Parsons.—*Mining Magazine*, Sep., 1915. Vol. xiii., No. 3, p. 151. (J.E.T.)

MISCELLANEOUS.

"Natalite," A New Alcohol Fuel. "A certificate has been published as a result of a test by the Royal Automobile Club of a new motor fuel known as "Natalite." The fuel is reported to consist of alcohol and ether, about 40% of ether being present. The fuel consumption in an official trial was at the rate of 16·4 miles per gallon, or 26·2 ton miles per gallon. The engine started easily on all occasions either from cold or when warm. The valve caps and heads were found to be very clean after the trial. "Natalite" is wholly produced in an alcohol distillery from alcohol crops or their by-products.

The official trial does not give any comparative data, but an opportunity has been afforded *The Engineer* of testing this fuel against petrol, as a

result of which they are led to the conclusion that the fuel consumption in the two instances is almost, if not quite, identical. The only alteration necessary to get the best results with "Natalite" is a slight reduction in the supply of air to the carburetter. It is impossible to detect any difference in the running of an ordinary petrol engine when "Natalite" is used, and the exhaust is clear. The fuel appears to have the capacity of actually cleaning an engine in which deposit has formed, and has no tendency towards corrosion of the metal of the cylinders or other parts. It is intended as the first instance to manufacture the fuel in quantity in South Africa from molasses. In addition to alcohol and ether, the fuel contains at least a trace of ammonia for the purpose of neutralising any tendency towards acidity which might develop." *Chem. Trade Journ.*, June 19, 1915. *Journ. Soc. Chem. Ind.*, 15th July, 1915, p. 705. (J. W.)

STRENGTH OF TREATED STRUCTURAL TIMBERS.—"The United States department of agriculture has issued as Bulletin No. 286, the results of a series of tests made by the forest service, in cooperation with the Illinois Central R. R. and one eastern and two western wood-preserving companies, to determine how the strength of bridge stringers is affected by commercial creosote treatments. To do this, comparison was made between the strength of treated and untreated stringers of the same size and quality. The test timbers selected were loblolly pine, longleaf pine, and Douglas fir, which were taken from regular stock, in the form of sticks 8 in. × 16 in. in section and from 28 ft. to 32 ft. in length. From each of these were cut two eight foot stringers, which were tested with and without treatment, in bending, by supporting them at the ends and applying the load at two points located one-third of the span from each of the supports, corresponding closely to conditions of practice. Two methods of treatment were used, the "boiling" and the "steaming" processes. The deductions made from these tests were: (1) Timber may be very materially weakened by preservative processes. (2) Creosote in itself does not appear to weaken timber. (3) A preservative process which will seriously injure one timber may have little or no effect on the strength of another. (4) A comparison of the effect of a preservative process on the strength of different species should not be made, unless it is the common or best adapted process for all species compared. (5) The same treatment given to a timber of a particular species may have a different effect upon different pieces of that species, depending upon the form of the timber used, its size, and its condition when treated." *Ind. Engineering*, Dec. 4, 1915, p. 317. (J. A. W.)

Reviews.

THE BIRTH-TIME OF THE WORLD, AND OTHER SCIENTIFIC ESSAYS. By J. Joly, M.A., Sc.D., F.R.S. Published by T. Fisher Unwin. Price, 10s. 6d. net.

This interesting volume has recently arrived. The first article, from which the title is taken, was given as a lecture before the Royal Dublin Society.

From the amount of sodium salts in the ocean, the author calculates the geological age of the world at somewhere between 80 and 100 millions of years. His results have been checked by Sollas, F. W. Clarke and others. This book is illustrated with 28 photographic plates and numerous figures, and contains articles on: Denudation, mountain genesis, alpine structure, pleochroic haloes, the use of radium in medicine, and other kindred subjects. Some of the photos of mountain scenery are very beautiful and the type used in printing is large and clear. A highly interesting and original work. (J.W.)

THE THEORY AND PRACTICE OF ORE DRESSING. By EDWARD S. WYARD. Pages 420. Ill., index. McGraw-Hill Book Co., New York. Price $4.

"The art of ore-dressing has developed so rapidly and has experienced so many fundamental changes in the last few years that an up-to-date treatise on the subject as applied at metalliferous mines in Western America, written from the standpoint of the author's practical experience as a mill superintendent and designer of mills in various districts of the Western United States and Mexico, makes a timely addition to the technical literature. The author acknowledges the benefit received from personal contact with Prof. Richards as a teacher and is appreciative of the immense amount of information contained in Richards' encyclopedic volume. The arrangement of the subject matter is novel and pleasing. The book makes a good impression from the first and appeals to one as a business-like treatise on Western practice. The many working drawings are a distinct feature; the illustrations are good, clear, and not too greatly reduced in scale. The book will appeal to the mining engineer, to the metallurgist, and to the student. In the hands of an experienced teacher it should be useful as a text book. What little working theory has been developed to date on ore dressing is stated briefly but clearly. A single volume of 400 pages necessarily imposes limitations and the author has selected wisely without over-stepping the domain of metallurgy. Amalgamation, therefore, is not mentioned and stamps have received scant consideration. The author is to be complimented on having produced on the whole a fairly well balanced book with an unusual proportion of valuable new information." *The Mining and Scientific Press*, Sept. 25, 1915. (W. A. C.)

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

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STOKES, Major R., now Assistant Inspector of Mines, General Headquarters, British Expeditionary Force.

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Proceedings
AT
Ordinary General Meeting, May 20, 1916.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, May 20th, 1916, Mr. J. E. Thomas (President) in the Chair. There were also present:—

17 Members: Messrs. J. Gray, H. A. White, Prof. J. S. Cellier, E. H. Croghan, F. W. Watson, John Watson, E. M. Weston, A. Whitby, Prof. G. H. Stanley, J. R. Williams (Members of Council), E. J. Hastings, C. E. Hutton, J. A. Jones, G. Melvill, P. T. Morrisby, S. Newton and J. J. R. Smythe.

2 Associates and Students: Messrs. J. Ingram and F. J. Pooler.

6 Visitors and Fred Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General Meeting, as printed in the April *Journal*, were confirmed.

GENERAL BUSINESS.

The Secretary: I had been under the impression that Mr. W. Nicklin, one of our members was on the Nourse Mines, but we have just received a letter from him stating he is now a Second Lieutenant in G. Company, 2nd Battalion, Special Brigade, R.E., B.E.F., France. He states they are not allowed to state the nature of their work, that he is in the best of health and thoroughly enjoying himself, and wishes me to convey his best wishes to all C.M. and M. friends. Lieut. Nicklin even in his new position does not desire to sever his connection with the Society.

ANNUAL ELECTION OF OFFICERS AND COUNCIL.

The Secretary: I wish to draw the attention of members to the fact that the nomination form accompanied the last agenda paper. I have to receive the nominations for Officers and Council not later than the 25th May, and their election will be notified at the Annual Meeting, which will be held on the 24th June, at the usual time and place.

APPOINTMENT OF SCRUTINEERS FOR ANNUAL BALLOT.

The President: There is one more item that is, the appointment of scrutineers for the ballot of Officers and Council.

Mr. H. A. White *(Vice-President)*: There has been no opportunity of asking the gentlemen whom I am going to propose, if they are willing to serve. If any drop out, we shall have to rely on those who are left. I propose the following —

Messrs. A. Thomas, S. Newton, P. T. Morrisby, E. A. Osterloh, H. R. Adam, F. J. Pooler.

Mr. John Watson *(Member of Council)*: I second those names. I think most of these gentlemen have acted in a similar capacity before.

The scrutineers as nominated were unanimously elected.

MINERS' PHTHISIS.

Mr. E. M. Weston *(Member of Council)*: I believe the subject of miners' phthisis is to be brought before Parliament very shortly and, no doubt, a lot of stress will be laid on the necessity of destroying the tubercle germ in the mines themselves. It seems to me, from observation, that the chief source of infection down below lies in the vicinity of the boxes of the miners, many of whom are already infected with the tubercle germ. Here the miner sits and expectorates. I think that a thorough spraying with disinfectant in the vicinity of all these boxes every week would be most efficacious. It would involve practically no expense to the mines, because they already have a sanitary

staff. I do not know that one need suggest legislation in this direction, but I think this idea should be borne in mind, and that something should be done in the way I have indicated.

THE RECOVERY OF ZINC FROM ZINC GOLD SLIME.

Messrs. R. Fiander *(Member)* **and R. Murdoch** *(Associate)*. Now that stores have, owing to the present war, advanced in price, working methods which under normal conditions would be economically impossible might be highly profitable.

Zinc is an instance. Taking the average consumption of zinc on the Rand at 0·3 of a lb. per ton treated, and the tonnage at 25 million this gives 7½ million lb. of zinc used. Assuming that 25% of this is consumed in the acid vat, or 1,875,000 lb. and put its value at 5d. per lb., zinc at the value of £39,062 a year is at present being run to waste.

Efforts have been made to recover the dissolved acid vat zinc as marketable carbonate. Yet it is known that the zinc can be distilled direct and recovered as metal and what is known as "blue powder." Prof. Prister did some work on these lines, *i.e.*, distilling zinc from the box product direct, many years ago but the writers have been unable to learn any particulars whatever of his efforts, successful or otherwise. Zinc silver crusts from the Parkes' process are, however, regularly distilled in furnaces of varying types for the recovery of the zinc, and the writers have therefore experimented on these lines.

Experiments made with dirty waste zinc showed that the distillation itself is readily carried out and that the fine state of division as filiform favours rapid and complete distillation well under white heat. All authorities are agreed that the loss of gold by volatilization is practically nil, and the much reduced residue could readily be pot smelted or put through the pan furnace. The blue powder or zinc fume formed could be used by the mines having Merrill's process, any surplus being put back into the retorts with the next clean-up. This is done by zinc ore smelters who use those retorts at the lowest heat for the purpose. The zinc fume resulting from the writers' experiments was much cleaner, and obviously better than that usually supplied by zinc ore smelters.

It is obviously one thing to improvize a retort for a pound or two of zinc shavings and another to put through 1,000 lb. from a clean-up, but is it impossible? Information available regarding retorts deals entirely with those made for treating zinc ores where a white heat has to be maintained for a number of hours. The length of cylindrical retorts is about 4 ft. with an inside diameter of 8 in., and with walls ¾ in to 1½ in. thick. The muffle type again, which can be supported along its whole length (as indeed could the cylindrical, if need be), varies in length from 4 ft. to 7 ft., in width 7 in. to 8 in., and in height 18 in. to 24 in. The conditions governing the construction of retorts are, of course, the possession of sufficient strength while remaining as thin as possible for the penetration of heat, the ingredients to be as refractory as can be and of such density as to be as nearly impermeable to the zinc vapour as possible. The suitability of fireclay for any particular purpose is said by one authority to be a matter of experiment, but when one sees the assay pots made in this country one can be sanguine, and the necessary experience and machinery is certainly not lacking, for large cylindrical drain pipes are regularly made here.

Given the retorts or muffles, then a suitable type of furnace to give the necessary heat is easily evolved (indeed an experimental retort could be fixed in an ordinary pan furnace without interfering with its regular work), and the preparation of the filiform zinc for distillation, because of its bulky nature, would necessitate some form of press, such for example as that used for compressing zinc silver crusts, or even the modest amalgam press, so that three cylindrical 4 ft. retorts or two muffles 4 ft. long by 18 in. high should take the bi-monthly clean-up from a 100 stamp plant.

The large tilting furnaces of the Faber du Faur type as used in Australia take large graphite crucibles made in England, and are manipulated by expensive and elaborate gearing.

No facts and figures capable of being applied to our conditions on the Rand could be found in any authorities consulted. Nothing but experiment will give us that. The writers, however, think it is well worth while discussing at least, even if the result is the pricking of their *ballon d'essai*.

They are of opinion that the necessary retorts can be made in the country, that the rolling into sheets is not impossible when the metallic zinc is distilled, and that furnaces can readily be built or adapted, while

the messy and expensive acid treatment will be saved.

Mr. J. R. Williams *(Past-President)*: I would just like to mention, as one who has had a very large experience in connection with distillation of zinc, that many years ago I was in charge of a zinc plant. I think the writers of this paper would find it practically impossible to get the last 2 or 3 of zinc out of their gold. In South Wales, the tips, that is to say, the scrappings from the retorts, amount now to many thousands of tons, and go anything from 10 to 15 of zinc. It is practically impossible to get bullion sufficiently free from zinc, and I am afraid that would not do for work on the Rand, particularly if the zinc slime happened to carry anything like a perceptible amount of iron. It seems the zinc is then formed into a kind of ferrite of zinc, and distillation is well nigh impossible. I only mention this as a suggestion that difficulty might be found in getting the bullion fine enough, admitting that all the rest of the paper is in order.

ON SOME DISEASES OF THE RESPIRATORY ORGANS INCIDENTAL TO MINERS, AS PORTRAYED BY DR. AGRICOLA, IN A.D. 1550.

By J. de Fenton, Ph.D. (Librarian, Seymour Memorial Library).

I have two main objects in penning these notes:—

(a) The first is to point out that centuries ago, lesions of the lung due to particles of dust,—in other words, silicosis, or miners' phthisis,—were known to exist among certain classes of mine workers, and that the description given in Dr. Agricola's work definitely connotes that disease.

(b) My second object is to demonstrate the absolute necessity of thoroughly mastering a language it is proposed to translate.

We are perhaps rather inclined nowadays to imagine that anything we happen to notice for the first time is new.

Now the wisest man of all (1) has laid it down in everlasting script, that "there is no new thing under the sun." And this paper will show that miners' phthisis is by no means the new disease it might be thought to be. It has a history extending certainly over 300 years.

There is no need for me to elaborate the distinguishing features of miners' phthisis. Under the different names of miners' phthisis, silicosis, pneumonoconiosis, etc., this fell disease has acquired a wide notoriety and prominence in connection with the Rand; though, even here, we are far from equalling the unenviable experience of Agricola's day. For, says he (2) "in some mines one finds women who have been married to seven husbands, all of whom this terrible disease has carried away by a premature death."

> Hinc in metallis Carpathi montis inveniuntur mulieres quae septem viris nupserunt, quos omnes dira illa tabes immatura morte affecit.

This paper will, of necessity, be largely historical and philological; but for all that it should be interesting. Most, if not all our knowledge has come to us from speech, and it is on our mother's knees that we learned the wisdom of the Ages; and so it is but fitting that other and later knowledge should accrue to us through the medium of tongues, be they never so outlandish and foreign.

Far back in the ages, we get glimpses of a disease special to miners; thus, Hippocrates (3), in his "Epidemics," speaks of the metal digger (*ὁ ἐκ μεταλλων*) who breathes with difficulty, (*πνευματώδης*) and is of a pale, wan complexion.

Pliny the Elder (4), in his "Natural History," speaks of the use of respirators to avoid dust inhalation.

> Qui minium in officinis poliunt, faciem laxis vesicis illigant, ne in respirando pernicialem pulverem trahant et tamen per illas spectent.
>
> Those employed in the works preparing vermilion (5), cover their face with loose bladder skins that they may not inhale the pernicious powder, and yet may see through the skin.

And in a book on "Mines and Mine Workers," published in 1699, Loehniss (6) has the following:

> The dust and stones fall upon the lungs, so that the men have lung disease, breathe with difficulty and at last take consumption.

Further, Diembrock found in men's lungs such heaps of sand, that in running the knife through the pulmonary vesicles, he thought that he was cutting some sandy body (6). This, by the way, is the very

experience of the author who, in the early days of the Rand, chipped many a razor blade while endeavouring to cut sections from dust-congested lungs.

But it is not till we reach the days of Agricola that we get what may be called a real practical diagnosis of miners' phthisis.

As a modern writer (W. S. Lilly) has it, human nature loves biography. "We are curious—it is a sort of instinct—for information as to what manner of men the most highly gifted of our race have been. Besides, personality stands for so much, that to properly appreciate the bearing and value of a man's words one should know something of his antecedents. A knowledge of the person helps one better to appreciate his sayings and to attribute a truer value to his expressions; therefore a short sketch of of Dr. Agricola's life will perhaps be an illuminating commentary on his works.

Georgius Agricola (the latinized form of Georg Bauer), known as the "Father of Mineralogy," was born at Glauchau, 24th March, 1494, and died at Chemnitz, 21st November, 1555, at the age of 62 years. Thus, he was born the year after Luther; the great Columbus had just returned from his first voyage to America; and Vasco da Gama was to round the Cape of Good Hope only three years later. Thus the revival of learning with its train of Humanism, the Reformation with its stimulation of exploration and the reawakening of the arts and sciences, was still in its infancy with Agricola.

Gifted with precocious intellect, young Agricola graduated Bachelor of Arts in 1517, and at the age of 21 was appointed principal of the Municipal School of Zwickau. Two years later he proceeded to the University of Leipzig, where he studied physics, chemistry and medicine; and he took his degree of M.D. at the world-renowned University of Padua.

On his return, after meeting Erasmus (?) at Basle, he settled in 1526 as practising physician at Joachimsthal in Bohemia, a centre of mining and smelting work, partly to fill the gaps in his study of the art of heading, and partly to test what had been written about mineralogy, by careful observation of ores and the methods of their treatment. Here, according to his own statement (?), he spent all the time not required by his medical duties in visiting the mines and the smelters, in reading up in the Greek and Latin authors all references to mining, and in associating with the most learned of the mining folk.

1528, he published his first systematic treatise on metallurgy (?). This he named "Bermannus" after a miner, one Lorenz Berman, whom Agricola idealised as the "learned-miner." For this work, Erasmus, who was then at Freiberg, wrote a eulogistic preface (?).

In 1530, Agricola was appointed historiographer to Prince Maurice of Saxony, who further gave him an annual allowance. The doctor then moved to Chemnitz, a centre of mining activities, where in addition to his mining duties, he was appointed town physician; later he became Burgomaster (?), being re-elected four successive years. He published a number of works on geology and mining and metallurgical processes.

His most famous work, "De Re Metallica" (?), was published at Basle, by Froben (Frobenius) (?), in 1556, the year after the doctor's death, though the dedication is dated 1550. From contemporary documents we gather that the writing of this work extended over a period of 20 years (?).

Agricola's masterpiece was published in Latin, German and Italian; and it is interesting to note that the Italian edition is dedicated to our own Queen Elizabeth (?). That it passed through some ten editions in three languages at a period when the printing of such a volume was no ordinary undertaking, is in itself sufficient evidence of the importance in which it was held, and is a record that no other volume on the same subject has since equalled.

Until four years ago, however, no translation into the English tongue appears to have been made; but in the year 1912, one was published by Mr. and Mrs. Hoover (?).

As far as the mining and metallurgical portions of the work are concerned, I am not in a position to judge of the correctness or otherwise of the translation. In the medical references however, I think that the translation offered leaves something to be desired; in fact, one may apply to it the "punning" saying of the Italians, *traduttio e tradittio* (translation is treason), or as the "Daily Times" (?), a journal issued by the Engineering and Mining Congress of America, has it:—

> "The language you wrote in was horribly pied;
> Your German was low and your Latin was side;
> But Mrs. Hoover and Hoo put it all over you
> When they invented your book all anew.
>
> O Agricola! O Agricola!"

Diseases of the respiratory organs, as referred to by Dr. Agricola, may be conveni-

ently classified under two heads, namely:—

(1) Those diseases which arise from stagnant air, or want of ventilation.

(2) Those which are caused by the corrosive action of dust on the lungs of the miner [18].

With regard to the first class I have found four references in Dr. Agricola. These references I am quoting in full, with the translations and sundry remarks.

(*a*).—. . . . quod metallorum fossores interimantur, modo ab aere pestifero, quem spiritu ducunt; modo . . . . [19].

. . . . . because the metal miners are liable to be killed, sometimes by the pestilential air which they take in by breathing; sometimes . . . . .

(*b*).—Tum difficultatem anhelitus parit aer immobilis, manens tam in puteo quam in cuniculo; cui malo remedia sunt machinae spiritales, quas paulo ante exposui [20].

Stagnant air, remaining either in a shaft or in a tunnel, produces a difficulty in breathing. The remedies for this evil are the ventilating machines which I have explained previously.

(*c*).—Cum puteus fuerit valde profundus, ad quem nullus cuniculus, nullave fossa latens ex altero puteo pertinet aut cuniculus admodum longus, ad quem nullus puteus pertingit, *tunc aer, quod exhauriri non possit, fossoribus offunditur, atque difficulter spirant.* Interdum etiam suffocantur: ardentes quoque lucernae extinguuntur. Itaque opus est machinis, quas etiam Graeci πνευματικα. Latini *spiritales* appellarent; etsi vocem non mittunt; nam ipsae efficiunt ut fossores ex facili spirare et opus institutum perficere possint [21].

If a shaft be very deep, and no tunnel or drive from another shaft reaches to it: or, if a tunnel be of great length, and no shaft connects with it, then the *air does not replenish itself. In such a case it weighs heavily on the miners, and they breathe with difficulty.* Sometimes indeed they are suffocated: and burning lights are extinguished. And so, one needs those machines, which both Greek and Latin call "breathing," though they give forth no sound: for they enable the miners both to breathe easily and to complete the work in hand.

In this last quotation I think that the italicised sentences should read:—

Then the air, because it cannot be diluted, is heavy, and sinks down (or is poured down), into the miners and they breathe with difficulty.

The crucial word here is "*offunditur*" and the reference is evidently to gases such as CO_2.

(*d*).—Aer vero manet immobilis, etiam tam in cuniculo quam in puteo. In puteo quidem profundo, si solitarius fuerit, hoc est, si nec ad eum cuniculus pertineat, neque cum alio puteo sit fossa latente conjunctus: in cuniculo vero si longius in montem actus fuerit, nec ullus puteus adeo fuerit depressus, ut eum attingat. In neutris enim aeris motiones et commutationes fieri possunt: qua de causa halitus fiunt graves et nebulae similes, ac situm testudinis vel subterraneae alicujus cellae, multos annos undique conclusae instar, redolentes; quocirca fossores in his diu laborem, etsi fodinam abundaverit argento vel auro, non sustinent: aut si sustinuerint, anhelitum trahere libere nequeunt, *et capitis dolores habent;* quod magis accidit si laboraverint in ipsis multi, multasque lucernas, quae tunc languidum lumen eis praebent, adhibuerint. Halitus namque quos tam lucernae quam homines exspirant, alios halitus graviores efficiunt [22].

Air indeed remains stagnant both in tunnels and in shafts. This will occur in a deep shaft if it be by itself, *i.e.*, if no tunnel connects with it, and it is unconnected with any other shaft by a drive. It will occur in a tunnel if it has been driven too far into the hillside, and no shaft has been dug deep enough to connect with it. In neither case can there be either motion or circulation of the air. For this reason the vapours become heavy and like mist, and smell of mouldiness like a vault, or some underground cellar, which has been entirely closed for many years. Wherefore the miners are unable to continue work in these places for long (even though the digging may abound in gold and silver); or if they do, they are unable to breathe freely, and suffer from headaches. (Does this perhaps refer to "gassing"?) This happens the more often if they work in such places many together and employ many lamps, which then supply them with but a feeble light. For, the vapours which both men and lamps give out produce in their turn vapours which make the previous atmosphere fouler still.

2. With reference to the second class, namely, those diseases whose causation and inception may with justice be assigned to the action of dust, I have two quotations from our author.

(*a*).—Contra vero aliquae fodinae adeo siccae sunt ut prorsus aqua careant: quae aridatas majus etiam malum dat operariis: siquidem pulvis qui cietur et agitatur fossoribus, penetrans in asperam usque arteriam et pulmones, parit difficultatem anhelitus, et vitium, quod ἀσθμα Graeci nominant. Quod si vim corrodendi habuerit, *pulmones exulcerat et tabem ingenit in corporibus;* hinc in metallis Carpathi montis inventae sunt mulieres, quae septem viris nupserunt, quos omnes dira illa tabes immatura morte affecit [23].

On the other hand, some diggings are so dry that they are entirely deficient in water; and this dryness produces even greater harm for the workmen. For the dust which is stirred up and put in motion by digging, penetrating right into the windpipe and lungs engenders a difficulty in breathing and the illness which the Greeks call asthma. And if the dust happen to have corrosive powers *it eats away the lungs and engenders consumption in the body.* Hence in the mines of the Carpathian Mountains, one finds women who

> have been married to seven husbands, all of whom this terrible disease has carried away by a premature death.

(*b*). Tum dicunt periculosum esse metallicae operam dare: quod metallorum fossores interimantur, modo ab acre pestifero, quem spiritu ducunt: modo *horribiles pulveris pulmones exulceratos, unde extabescunt*: modo intereant *ruinis* montium oppressi: nunc vero, de scalis in puteo delapsi, brachia, crura, cervices frangunt: nullum autem utilitatis fructum tanti aestimari debere, ut propter eius magnitudinem, *salus* hominis et vita, addicatur in maximum quodque periculum et extremum discrimen (2).

Hoover's translation reads as follows: —

> The critics say further that mining is a perilous occupation to pursue, because the miners *are sometimes killed* by the pestilential air which they breathe, *sometimes their lungs rot away;* sometimes the men *perish by being crushed in masses of coal;* sometimes, falling from the ladders into the shafts, they break their arms, legs or necks: and it is added there is no compensation which should be thought great enough to equalise the extreme dangers to *safety* and life (1).

Now it was with a special view to this extract that I have written this paper. The translation of this is really so inaccurate that one feels sure it can only have been intended to convey a general meaning; and yet, as we shall see, Agricola's phrasing is most pregnant, and pithily and accurately conveys to our mind the nature of the disease intended to be expressed. Remembering that the writer was a medical man, with some 20 years' experience as a mine doctor, it behoves us to pay the greatest attention to the text; for a technical man is not inclined to waste words, but intends that each word, nay, in Latin, each syllable should carry its full weight.

Collis, in his Milroy lecture, 1915, "On Pneumonoconioses," has adopted the Hoovers' translation; so, I believe, have the British Commissioners on Miners' Phthisis; but, as we shall see, this rendering fails to convey the striking technical accuracy of the original diagnosis furnished by the old Chemnitz physician.

Many years ago, I learnt the lesson of never trusting to translations or quotations, and of endeavouring, in every instance to trace and verify the original. It is not so much a matter of "much ado about nothing" or inherent carelessness, as laziness, and the general tendency we have of "saying in a few words of an author's opinion more or less the sense of the same."

In the present instance

(*a*) I understand the use of the conjunctive to be special in this case, so that the meaning is *not* that "miners *are* sometimes killed," but that they "*are liable* to be killed," and so on through the whole quotation. I am aware that some may query this; but know it is right.

(*b*).—With regard to the second italicised passage, the published translation says:— "Sometimes their lungs rot away," leaving out altogether the reason or cause why they *should* "rot away," whereas Dr. Agricola gives explicit details. The passage should read:—

> At times they (*i.e.*, the miners) are liable to waste away and become emaciated, breathing in a dust which completely eats away the lungs.

Now, the point to note is, that the published translation mentions nothing about either "dust" or "emaciation."

The important word in the Latin text is "*exulceratum*." Now "*ulcus*" means a sore, an ulcer, and "ἕλκος" in the Greek always implies suppuration; hence, a festering wound. "*Ex*" adds an idea of intensity, finality, exhaustion; hence an action carried out to the very end.

The words "*pulverem pulmones exulcerantem*" would thus connote in the text:

(*a*) A dust causing lesions to the lungs;

(*b*) Which lesions suppurate;

(*c*) Until exhaustion of the subject matter.

To the lay mind this appears a very good delineation of miners' phthisis, especially if due regard be paid to the next phrase, "*macie extabescunt*." "*Tabes*" is always used in Latin to denote a "wasting-away," "consumption," phthisis.

Further, in connection with the present text, I might say with regard to the words "*ruinis*" and "*salus*," that "*ruina*" (from ruo, I fall down) means a "falling down" or a "sliding down," and could not possibly be rendered as "mass." In the text the meaning is, of course, "*fall of rocks*." In the same way "*salus*" never meant "*safety*," but "*health*"—as in the ordinary salutation, "*salve*"—or in the word "*salutary*." Compare "hail, health, whole, wholly, holy."

NB. (1 Except where specially mentioned, all translations given in this paper are my own, *i.e.*, Hoover's ground work adjusted by the writer.

2 The Hoovers translated the 1556 ed. My remarks are based on the 4th ed., 1657. As, however, Agricola died in 1553, the date of the editions all of which were published

after his death can make no difference to the text.

(1) Solomon. Ecclesiastes, chap. 1, verse 9.

(2) Agricola, De Re Metallica. Ed. 1657, Basel (lib. 1, fol. 3).

(3) Œuvres complètes d'Hippocrate, avec le texte grec en regard, par E. Littré. Vol. v., 1846, p. 167. J. B. Baillière, Paris.

(4) Caji Plinii Secundi Historiæ Naturalis, lib. xxxiii., sec. xi.

(5) Bericht van Bergwercken, durch G. E. Loehniss; folio; 1690, p. 56. Stockholm und Hamburg.

(6) A Treatise of the Diseases of Tradesmen, written in London, by Bernard Ramazzini, Professor of Physick at Padua, and now done in English by Dr. James, London, 1705, p. 103.

(7) Erasmus was at that time superintending the issue by the Froben Press, of his own works, and the series of the Latin and Greek Fathers, together with the New Testament in Greek, which was, later on, used by Luther for his translation.

(8) Agricola. De Veteribus and Novis Metallis Preface.

(9) Bermannus (1st ed.), Basel, 1530.

(10) I understand that to this day, in Germany, a miner and his work are designated by the words "Bermann," "Bergmann," and "Bergwerk."

(11) Baumgarten Crusius, Georgii Fabricii Chemnicensis Epistolæ ad W. Meurerum et Alios Aequales, Leipzig, 1845, p. 26.

(12) The four Latin Editions of this work are:—

(a) Froben, Basel, folio, 1556.

(b) Froben, Basel, folio, 1561.

(c) Ludwig König, Basel, folio, 1621.

(d) Emanuel König, Basel, folio, 1657.

(13) Froben (or Frobenius), Joannes (c. 1460-1527), German printer and scholar, was born at Hammelburg, in Bavaria, about the year 1460. After completing his University career at Basel, where he made the acquaintance of the famous printer Auerbach (1443-1513), he established a printing house in that city about 1491, and this soon obtained a European reputation for accuracy and for taste. He was on terms of friendship with Erasmus, who for some years superintended his publications. He also employed Hans Holbein (the celebrated artist) to illuminate his texts.

(14) Briefe an Desiderius Erasmus von Rotterdam . . . . xxvii. Beiheft zum Zentralblatt fur Bibliothekswesen, Leipzig, 1904, p. 27.

(15) Georgius Agricola, De Re Metallica, by H. C. and L. H. Hoover, *The Mining Magazine*, London, 1912.

(16) Ibid.

(17) The Daily Fume, I. 4, 17-2-16.

(18) A practical miner on the Rand told me (after the reading of this paper) that he thought, from his own experience, that miners' phthisis resulted from these causes combined.

Conversing with a Native Labour Recruiting Agent, I gathered that natives very often developed miners' phthisis as late as two years after leaving the Witwatersrand.

(19) De Re Metallica, 1657, lib. i., fol. 3.

(20) Ibid, 1657, lib. vi., fol. 172.

(21) Ibid, 1657, lib. vi., fol. 159.

(22) Ibid, 1657, lib. v., fol. 81.

(23) Ibid, 1657, lib. vi., fol. 172.

(24) Ibid, 1657, lib. i., fol. 3.

(25) Hoover's translation, 1912, lib. i., fol. 6.

The President: In proposing a hearty vote of thanks to Dr. de Fenton for his paper, we are glad to know that the history of miners' phthisis reaches so far back. The only thought that occurs to one is that our medical and mining people must have been very lax in not following it up sooner. I trust that some of our scholastic members who have kept up their proficiency in Latin, not only the literal translation of it, but the rendering of it into our language in order to explain Dr. Agricola's meaning, will take this paper up and raise some discussion on it.

Prof. J. S. Cellier (*Member of Council*): I have much pleasure in seconding the vote of thanks to Dr. de Fenton, more especially as most of us who are employed in mining, have not had sufficient time to keep up our Latin and Greek.

The vote of thanks was carried by acclamation.

THE CHIEF SOURCES OF ACCIDENTS IN WITWATERSRAND MINES.

By Chas. E. Hutton, M.Inst.M.M.,
Inspector of Mines (Member).

(*Printed in Journal, November, 1915.*)

DISCUSSION.

Mr J. R Williams (*Past-President*): I do not think Mr. Hutton has put one of his points quite as clearly as he would like to have done.

He shows us very clearly that the want of proper detonators was responsible for something like 147 deaths and a large number of other serious accidents last year. Reading between the lines, he seems to be

of opinion that the detonator used on Witwatersrand mines is not strong enough. He has not said so in so many words, however, but I say definitely that the detonator, as used to-day, is not strong enough to do the work expected of it. In early days on this field it was rare to use more than, say, two sticks of dynamite in a hole.

In those days we used the No. 6 detonator; in many cases a No. 6 detonator is used now, notwithstanding that as many as nine sticks and upwards of blasting gelatine, an inch and a quarter in diameter are put into one hole.

I think it is asking too much that a No. 6 detonator should give complete detonation of nine sticks of blasting gelatine. I know that many of our best mining people have said, and have held it down as an absolute fact, that if one stick of dynamite is exploded, that it will explode the next, and the next, and the next, and so on. I am sorry to say I must differ. I will just illustrate to you, on the black-board, one or two experiments conducted by myself. A manager of a dynamite works who also held it down as a fact visited Johannesburg some years ago. I told him he was wrong; and made it my business to prove he was wrong. This experiment I conducted on the top of the Ferreira Deep dam. Sticks of dynamite were laid, touching each other, for a distance of 40 ft., or upwards. I really forget the exact measurement now, but we used a couple of boxes of blasting gelatine. We inserted a detonator at the end, and a piece of fuse and lit it. It went off in the ordinary way, right through the 40 ft., apparently confirming his view, but what really happened was this: (drawing sketch on black board showing hole widest at detonator end). Had the explosion taken place perfectly all the way, there would have been a parallel hole, instead of which, this was the shape of the hole. Had the dynamite gone off equally all the way, it would not have had the "cheroot" shape. Furthermore, this experiment was made with freshly manufactured dynamite, and I am afraid that some experiments which have been made with freshly manufactured dynamite have been certainly very misleading.

I will now mention another experiment which was conducted. Two sticks of dynamite, or rather blasting gelatine, were placed with a gap the width of a match box between with a detonator in one only and fired as usual. The result was a perfect cavity of a shape indicating that the explosion was instantaneous and, so, perfectly even.

Now, precisely the same experiment was conducted with two sticks of dynamite, which were about five months old. The result was what I call a figure "8." Carrying the thing further, using explosives which were old—decidedly old, in many cases—what happened was that the second stick of dynamite was blown a very long distance away. In some cases, it was blown up in the air and came down alight, burning. In other cases, the stick of dynamite, instead of having its usual shape, was just mushroomed all over; in other words, was shot all to pieces, and had not exploded. I am mentioning this to show you that it is not fair for people to say: "I have proved that a No. 6 detonator will put off 9 sticks of dynamite," though it will very often when the stuff is perfectly fresh.

During the Anglo-Boer War we were afraid that we might be short of explosives in this country, and the Mining Industry made preparation by getting a ship load of explosives, which was landed in either Port Elizabeth or East London. Well, the Boer War lasted very much longer than we thought it was going to last. The result was that when the explosives were brought up here, they were universally condemned: no miner would use them, they only went off partially. I investigated the matter and found that the explosive had been prepared from absolutely pure chemicals, the percentage of nitro-glycerine to gun cotton was pretty near the mark, or might have been a ½ too much gun cotton; it withstood all the Board of Trade tests, and so on; and was passed as first class gelatine. By experimenting with it, I found, by the use of the No. 8 detonator, it gave perfect detonation; and I may say that every case of that dynamite that was so strongly condemned at first was used on this field, and always proved first-class stuff.

In connection with the first Miners' Phthisis Commission appointed on the Witwatersrand, very shortly after the British occupation, I argued this point of detonators with the then State Mining Engineer and one or two mining experts. They contended that the No. 6 detonator was big enough to do the work. I maintained it was not. There were two drives from a crosscut; one going east, and one west. We had the same number of holes drilled, by rock drill, in each. We weighed the

quantity of dynamite in each hole, east and west. The eastern lot were exploded first, using a No. 8 detonator, and, I may say, we used nine sticks of blasting gelatine in each hole. I went into this drive two minutes afterwards and took samples of the gases, and I felt perfectly certain I could stay there for ever, because, as was proved afterwards by analysis, complete detonation had taken place. There had been no carbon monoxide in the gas.

In the other one, No. 6 detonators were used, and I went in as soon as I could and took samples of the gases there. I did manage to walk back as far as the cross-cut with my samples, had to be assisted to the platform, had to remain there a very long time before I could be brought to the surface: and it is not my fault that I did not die of gassing. The sample I took from there, carried a very large amount—I forget the exact amount, now—of carbon monoxide; showing that detonation had not been nearly perfect, and certainly nothing like it was at the other place. I felt the effects of this carbon monoxide poisoning for a couple of days. Although there were no explosives found in the bottom of the holes there is no doubt in my own mind that a large amount of the explosive must have burnt off.

I should like to ask anyone interested in the question to visit the sorting tables. It is a very common thing to see as much as half a paraffin case of unexploded gelatine. I cannot help saying "Fools rush in where angels fear to tread," when the No. 6 detonator is expected to do the work with nine or more sticks of gelatine in the hole.

Mr. Hutton has very rightly brought up the question of tamping between the sticks of dynamite. There, he is perfectly right. Air spaces, I know, are bad, but water spaces are not.

My opinion, after reading Mr. Hutton's paper through carefully many times, is this: I think it behoves the Government to legislate. I go so far as to say that, in no mines where rock drills are used, would I allow a smaller detonator than No. 8 to be used in the mine. What is the cost on a 200-stamp mill proposition, crushing, say, a thousand tons of rock per day: it will cost roughly, £30 more if the company give the detonators to the contractors at the same price as they do the No. 6. £30—that is on the debit side. What would be found on the credit side? I think a very substantial profit. Every man in that mine would actually be doing more work without knowing it. The air would be comparatively pure, because I do not consider that carbon dioxide should be considered more than a diluent. I have lived in very large quantities of it, without any deleterious effect. I think it is criminal, in the face of Mr. Hutton's paper, to allow a No. 6 detonator to be used in a mine where there are rock drills.

Mr. F. J. Pooler (*Associate*) In connection with this matter, I might mention that, a few weeks ago a member of this Society was at a lecture, not under the auspices of this Society, but as part of the Safety-First Movement, where a well-known lecturer was referring to the question both of detonators and fuses, and amongst other things was laying stress on the necessity for using the most reliable fuse that could be obtained, pointing out, undoubtedly, that accidents, as has been shown in the paper, do arise from negligence in these cases. As soon as the lecture was over, a head official on the particular property where the lecture had been given, jumped up and said he was very glad the lecturer had raised this point, because he wanted to point out that his property not being a very highly paying concern—that is, of course, all a matter of opinion—they had really to consider the cost of such small things as fuses, and that, if they used the fuse the lecturer recommended, the costs would go up to quite a considerable amount. The lecturer replied that it would make a difference of twopence or a penny a coil. A day or two afterwards, this same mining official came down to that part of the property where the member of this Society—a contractor—was working, and asked what fuses he was using. He told him. He said: "That is very expensive." "Why do you use it, it is costing a penny farthing more than it ought to?" I would not have told you of this incident, had not Mr. Williams given me the lead. The contractor pointed out that he had a number of men under his charge, and that the lives of most of them were probably worth more than a penny farthing, and getting rather angry, he told him, if, by using poorer fuses on his mines there was any accident which could be traced directly to that, he believed, by the law of the land, he ought to be charged with murder.

THE CONGLOMERATES OF THE WITWATERSRAND.

By E. T. Mellor, D.Sc. (Lond.), M.Inst.M.M.

(*Printed in Journal, February, 1916.*)

DISCUSSION.

Prof. R. B. Young, D.Sc. (*Visitor*): Dr. Mellor's two main topics in this paper are (1) the conditions of deposition of the gold-bearing conglomerates and (2) the origin of the gold in the beds. I experience some difficulty in accepting his views regarding the first subject.

In the first place, if the Main Reef Leader and certain other beds in the Main Reef Series were laid down rapidly through the agency of violent floods we should expect the direction in which the floods moved to be indicated by the mode of imbrication or overlapping of the pebbles in the beds. However, as far as I have observed, the imbrication of the pebbles is irregular and confused as in the case of ordinary marine gravels and shingles, in which the arrangement of the pebbles is largely due to the action of waves and the currents generated by them. I think that Dr. Mellor will find that the storm-water deposits he refers to on p. 156 are characterised by regular imbrication.

Secondly, the shapes of a large proportion of the pebbles are such as are characteristic of marine gravels and shingles, viz., rounded but with one prominent flat face.

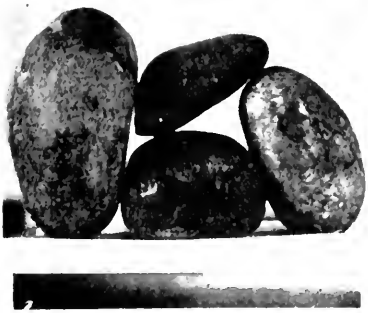

Fig. 1. Boulders from the banket of the Main Reef Series, to show the form which they almost invariably have. Their size is indicated by the footrule underneath.

This is especially noticeable among the larger pebbles and the occasional boulders that one meets with in the banket. Pebbles that have been rounded by being driven along the channel of a stream tend to be more regular in form owing to their motion being in most part rotatory, whereas under marine conditions the motion is more liable to be oscillatory or to and fro, and this has the effect of accentuating the flatness of the side on which the pebble or boulder rests. If this is the case, we should hardly expect that very many pebbles formed in the course of a deltaic river would have this peculiar muffin-like shape, though they might quite well have been formed along the shores of a wide estuary, where the conditions are very similar to those prevailing along the sea-coast.

Fig. 2. Large thin section of banket photographed by transmitted light, to show the shapes of the pebbles. Diam. ×7·12.

Both of these features of the pebbles of the banket that I have just referred to were pointed out twenty years ago by Becker in his well-known paper* on the Witwatersrand conglomerates.

In the third place there is a peculiar occurrence in the Ferreira Deep Gold Mine

*8th Annual Report, United States Geol. Survey, 1896-7, pp. 160-161.

which has a bearing on this question. I shall quote from a description of this in one of my papers in the Transactions of the Geol. Soc. of S.A.*

"At a point on the 10th level drive the Main Reef Leader is seen to bulge abruptly into the foot-wall, presenting on each side of the drive an appearance resembling a section through a small contemporaneously eroded channel. Among the pebbles contained are numerous ones composed of schist. . . . . The schistose pebbles vary in size from 4 or 5 inches in longest diameter downwards. In colour they show various shades of brown, grey and green. They are almost all of them comparatively soft, and many of them show a tendency to fall away in small or large flakes. Though doubtless their friability is in great part due to changes they have undergone subsequent to their inclusion in the rock, they must from the beginning have been much softer that the normal pebbles of the banket. It may have been that they were transported from the land more or less directly to their present position by a current or stream, and thus escaped the wear and tear to which the more evenly distributed pebbles were subjected. In composition the pebbles vary from sericite schist to chloritoid schist, and some of them show a marked resemblance to rocks from the Swaziland System in the Barberton district and at Mount Mavé."

This occurrence, to my mind, is clear proof that at no very great distance from it there existed a source of soft schist pebbles, and if these could be carried out in considerable numbers to this position by a small current, why is it that they were not deposited in noticeable numbers among the vein-quartz and other hard pebbles by the violent flood which Dr. Mellor pictures?

One of Dr. Mellor's arguments for the rapid deposition of the Main Reef Leader by a flood is its persistence as an individual bed over a great area. At the same time he states† that in some localities the Kimberley Reefs, which he apparently regards as ordinary marine gravels laid down at no great distance from a shore line, exhibit a similar persistence over a distance of a mile or more. Does he ascribe the persistence of the reefs in such localities also to floods, though of less extent? In my opinion it is more likely that the persistent portions of the Kimberley Reefs are due to the working over by the waves of contusedly bedded gravel beds originally deposited by currents, and a similar explanation can be offered for the more marked persistence of the Main Reef Leader and other beds in the Main Reef Series.

The various points that I have referred to above as difficulties that have suggested themselves to me when reading over Dr Mellor's paper might be elaborated at greater length, but perhaps I have explained them sufficiently to enable Dr. Mellor to deal with them.

With regard to the second topic of Dr. Mellor's paper, viz., the origin of the gold, I agree with his main conclusions, though I am surprised that he has not given prominence to the bearing on the question of the banded pyritic quartzite.

This rock occurs in large bodies associated with the Main Reef Group of conglomerates in the Meyer and Charlton, Wolhuter and other mines. It consists of quartzite through which run numerous pyritic bands of varying thickness. The bands have obviously had a sedimentary origin and on microscopic examination prove to be identical in composition to the matrix of the banket. They contain heavy detrital minerals such as chromite and zircon, and have undergone the same mineralogical changes as the matrix of

Fig. 3. Large thin section of banded pyritic quartzite photographed by transmitted light. The dark bands are pyritic. Diam. $\frac{2}{3}$.

*Vol. XV., 1912, p. 86.
†This *Journal*, Feb. 1916, p. 148.

the banket. Occasionally pebbles appear in the bands and then the rock merges into normal banket.

Now these pyritic bands are auriferous and, in the mines just mentioned, sufficiently so for the quartzite to be mined. Yet originally they could not possibly have been more permeable to auriferous solutions than the barren quartzite in which they lie. No supporter of that form of the infiltration theory which depends on the greater permeability of the conglomerate beds as compared with the beds among which they lie, has ever attempted to explain the origin of the gold in the banded pyritic quartzite, which rock, in my opinion, constitutes an unanswerable argument against the theory.

In conclusion, I wish to express my admiration for Dr. Mellor's work on the geology of the Rand and for the very weighty contribution which he has made in the paper we are discussing to the controversy about the origin of the gold in the banket.

Mr. J. S. Curtis (*Member*): As far as I can gather from Part II. of the paper under discussion, Dr. Mellor bases his repudiation of the infiltration theory upon the chutes and patches together with other features of sedimentation which are particularly observable on the Far East Rand. He takes an opposite view of chutes to what Prof. Gregory does, for the latter says in his paper "On the Origin of the Gold in Banket," III. "The objections to the infiltration theory include *(a)* The absence of ore-shoots."

Referring to Dr. Mellor's contention that these ore chutes are evidence of the detrital origin of the gold, I may say, that notwithstanding an experience extending over many years I have never encountered any so called "vorticals" in well authenticated infiltration lodes, taking this definition to mean pipes or conduits, unless this term may be applied to chutes themselves or chimneys as they are frequently called in America. Infiltration should be taken literally, meaning the process of penetrating the pores or interstices of a substance. In examining these phenomena the fact should not be overlooked that the water courses, chutes and hollows in the flat conglomerates of the Far Eastern Rand offered the same facilities for chemical precipitates as they did for detrital gold and, therefore, their presence can scarcely be used as an argument in favour of the placer theory. Investigations of gold quartz veins have not yet determined the reason for gold solutions taking any particular course, and if there has been no general rule discovered governing the distribution of the ore in quartz lodes, why should such an explanation be demanded for the ore bodies in the banket? The mineral solutions penetrated the plastic sedimentary beds in every direction, but it was in the open spaces between the pebbles that they percolated in greatest volume and had the largest range of circulation. I note the author says (p. 168-2) "one is apt to forget that the conspicuous spaces which appear between the pebbles, as represented, are really fully occupied by a matrix which may be practically as little pervious to the passage of solutions as the beds outside the limits of the conglomerate are usually supposed to be." With this I cannot agree, and I think that most lithologists will admit that this matrix is composed of somerounded grains of quartz, pyrite, crystalline quartz and a few other minerals. The rounded grains may have been deposited with the pebbles but the other minerals were deposited afterwards and in all probability by precipitation from the charged solutions. This pyrite is present in more or less quantity in all the strata of the Rand, and is everywhere more or less auriferous. How did it come there? I presume it will not be contended that it came from one source and that of the banket from another. Pyrite is not ordinarily found in the silt and sand of beach deposits nor in deltas, it is a manifestation of solfataric action. Detritalists have already called in the aid of chemistry to enable them to account for the crystalline and comminuted state of the gold by a hypothetical solution and redistribution of the placer metal, but they draw the line at that and do not admit the possibility of the original gold occurrence being due to previous chemical action of any kind. By what solvent the detrital gold was brought into solution the opponents of infiltration fail to state. The solvents of gold are not so many, among the principal ones being chlorine gas, aqua regia and the cyanide salts and nature's laboratory as far as we know does not supply these reagents. On the other hand, the sulphides of the metals including gold are freely soluble in solutions of the alkaline sulphides, and these are precisely the combinations produced by the hot springs brought into being by the molten igneous rocks. These solfataras are highly charged with hydrogen sulphide as is notice-

able in those still remaining active in other parts of the world. It is a powerful solvent of many silicates. Silicon is a strong reagent, and having been dissolved from the silicates became one of the elements of the solutions. Carbon dioxide is also a solvent, and the power of all reagents was much accentuated by the heat and pressure under which they acted. Time was also a powerful factor in mineralizing the banket as the igneous outburst did not spend itself at once but remained active for a very long period. Even were it possible, it is hardly necessary to explain the many and complicated mineralogical changes which may have taken place, but that such alterations occurred is amply proved by the metamorphism of minerals not only in the magmas of the eruptive rocks, but in the sedimentary ones as well, including the conglomerates.

Some quartz gold mines are known in which that mineral alone is the matrix, pyrite and other minerals being absent, but where the disulphide of iron is present it is almost invariably the gold carrier.

Advocates of the infiltration theory, in explaining the formation of mineral deposits are not entirely agreed as to the reactions which occurred when ore was segregated in certain portions of the earth's crust, but most of them with the exception of those who attribute these phenomena to some electrical action, ascribe them to chemical changes which were brought about when the molten magmas came into contact with surface and subterranean waters. On the Rand time and erosion have mostly obliterated surface traces of these springs, but evidence of their former existence can be observed in the ferruginous deposits to which the Dutch have given the name of "Ou' klip" (probably laterite), and their moribund manifestations, in the form of white quartz veins mentioned by Dr. Mellor, are often encountered.

A comparison of the Nome gravel deposits with the Rand formation is only possible, in my opinion, as far as sedimentation is concerned, for these placers were not accompanied by igneous rocks. It does not seem feasible to establish a parallel between loose coastal deposits and an extensive region composed of strata penetrated in so many directions by masses of eruptive rocks the effect of which has been to subject the different horizons to heat, pressure and chemical action for tens of thousands of years.

After all the main features upon which the infiltration theory rests are: The analogy with ordinary lode deposits which are almost always accompanied by some igneous rock or rocks to which the genesis of mineral bearing solutions is attributable. In the case of gold mines, especially in South Africa, this eruptive matter is frequently diorite, diabase or some allied rock. Also the occurrence of sulphides not only in the deposit itself but in greater or less quantity in the adjacent country, and finally the proofs of vigorous vein-action caused by extinct solfataras. These roughly are the usual characteristics of a lode undoubtedly due to infiltration. All these conditions are present on the Witwatersrand.

There are several statements in Dr. Mellor's most exhaustive paper which I should like to comment on but I fear that I have already taken up more time than I am entitled to.

The meeting then terminated.

Obituary.

Lieut. W. J. N. Dunnachie.

The death is reported of Lieut. William J. N. Dunnachie, an Associate Member of the Society. Mr. Dunnachie, who joined the Society in March, 1907, when a sampler at the Consolidated Langlaagte Mine, was afterwards associated with the Geldenhuis Deep (West Section) and the New Rietfontein Estate and G.M. Co., Ltd., in other capacities underground. Several years ago he returned to England, settling at Airdrie, where he became the manager of the Gartness Colliery. Soon after the war started he received a commission in the Royal Engineers and, after training, went to France in charge of a section of the Tunnelling Corps. Mr. Dunnachie was home on short leave in April, returning to duty on the 11th, and being killed in action on the 15th of that month.

Mr. Dunnachie who was only 34 years of age, married a few years ago, and leaves a wife and two children, the youngest being only a few months old. The sincere sympathy of the Society is extended to Lieut. Dunnachie's wife and family.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RESEARCH MEN FROM THE MANUFACTURER'S VIEW POINT. (Abstract of a paper read to the New York section of Soc. Chem. Ind.).—"Research, invention, discovery, are words which possess a magic charm to those interested in progress, growth and efficiency. . . . 'Research pays' is a phrase which is coming to have a status comparable to the expression 'It pays to advertise.' Everywhere we see manufacturers who are recognizing the value of research. Especially in these strenuous war times, under the motherhood of necessity, research work is receiving a great impetus. . . .

It has been found by experience that not all advertising pays, and it is quite as true that research, as it is frequently conducted, is not always profitable. Many manufacturers who are indulging in it are not doing it properly, and many men who are hitting themselves as research workers are destined to be unsuccessful because they lack certain essential qualities.

Having installed a research laboratory, the mistake is commonly made of burdening it with work of a strictly routine nature, such as control analyses, testing of materials in daily use, coal and fuel analyses, and the like. This kind of work has such a retarding influence on research that some manufacturers attempt a complete separation of the two kinds of laboratories. This has not become a common practice, however, and control and research work are commonly found in a state of confusion.

An instance is recalled where after operation for some time it was found that no records of any value had been kept, even though the work and results were of an important nature. Although the importance of an adequate system of notes and reports is obvious, it is a feature often neglected. There is no doubt that at the present time there are too few men being trained in the right way to supply the present and the coming demands for research workers.

As has been pointed out by Dr Whitaker, the fact that mechanical engineers have played the important part they have in the development of chemical industries in this country, in spite of their limited training in chemistry, has been due largely to their engineering training, and there is no doubt that in the establishment of chemical engineering courses a marked step is being made in improving the methods for training men to assist in chemical research.

By a reasonable administration and accounting it can be demonstrated that research pays, and coupled with the campaign to create interest in and support for research work those should be a greater effort toward meeting it directly. Systematic co-operation between the universities and industries is needed in the training, which is equally important with the question of research administration of research work in the . . . ."

C. F. BURGESS, *Journal of Society of Chemical Industry*, Dec. 15, 1915, pp. 1145–1147.

METALLURGY.

THE FUTURE OF ELECTRIC STEEL.—A good many authorities on steel-making assert as the very appearance of confidence that the making of rapid and tremendous progress of the electric melting of steel has begun, and will constitute a distinguishing industrial feature of the present decade. This view is strongly supported by the particulars of the activities of the past year in relation to this matter which are given below. The number of electric furnaces in the United States has increased during the past twelve months by 32 to a total of 73, and the American total now exceeds by 20 that of Germany, which had previously occupied the first place in this branch of industry. Moreover, of the German furnaces, 16 are melting ferro-manganese, and consequently are not a factor in the steel trade. The American development of electric melting has been mainly in the production of steel castings of ordinary analysis, but more solid and free from blowholes than can be obtained by the baby Bessemer or the open hearth process, and of alloy steels ranging from the high-priced 'quality' steels used for aeroplane and automobile construction up to tool steel itself. The latter kind of steel is being turned out in increasing volume by the electric furnace. An expert who, in the columns of the *Iron Age*, has been reviewing the year's progress, goes so far as to assert that

> The control under this process is so complete and oxidising gases can be so absolutely excluded from the bath, that material even better than crucible steel has been produced from the electric furnace. It would seem that, as far as the crucible process is concerned, the hand writing on the wall is plainly visible.

Such a prediction will not be completely endorsed in Sheffield, although even in that fastness of the crucible process electric melting is making rapid headway. In Sheffield 15 furnaces were installed last year, of which eight are employed in melting high-speed steel, whilst others are in course of construction and under consideration. It is quite natural that Sheffield should be reluctant to abandon the crucible. Its tool steel enjoys a unique reputation for superiority in all parts of the world, and the discontinuance of a process which has given brilliant results, both in quality and profits, is necessarily a matter that calls for the most careful consideration. Another factor is the large amount of capital which is invested in crucible plant in Sheffield. It has been argued in the past that in the proximity of the coal fields coke melting is more economical than the use of the electric current, but experience may establish that the drawback of cost is outweighed by the gain in uniformity, higher refining, and other advantages. At the moment crucible melting is extremely costly, owing to the price of coke having risen to nearly double the pre war figures and to the dearness of the crucibles. Whatever the truth may be as to the respective merits of the two processes in a centre which has coal at its back door, and where the cost of electricity is comparatively high, the attitude of Sheffield towards the question appears to be veering round to the newer practice. There is great scope for the electric furnace outside the manufacture of tool steel. New kinds of steel are in demand which can be successfully made only in that furnace, and the call of the aeroplane, automobile, and other progressive industries for steels of higher quality is compelling recourse to the same system.

The statistics indicate that progress is chiefly with the arc type of furnace, which is making rapid strides everywhere. In the United States

there are 68 against two of the induction type, and in other countries the number is 179 against only 35 induction furnaces. No doubt the simplicity of the arc type and its similarity of construction and operation to the small tipping open-hearth furnace have helped to influence makers in their choice. Incidentally, it may be remarked that the merits of the small tipping open-hearth furnace are not yet fully appreciated. There is a very promising future for this type not only as a "feeder" or preliminary refiner (a sort of baby metal-mixer) for the electric furnace, but also in small steel foundries for ordinary grade of castings, in malleable iron foundries, and similar works, apart from applications in the non-ferrous metal industry.

According to the *Iron Age*, the total number of furnaces in operation for the electric melting of steel has increased from 213 to 303. Germany now takes second place with 53, an increase of only six. Progress was greater in England than in any other European country, our number of furnaces having gone up from 16 to 46, and there is every probability of the same rate of increase being maintained in the near future. A considerable number of contracts for new installations have been placed or are under consideration, and many more will be needed if steel-makers in the country lay themselves out (as is highly probable) to capture a share of the huge trade in small steel castings which in the past has been left largely in the hands of Continental firms. There is a decided opening for a small electric furnace of moderate first cost and simple design, both for the production of small castings and special steels for various purposes.

During the year Sheffield has installed or contracted for electric furnaces as follows: Hadfields (Lim.) 5, Vickers (Lim.), Thos. Firth & Sons (Lim.), John Brown & Co. (Lim.), and Brown, Bayley's Steelworks (Lim.), two each, Arthur Balfour and Co. (Lim.), S. Osborn and Co. (Lim.), and the University of Sheffield, one each. The whole of these furnaces are of the Heroult type, and range in capacity from 3 tons to 10 tons. Eight of them are producing tool steel and the remainder castings and war material. The following British firms have also installed or are installing electric furnaces: London Foundry Co. (Lim.), Brimsdown; Daimler Motor Co. (Lim.), Coventry; Thwaites Bros. (Lim.), Bradford; and T. Summerson & Sons (Lim.), Darlington.

As regards types the Heroult furnace stands first with a world total of 115 out of 303, induction furnaces ranking second with 48, the Rennerfelt third with 35, and the Gronwall next with 15. The number of Heroult furnaces in the United States has increased from 19 to 43. Other figures for the same country are Snyder 11, Wile 7, and Rennerfelt 2. There are also 4 Rennerfelt installations in England at the following works: Edgar Allen & Co. (Lim.), Sheffield; A. Kounick & Sons (Lim.), Birmingham; Steel Castings (Lim.), Birmingham; and Electro Flex Steel Co. (Lim.), Dunston. There are 11 electric furnaces in Russia, 23 in Sweden, and 6 in Norway. In Canada the manufacture of steel by electric melting is carried on by Electric Steel and Metals (Lim.), Ontario; Armstrong, Whitworth, of Canada, Quebec; Thomas Davidson Manufacturing Co., Montreal; and the Canadian Brake & Shoe Co., Quebec, these firms having eight furnaces among them.

In view of the above facts it would appear that, although we have been perhaps a little tardy in recognising the full merits of the electric furnace, yet, as in the case of aircraft, submarines, and many other technical developments, recognition is being followed by industrial application which will place British engineering science in the forefront of progress." *The Ironmonger*, Feb. 12, 1916 (G. H. S.)

Electrolytic Zinc.—"The 'French' process consists in the use of a solution of bisulphate of soda, which is a cheap waste product, and a small quantity of manganese, an ingredient which occurs with most zinc ores. Bisulphate of soda rapidly dissolves the zinc from the roasted ore. Along with the manganese in solution, it completely prevents the anodes from being affected. This solution has little resistance to the passage of the electric current in the electrolytic vats, and the consumption of current is thus lowered. It has also the peculiarity of throwing out of solution practically all the impurities which usually contaminate the zinc. Early in 1912 the Consolidated Mining & Smelting Company of Canada acquired the sole rights for the use of this process in Canada, but after they had produced a few tons of zinc, the contract between French's company and the Consolidated was cancelled on account of disagreement and alleged violation of contract. A demonstrating plant was then erected at Silverton, on Slocan Lake, by the Standard Silver-Lead Mining Co. to prove that the process would work on a large scale. It comprised two completely equipped dissolving vats, each capable of dealing with 2,500 lb. of liquor at a time and about 1,000 lb. of ore. The zinc-depleted liquor containing bisulphate of soda from the electrolytic vat is pumped into the upper of these two vats. There is then added 700 lb. to 1,000 lb. of roasted ore from which the zinc had been partly extracted in a previous operation. In about an hour, solution of the zinc is finished, and when the liquor has settled for a short time, it is allowed to run down into the second dissolver, placed at a lower level. The sludge remaining in the bottom of the dissolver is then pumped through a filter-press to remove the water and the residue contains all the silver and lead. As the sulphur to the extent of more than 20% has been driven off in the roasting operation and zinc to the amount of about 40% has been extracted, the residue of the ore only weighed about 45% of the original ore used. As it contained all the silver and lead originally in the roasted ore, the percentage of these metals was more than doubled. For instance, the ore treated assayed 31% zinc, 3.5% lead, and 32 oz. silver per ton. The zinc-depleted residue contained 76 oz. silver per ton, and 87% lead, and 92% of the zinc had been taken out, that is, 2.7% of the zinc counted on the original ore used was left in the residue. This residue is now ready for smelting or other treatment for the recovery of its lead and silver in the usual way. To the liquor which has been run down into the second dissolver, a quantity of roasted ore is again added to neutralize the bisulphate of soda. As soon as this has taken place, the liquid settles rapidly to a clear solution, which is pumped through a clarifying filter-press, and is then ready to have its zinc deposited in the electrolytic vats. At the Standard Company's plant only one electrolytic vat was used, as the

dynamo power at disposal was limited. Nine anodes were used and eight cathodes of the largest size, each with an area of 8 sq. ft. Although 4,000 kilowatt hours were allowed in costing for the deposition of one ton of zinc, only 3,000 kilowatt-hours were actually required, the lowest reading over a period of 18 hours having been 2,680 kilowatt hours. As the zinc is deposited from solution in the electrolytic vat, the bisulphate of soda with which it was combined is regenerated and is used again in the next dissolving. The working costs vary with the ore, but they are now well defined. The plant is expensive, but much less than that required for a zinc smelter of similar capacity. Ores of almost any grade above 10% zinc can be treated equally well, but naturally the cost of treatment for an ore with a larger zinc content is less." A. G. FRENCH, Bulletin, Canadian Mining Institute, Aug., 1915. *Mining Magazine*, Sept., 1915, Vol. XII., No. 3, p. 167. (J. E. T.)

METALLURGY AT SONS OF GWALIA. "The *Monthly Journal* of the Chamber of Mines of Western Australia for June contains an article written by A. Wanchope, the general manager, describing the present system of metallurgical treatment at the Sons of Gwalia gold mine, at Mount Leonora, West Australia. This mine has been in operation successfully since 1898. Four years ago the results of development on the main lode became disappointing, but by a campaign of lateral exploration suggested by Malcolm Maclaren a parallel lode was discovered. The latest report recorded that the ore reserve was sufficient for four years and that developments had continued to be satisfactory. Until two years ago the metallurgical treatment consisted of reduction in stamp mills, amalgamation in pans, removal of sulphides by Willfleys, roasting concentrate and subsequent amalgamation and cyaniding, leaching sandy tailing, and agitating slime tailing. Experiments, however, showed that the process could be greatly simplified by omitting the concentrates and roasting furnaces, and reducing the tailing after amalgamation to slime. Another alteration made was the substitution of gas engines for steam plant, and the manufacture of gas from wood. According to the last yearly report, 160,963 tons was treated, for a yield of gold worth £250,501, or 31s 1d per ton. The ore first goes to 50 stamps, weight 1,150 lb., with screens having 6 to 8 holes per linear inch. The discharge averages 40% of −60 and 50% of −150. Throughout the plant, crushing is done in weak cyanide solution. No amalgamation is done in the stamps or on plates, but the pulp is delivered to ten 5 ft. Fairwood Down grinding and amalgamating pans. These pans are fitted with the Freeman selective pipe discharge described in our issue of December, 1910. Mercury to the extent of 0·23 oz. per ton of ore is consumed in these pans, and about 20% of the total gold recovered at the mine is obtained in the amalgam. The tailing from the pans is elevated to classifiers, going first to two primary diaphragm cones 11 ft. high and 8 ft. diameter at the top. The fine overflow passes through a second sand trap to thickeners, and the sandy underflow passes to three secondary diaphragm cones 8 ft. × 6 ft. The underflow of the secondary cones goes to three tube mills placed in circuit. The tube mills measure 5 ft. × 22 ft., revolve at 27 r.p.m., have a capacity of 40 tons per day each, and produce a pulp containing 10% of −150 material. The linings are Ward corrugated iron, flint pebbles are used, the total load of pebbles for each mill is 7 tons, and the consumption of pebbles 1½ lb. per ton of ore. The overflow from the primary cones passes into three involute-arm thickening vats, each 10½ ft. deep and having a settling area of 527 sq. ft. To prevent undue disturbance in the vat the product is delivered at a point about 12 in. below the surface of the pulp in the centre of the vat. The involute collecting arms travel at the rate of one revolution in three minutes, and continuously sweep the settled slime on the bottom of the vat to a centrally situated discharge. The settled product contains 45% water. The underflow from the thickeners is elevated by an air-lift to a self-rotating distributor, by which it is delivered to the agitators. There are four agitating vats connected in series, through which the product is continuously passing. The contents pass through an inclined 6 in. pipe from the bottom of the first vat to the top of the second vat and so on. The vats are 28 ft. in diameter, and have a total capacity of 15,890 cub. ft. A small continuous stream of dissolved cyanide drips into the first of the series of agitators. Agitation is effected by mechanically-driven arms, operating at 6 r.p.m., supplemented by low-pressure compressed air introduced through a small pipe down the centre of the hollow agitator spindle to the bottom of the vat. The product from the last of the agitators passes by gravity to a vacuum filter, Cassell type. The filter vat is a rectangular chamber 34 ft. long by 10 ft. wide, and is provided at the bottom with three hopper discharges fitted with Shaw-type gates, delivering into a sunken concrete agitator provided with two sets of agitating gear. The vacuum tank accommodates 11 vertical filter leaves, each 9 ft. × 5½ ft. The filtering medium is of cotton duck stretched over frames made of 1 in. pipe attached to solid headboards. Cocoanut matting, reinforced on each leaf with about 34 vertical wood slats each 5 ft. × ¾ in. × ½ in., is rigidly fixed between the two layers of duck by numerous rows of vertical stitching. The total clear filtering area is 4,461 sq. ft. and the total load of slime (dry weight) is from 40 tons to 45 tons. The cloths have a life of about six months, a long life solely due to the partial softening of all water taken into the plant, the object of which is to prevent any precipitation of magnesia upon the mixing of water with plant solutions. Were this precaution not taken, the cloths would quickly become petrified with magnesia, with a consequent clogging of the pores. The cloths are not treated with acid or otherwise, but are simply discarded when inefficient. A complete cycle of filtering operations occupies about two hours. The maximum vacuum during filtering is about 25 in., and the minimum during transfer about 7 in. barometrical pressure at an elevation of 1,400 ft. above sea level. The cakes are blown off the filter-leaves with compressed air, and fall through the hopper bottom into the sunken agitator, whence the product, diluted with mine salt water, is pumped to the tailing area. The excess pulp and washes are stored in their respective agitating vats, whence they are re-elevated to the filter vat by a 10 in. centrifugal pump. Vacuum is maintained by wet vacuum pumps, which also elevate the filtered liquors to the clarifying vats. A device for preventing the cakes from cracking and falling off the filter leaves between the changes of wash consists of a number of sprays which are in operation during the changing periods when

they drench the cakes with a cloud of fine spray. At the same time this spray of solution also washes down the sides of the vat during the withdrawing of the pulp or washes, and so prevents salting of the following wash. This device has assisted in reducing the soluble gold losses in the discharge of pulp to a minimum. The vacuum pumps deliver the solutions from the vacuum-filter to three vats, each 29 ft. in diameter by 7 ft. deep, and fitted with a nest of filter-leaves, similar to, but smaller than those employed in the vacuum-filter. The solution gravitates through these leaves to the tonnage recorder in a perfectly clear state. Approximately 750 tons of solution is clarified daily in these clarifiers. The remainder of the installation, comprising the precipitation and smelting plant, do not call for special notice."—A. Warchope, *Monthly Journal*, Chamber of Mines, W.A., June, 1915.—*Mining Magazine*, Sept., 1915, Vol. XIII., No. 3, p. 168. (J. E. T.)

Chromium and Tungsten in High-Speed Steel.—"It will be difficult to over-rate the importance and value of the paper contributed to the recent meeting of the Iron and Steel Institute by Prof. C. A. Edwards and Mr. H. Kikkawa. The primary object of the research was to determine the effect of chromium and tungsten upon the hardening and tempering of high-speed tool steel. Prof. Carpenter had previously reported upon the tempering of a series of high-speed steels by heating hardened specimens at the desired temperature and cooling in the furnace. The results obtained appeared to show a gradual tempering as the re-heating temperature was increased, and the results, as pointed out by Mr. Taylor, did not bear out practical experience. Mr. Taylor's results showed that to obtain the maximum efficiency it was necessary to reheat the hardened tools to a temperature somewhere between 370° C. and 671° C. The best temperature is suggested as 621° C. and the period of heating about five minutes, after which the tool may be cooled either rapidly or slowly.

In a previous report upon this subject, Dr. Edwards suggested that the failure of high-speed cutting tools was due to the formation of a brittle constituent which destroyed their cutting edge. It is found that this is not the case and that the brittle constituent is only formed when the steel has been overheated during the high heat treatment.

The present research consists of an examination of eleven samples of steel subdivided into two series. The first contains a uniform content of about 6% of chromium with tungsten increasing from nil up to 12·5. The second series contains a uniform content of tungsten of about 19 with chromium increasing from nil up 6·24. The carbon is practically constant in all the steels at about 0·65, whilst the silicon ranges from 0·04 to 0·10. The investigation consisted of taking specimens 1 in. square by $\frac{3}{4}$ in. thick. These were hardened by heating in a crucible contained in an injector furnace to about 1,350° C. for ten minutes, followed by quenching in an air blast. The hardened specimens were then carefully ground on a whet-stone and examined microscopically. In the tempering experiments specimens were heated for one hour after reaching the desired temperature, and then followed to cool on an asbestos pad in the ordinary atmosphere. The hardness determination and microscopic examinations were made after each treatment, and then the same specimens were heated to the next higher temperature.

The results are extremely interesting and are set out in a very lucid manner in the form of curves, whilst the micro structures are illustrated by a few carefully selected micrographs.

A careful reading of the paper will fully repay anyone interested in the constitution and treatment of high-speed steel, and the following conclusions may be stated as showing the ground covered in the research :—

(1) The first effect of tempering hardened high-speed steels is to make them softer, but when they are tempered at higher temperatures they again become harder, and after heating at or about 614° C. they are much harder than in the initial air-quenched state. There can be no doubt that this secondary hardening is the cause of the improved cutting powers of a tool, which Mr. Taylor found was brought about by a secondary low heating to about 620° C.

(2) Chromium in conjunction with carbon is the cause of the great hardness of hardened high-speed steels, and further, it materially lowers the temperature at which these steels can be air-hardened.

(3) In the absence of chromium, tungsten raises the temperature at which tempering or annealing begins, and in the presence of chromium it increases the intensity of the secondary hardening and raises the tempering temperature.

(4) Tungsten steel containing 18·0% of tungsten and 0·63% of carbon can be air-hardened only by rapid air-quenching from temperatures above 1,050° C.

(5) When high-speed steels are hardened at low temperatures, say, 1,050° C., the tempering properties approximate to those of a pure chromium steel, by softening at a low temperature and developing little or no secondary hardening. This is due to the tungsten being undissolved and remaining inactive.

(6) The maximum resistance to tempering and the greatest degree of secondary hardening can only be obtained by getting the tungsten into solution, and with modern high-speed steels this is not complete until a temperature of about 1,350° C. is reached.

(7) Specific gravity determinations seem to indicate that there is a direct connection between the hardness and volume of these steels. On tempering, an increase of hardness is accompanied by an increase in volume." Thomas Swinden, Transactions Iron and Steel Institute.—*The Engineering Review*, Oct. 15, 1915, Vol. 29, No. 4, p. 115. (J. E. T.)

Admiralty Gun Metal.—"Dealing with the raw material, the writer emphasizes the fact that the use of reliable metals is most essential and that considerable trouble is met with by using cheap material. In regard to copper the reader is cautioned in using mysterious ingots of unknown origin, and even the initials 'BS,' which at one time denoted a high grade material containing no more than 0·05% arsenic, and only a trace of antimony, are now without meaning. A series of analyses of these samples of commercial copper is given, the one recommended for alloy making is a good type of English 'BS' copper containing 99·75% copper, 0·025% arsenic, 0·006% iron, 0·021% lead, 0·041% nickel, 0·011% bismuth, and 0·143%

oxide. There was a trace of arsenic and no tin. Some interesting notes are added in regard to the tests of the metal.

The next most important article in the manufacture of gun metal is tin. Reliable samples contain over 99% tin with small quantities of antimony and lead, and very small amounts of iron, bismuth, arsenic, and sulphur.

Zinc also plays an important part as a constituent of gun metal. Good grades of commercial spelter contain about 99·8% of zinc, with small contents of lead, iron, cadmium, tin and antimony.

Lead is not, as a rule, an intentional constituent, but is usually present, and is now tolerated to the extent of 0·5%. It is particularly useful in assisting machining.

In the preparation of gun metal it is always necessary to melt the copper first, protecting from the furnace gas as possible by means of a suitable cover of powdered charcoal, glass, or salt. The tin is added to the crucible containing the copper at a temperature of about 100° above its melting point. The tin must be well stirred in, and added in quantities small enough to preserve the fluidity of the copper.

The tin is readily oxidised, and even when the alloy is raised to full red heat again the alloy contents have mixed through it some particles of tin oxide. The small quantity of zinc is added to assist the removal of the latter by reducing the tin oxide to metal. The zinc suffers loss both by volatilisation and oxidation to the extent of about 0·05%." Thomas Swinden, *Metal Industry*, Sept., 1915. *The Engineering Review*, Oct. 15, 1915, Vol. 29, No. 4, p. 116. (J. E. T.)

MINING

Mineral Production of South Africa.—"The total value of the mineral output of the Union of South Africa for 1914 was £15,366,224, as compared with £52,924,990 in 1913.

The following table shows the production of the chief minerals:

	1913	1914
	£	£
Gold	37,371,553	35,664,230
Silver	115,822	102,471
Diamonds	11,389,807	5,487,194
Coal	2,210,458	2,258,896
Coke	15,862	12,377
Copper	507,856	692,355
Tin	136,550	311,391
Asbestos	16,028	20,087
Graphite	1,257	970
Magnesite	1,194	1,151
Lead	1,199	1,396
Salt	77,142	67,648
Lime	118,984	109,041
Flint	3,782	2,508

[illegible]

All diamond mines were closed down at the outbreak of the European war. The output of gold in 1914 represented 48% of the world's production. The total production of gold in the Union of South Africa since 1886, when gold was first discovered, amounts to approximately £145,840,000 up to the end of 1914, and was practically all won in the Transvaal." *Board of Trade Journal*, Sept. 9, 1915.—*Journal of the Society of Chemical Industry*, Sept. 30, 1915, p. 966. (J. W.)

Air and Temperature in Mines.—"The deepest mine workings ever attempted in any part of the world are situated in the hot regions of Brazil. One of the mines of the St. John del Rey Mining Company, Limited, has reached the vertical depth of 5,826 ft., and since the vein shows no sign of losing its size or value, the company is considering means of continuing to a vertical depth of 7,626 ft. The question seems to be one of temperature and air supply. In the following abstract of a report which appeared in the *Canadian Mining Journal*, G. Chalmers, superintendent of the Morro Velho mine, discusses the problem of ventilation of the workings at these unprecedented depths.

Investigations in connection with ventilation at the Morro Velho mine of the St. John del Rey Mining Company, Limited, substantiate previous reports that the system of distribution of the air, although meeting the case satisfactorily and in a most economical manner in the past, is becoming as greater depth is attained, defective, and requires modification. Even when on rare occasions the downcast air at the bottom of the mine is comparatively low in moisture, by the time it has passed over two horizons it has become saturated, partly from the watering of the ways to lay dust, the sweat from the bodies of workmen, and from the breath of the animals and men. Consequently the efficiency of the men on the stopes above must be considerably impaired; in fact, according to authorities on the subject, 86° wet bulb (which represents the conditions of, say, horizon 16, at any rate during the rainy season) is supposed to be the limit at which men can work with full efficiency, and as the mine deepens this will, of course, be more seriously felt. But by largely increasing the volume of air passing through the deepest explorations, and by leading the allotted quantity (after it has passed over the stopes of one horizon, or at the outside two) directly to the upcast and away to the surface, instead of passing it over the stopes above as in the old and existing system, the best possible conditions will be obtained.

Due to the fact of our now being in a much better position as regards the mainway into depth, namely, of its being in advance of the explorations, Captain Watts has proved that it is possible to work each block by one vertical winze only, and our calculations show that this one winze and a steel pipe of 3 ft. to 4 ft. in diameter will, with the eastern paved airway, give ample ventilation to the stopes during the working out of a block of mineral. Consequently, one winze for mineral and filling to act as a temporary upcast, which during the working out of each block disappears as the mineral is removed, is sufficient for the mining operations, and by sinking another vertical winze from one horizon to the other, never less than 50 ft. from the average line of the north wall of the lode, and some connecting tunnelling, a permanent return airway will be formed at a cost no greater than that of the old system of working out each block with two vertical winzes and a western wood block airway and the eastern paved airway, as the large saving in the temporary western airway in the form of a 3 ft. to 4 ft. steel pipe over the wood block airway formerly used, is more than sufficient to pay for the tunnelling between the top of one permanent upcast winze and the bottom

of the next, and that connecting the temporary winze with the permanent. There is another advantage in this system—namely, that when it is completely established we have a second permanent way into depth from horizon 17 down.

Independent Air Ways.—It was difficult to determine the point at which the improved system should commence, but for various reasons No. 23 winze offers the best opportunity for an independent upcast air-way, and consequently it has been decided that the new system should commence from this point.

The benefit of the proposed alteration in the ventilating system will not be felt on the blocks of mineral to be worked out from horizon 17 up, but from 17 down. Those of us who have been responsible for the reopening of this mine from surface to its present depth of 5,826 ft. have had the opportunity of realising the steady rise in temperature as the depth has increased and have been frequently reminded of the difficulties in store for the company, especially at such times when an inadequate volume of ventilating air had been passing over the stopes or through explorations, caused by the partial closing of air-ways, the failure of fans, or by the explorations progressing more rapidly into depth than the main ventilating system. However, for many years it was possible to keep down to a reasonable figure the temperature to which the miners were subjected, and it appeared that by increasing the volume this condition could be maintained.

In the first place, natural ventilation was sufficient, later, furnaces had to be adopted during the hot season to assist the natural ventilation. As the mine became still deeper a Capel fan was installed at the top of the upcast "C" shaft to ensure a more constant volume of air. Finally, a Sirocco fan of large capacity was installed in place of the former, the output of which could be increased by additional power when required. Besides this, auxiliary fans are used on all explorations beyond the main system.

The existing ventilating system, which is referred to above as not satisfactory at the present depth, was forced upon us at a time when a more elaborate one of an entire independent air-way from surface down, which naturally recommended itself, would have been too expensive. The former represented the cheapest means of ventilation, and until quite recently met the case satisfactorily, as long as the system and supply of air was not interfered with by the above-mentioned causes. However, as depth has been attained the necessity for modification in the distribution of the air becomes imperative if the mine is to be worked to a much greater depth, of which there is every probability, seeing that the lode has so far shown no signs of failure in size or value in the lowest explorations made upon it, and as also there is apparently no geological reason against its continuance in depth; and the prospects of this company being able to work it profitably to a great depth seem only to be limited by the measurable additions in cost of winding, hauling, handling, mineral, and other engineering matters. All these increase a little more or less proportionately to the depth, with one exception, viz.: that of increasing heat in depth, which has presented itself to our minds as by far the most important obstacle—that is to say, if it cannot be overcome by some satisfactory artificial means of cooling.

It was in consequence decided some years back to institute investigations with the object of obtaining information as to the increasing air and rock temperatures and other factors that go to make against the miners' efficiency, and add to cost the deeper the mine is sunk. As regards the rock temperature, bore holes were made at the different horizons or vertical depths, and the temperatures taken. For a considerable time, due to the holes being too shallow, or badly situated, and to our want of experience in the matter, and somewhat unreliable instruments, our readings were only very approximate. Gradually, however, the conditions for performing this work were improved and our preliminary investigations from year to year went to show that in spite of the increasing heat found at the moment of opening the rock the deeper we went, it gradually became cooler after it had been in contact with a current of ventilating air.

From the temperature readings we obtained of the rock at the moment of opening different horizons we were able to construct a rock temperature grade as a guide to what we might expect in depth. Since then, other horizons in depth have been opened, and our grade agrees closely with the readings taken, with the exception of that at horizon 20, which is lower than it should be owing to the exceptional volume of ventilating air forced to that point by the auxiliary fans cooling down the rock before its real temperature could be taken.

Probable Temperature at 7,000 ft.—As before stated, without this rise in temperature from the rock and other causes, the probable running engineering expenses in working the mine down would not mean a very serious increase, and this has proved itself in practice as far as we have gone. Supposing the lode continues the same in size and value to horizon 20 (a vertical depth of 7,626 ft.) it could be worked profitably to that point and even to a much greater depth, but from the grade on our temperature chart, the rock at horizon 26 would be no less than 126·5° Fahr. at the moment of opening.

We have, of late, realised from practice that the best we can now expect in the absence of artificial cooling would be a point where the efficiency of the men would be affected.

The temperature of the rock that had been laid open and exposed to the ventilation for some time year by year fell to some extent, but the deeper we went the cooling effect of the ventilating air would become less efficient, as we were year by year largely increasing the rock area; but beyond certain reasonable limits we could not increase the volume of the air, and consequently, unless the air could be artificially cooled on surface, advantage being taken of the operation to deprive it of some of its moisture, the possibility of working the mine to horizon 26 would be small, unless the working hours were reduced by the introduction of four shifts in the place of three, which would not only mean a large direct increase in labour cost, but necessitate addition in house accommodation and other matter representing a serious item in an isolated place of this kind where all work is costly. It was accordingly decided to make further efforts in obtaining more reliable information as regards this important question so as to arrive at the most economical and efficient means of reducing both the temperature and moisture of the atmosphere within the mine.

The increase of air temperature due to the air becoming more dense as it passes down the mine,

is explained by authorities on the subject, and checked by us, showed under normal conditions an increase in rock temperature of approximately 1° F. for every 180 ft., whilst the increase in rock temperature is approximately 1° F. for every 125 ft.

Heat from Increased Density of Air with Depth

There are many other causes for increase in temperature in the mine, viz., those due to the friction of the air, heat from the bodies of men and animals, etc., but the heating effect from the rock, and that due to the compressing of the atmosphere as it descends into depth, are, in the case of this mine, the most serious, and it is evident that unless the air as it enters the mine is made considerably cooler than the average surface temperature, it could in descending into depth arrive at a point below the present bottom where its temperature could not allow men to work. But if the air entering the shafts can be cooled to something very considerably below that of the normal atmosphere, and the supply is abundant and constant, the rock will be progressively cooled from surface down, and, as time goes on, the cooling by air will be more effective in the bottom of the mine than at the time of starting the plant.

During the cold season an experiment was tried by night, when the temperature was considerably below the normal, in which a known volume of air was passed through a measured distance of the adit tunnel into the mine, the rock temperatures being taken throughout the distance of the tunnel and period of trial, and at the end of five hours it was interesting to find that a considerable fall in the rock temperature had taken place, and it would no doubt have continued to do so to some extent had it been possible to maintain a supply of cool air, but as the sun rose, the air became warmer and the rock rose again to its original temperature.

As before stated, the drying of the air has always been considered necessary in conjunction with the cooling operation, but its importance was not so fully understood until information was gained from a valuable paper by Prof. John Cadman, and the opinions of other authorities on the subject had come to our notice. Unfortunately, the drying of the atmosphere is somewhat antagonistic to a satisfactory solution of the dust question, which necessitates watering of the roads and stopes to keep down the dust. However, the proposed distribution of the air, previously referred to, will apparently do far more to overcome the dust trouble than the drying of the air will make against our efforts in this respect, as the allotted quantity of air, after passing one stope, or at the outside two, will go directly with what dust it has taken up, to the upcast and away to surface, instead of the present system, in which the same air passes the bottom stopes, from thence over the remaining stopes, with some slight addition of fresh air at each horizon, but accumulating dust all the way, until it finally reaches the highest stope, and from thence goes to surface.

The actual necessities as regards cooling and drying to ensure that as greater depth was attained the miners would work in a perfectly healthy atmosphere (even more so than in the upper sections, having been appreciated), the next point to determine was where the operation of cooling and drying should take place. Some advice had been offered to the effect that it would be most advantageous underground, and it is true that some benefit would be derived from this; but the increase in mechanical appliances underground represents a serious obstacle to our particular form of mining operations, and, further, on account of our principal cooling agent (water) being found on surface in abundance, whilst underground there is very little, and that at a high temperature, it has been decided that the cooling plant should undoubtedly be installed on surface—namely, at the top of the "D" downcast shaft.

Mr. Eric Davies, whose duty it has been to carry out the investigations, has also been entrusted with the designing of a plant which we consider most suitable for the purpose, and careful study has led to the proposing of this plant, not for extreme cooling or drying of the air, but one in which each operation will be performed to the most economical point, at the same time making use of ordinary cooling appliances which in practice have long since proved their reliability in cold storage, dry blast for furnace and other installations, and in consequence it represents a thoroughly practical scheme, which will effectually remove the one serious obstacle against the company being able to work the mine to a great depth. It will, however, be placed before specialists on cooling installations before it be finally determined upon. As regards the necessary power for driving the plant, this is not excessive, amounting to approximately 400 h.p., and as to the provision of this power, with that of 200 h.p. for ventilation, and other additions necessary for the working into depth, it has been satisfactorily arranged for.

The mine captain considers the efficiency of the miners is very little affected so far, and the writer is inclined to think that men who have been used to the damp, hot climate of Brazil for some years are probably more capable of working efficiently in 80° wet bulb temperature than men in England accustomed to a damp, cold climate on surface. Often enough in the hot season here the change from surface to the bottom of the mine is only slight, but the change in England would, of course, be a far more severe trial. Even in the hottest places in the mine the contractors are eager to work twelve hours, and had it not been for our complying with this to some extent, we should have lost our men. It is, however, evident that in spite of the apparent possibility that this climate is favourable to the efficiency of men working in hot mines, eight hours is quite sufficient for steady workers at the present depth."—G. CHALMERS, *Mexican Mining Journal, Queensland Government Mining Journal*, Feb., 1916, p. 74. (H. A. W.)

Abstracts of Patent Applications.

(C.) 13.10. David Hibbert Thomas (1), Arthur Douglas Beck (2), Frederick John Morgan (3). Improvements in and relating to spray producers or atomisers for liquids. 20.1.16.

This application relates to a spraying device consisting of a method of atomising and spraying liquids consisting in conveying or causing to be conveyed by gravity or otherwise, the liquid to be atomised separately to the point of convergence of two or more streams of a gas under pressure, such as steam or compressed air.

(C.) 1bl. 15. C. H. Perrins. A new and improved method and means for the disinfection of mines and underground workings.

This application refers to a means for the generation of chlorine gas evolved by heating a mixture either through the usual process of blasting or other means of ignition.

APR 1 1920
UNIVERSITY OF TORONTO

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Vol. XVI. JUNE 1916. Last part of Volume XVI. No. 11

Proceedings
AT
Annual General Meeting, June 24, 1916.

The Twentieth Annual General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, 24th June, 1916, Mr. Jas. E. Thomas (President) in the chair. There were also present:—

24 Members: Messrs. J. Gray, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, W. W. Lawrie, H. Meyer, C. Toombs, F. W. Watson, J. Watson, Prof. G. H. Stanley, J. R. Williams (Members of Council), H. R. Adam, J. Chilton, H. W. Gill, C. E. Hutton, H. H. Morrell, P. T. Morrisby, S. Nettleton, S. Newton, E. A. Osterloh, J. J. R. Smythe, J. A. Taylor, A. Thomas and T. P. Waites.

4 Associates and Students: Messrs. D. W. Pugh, H. Ward, L. A. Womble and J. A. Woodburn.

10 Visitors and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

OBITUARY.

MR. F. J. POOLER.

The President: It is with great regret that I have to announce the death of one of our associates, Mr. F. J. Pooler. He was present at the last meeting, and contributed some remarks to discussion. I, in common with some other members of the Society, attended his funeral just a fortnight ago to-day. I would ask all present to rise as a token of sympathy with Mr. Pooler's family.

All present rose to their feet.

The President: I will now ask your Secretary to read the Annual Report, and the Statement of Accounts.

ANNUAL REPORT OF COUNCIL.

In submitting the report to the Twentieth Annual Meeting of the Society, the Council regrets that it is unable to state that the progress of the Society has been continued as in former years, the past year having been merely a period of "carrying on." When the war is over and matters become normal once more, it is hoped that the progress hitherto shown will be resumed.

Your Council understands fully that many of the members are in the various theatres of war nobly serving their Country, and that those left on the Rand (equally serving the Empire in producing the essential gold) are in many instances working at high pressure.

Yet now, of all times, it is of the utmost importance not only to the Industry, but to the Empire, that every opportunity and advantage should be taken to improve methods of work, and to devise and improvise means of utilising materials, etc., which in ordinary times would not be considered. Not only so, but every publicity should be given to these methods and devices for the benefit of others not only on this reef but in other parts of the world. It is the duty of all to consider such means as are outlined above as their small contribution to the vast organization of the Allies in combatting Prussianism. The Council acknowledges with appreciation several such contributions which have been submitted to the Society during the year, but there are, no doubt, many others which could be published. Nothing is too trivial, as by open discussion even small things can be improved to the common benefit of all. The policy of secrecy is fatal to progress, and the Council would appeal to the membership of the Society to consider seriously what each one can do to further the objects of the Society, and by doing so aid the cause of the Allies.

The Society needs more members (there

are many on the Rand who take full advantage of our *Journal* without assisting in its publication), more notes and papers, and particularly more discussion on the papers and notes which are submitted. The Council urges that a greater spirit of thoroughness and keenness should prevail in the dissemination of knowledge obtained in the work of the several professions represented in the membership.

Accounts.—The accounts submitted this evening show for the first time for six years a surplus of revenue over expenditure. The reason for debit balances in previous years has been explained in the reports for those years, but this year, as a further shrinkage in revenue was anticipated, the expenditure was carefully scrutinised, and it will be seen that a decrease is shown in practically every item, the total decrease, compared with last year being £239, of which decrease the *Journal* is responsible for £132. An increase is shown in the Revenue of nearly £80, due to the fact that about £190 was received from arrear subscriptions. In view of the crisis through which the world is passing, the accounts submitted can be considered as satisfactory, and with the co-operation and hearty support of the members which it is hoped will be forthcoming, combined with the exercise of care in management, the Council trusts that the Society will be able to continue its useful work for the industry and science generally, without experiencing many untoward effects from the war.

Membership.—During the year 11 Members have been elected, and 5 Associates and 1 Student admitted. 21 Members and 18 Associates have resigned. The loss by death has been 8 Members and 2 Associates, as follows:

Mr. R. G. Bevington (Past-President), Mr. M. Torrente (Member of Council), the loss of both of whom is deeply deplored, Mr. H. F. Olds (Late Member), Mr. C. A. Ferguson, Lieut. F. N. Gerds, Lieut. A. D. Johnson, Capt. R. W. Robinson, and Mr. J. Russell (Members), and Lieut. W. J. N. Dunnachie and Mr. F. J. Pooler (Associates).

This leaves a total membership to commence the new year of 773, as shown in the comparative table, appended, which is a decrease of 33 since the last report. Owing to the extreme difficulty of ascertaining exactly how many members are on active service, and as the Council under these circumstances has no desire whatever

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1900.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.	June, 1909.	June, 1910.	June, 1911.	June, 1912.	June, 1913.	June, 1914.	June, 1915.	June, 1916.
Hon. Members	5	5	5	8	14	9	13	18	18	18	16	17	18	21	22	20	23	25	25	26	26
Life Members		1	1	1	1	1	1	2	3	3	4	7	7	8	9	9	9	10	10	10	9
Members	49	81	101	197	281	291	441	639	680	680	680	683	618	630	731	711	733	618	559	481	463
Associates				180	238	243	281	338	321	314	314	342	357	312	402	380	412	396	351	287	272
Students								4	8	4	4	7	18	20	60	57	22	21	15	2	3
	54	87	107	386	534	544	736	1,001	1,030	1,019	1,018	1,056	1,018	1,021	1,224	1,177	1,229	1,070	951	806	773

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for the year ended 31st May, 1916.

Dr.

1914-15. £ s. d.	EXPENDITURE.	1915-16. £ s. d.	£ s. d.
	To CHARGES (including Postages, Bank		
(104 4 3	,, Charges, Insurance, etc.) ...	78 14 5	
106 0 7	,, PRINTING AND STATIONERY	60 13 4	
29 0 5	,, RENT, LIGHTING AND TELEPHONE	11 19 4	
596 0 0	,, SALARIES	596 0 0	
10 0 0	,, DEPRECIATION ON FURNITURE	20 0 0	
26 5 3	,, *Amounts Written Off as Irrecoverable*		
871 10 6			767 7 1
	,, GRANTS: Seymour Memorial Library	25 0 0	
35 10 0)	Prizes: School of Mines	8 8 0	
			33 8 0
	,, SOCIETY'S JOURNAL —		
(718 18 6	Cost of Reporting, Printing, etc.	548 19 3	
362 8 0)	*less* Received for Sub's. and Advts. after deducting commissions on		
356 10 6	Advts. and Collections	325 4 3	
			223 15 0
	,, BALANCE, Surplus on year's working		116 18 1
£1,263 11 0			£1,141 8 2

Cr.

1914-15. £ s. d.	REVENUE.	1915-16. £ s. d.	£ s. d.
	By SUBSCRIPTIONS		
(709 16 0	Members, current year ...	636 0	
12 12 0	do. arrears collected ..	117 12 0	
286 13 0	Associates, current year ...	268 10 9	
7 17 6	do. arrears collected	74 0 6	
1 1 0)	Students	1 11 6	
			1,098 0 9
1,017 19 6			
9 9 0	,, ENTRANCE FEES		12 12 0
19 10 5	,, INTEREST ON FIXED DEPOSITS		18 11 5
14 19 3	,, SALES OF TRANSACTIONS		12 4 0
201 12 10	,, *Balance deficit on year's working*		
£1,263 11 0			£1,141 8 2

BALANCE SHEET, as at 31st May, 1916.

Dr.

1914-15 £ s. d.	LIABILITIES.	1915-16. £ s. d.	£ s. d.
(163 13 0	To SUNDRY CREDITORS	131 17 5	
26 5 0)	,, SUBSCRIPTIONS paid in advance	22 1 0	
37 9 0	MINING EXHIBITION FUND	39 19 0	
			193 17 5
227 7 0			
	,, REVENUE AND EXPENDITURE ACCOUNT		
(5[illegible]	Balance at June 1, 1915 ..	377 9 9	
[illegible] 20)	*Add* surplus as per Revenue and Expenditure Account, May 31st, 1916	116 18 1	
			494 7 10
[illegible]			
£[illegible]			£688 5 3

Cr.

1914-15. £ s. d.	ASSETS.	1915-16. £ s. d.	£ s. d.
	By CASH AT STANDARD BANK		
(36 12 9	Johannesburg	118 7 3	
14 1 6	London ...	9 5 6	
	,, On Hand, Johannesburg	1 0 1	
50 14 3			128 12 10
405 13 4	,, Cash on Fixed Deposit with accrued Interest	405 13 4	
53 0 8)	,, Cash in Post Office Savings Bank with accrued interest	54 12 1	
458 14 0			160 5 5
70 0 0	,, OFFICE FURNITURE ...		50 0 0
25 8 6	,, SUNDRY DEBTORS		49 7 0
£[illegible]			£688 5 3

We have examined the above Balance Sheet and accompanying Revenue and Expenditure Account with the books and vouchers of the Society, and have obtained all the information and explanations we have required. In our opinion such Balance Sheet is properly drawn up so as to exhibit a true and correct view of the affairs of the Society, according to the best of our information and the explanations given to us, and as shewn by the Books of the Society.

Sgd. ALEX. AIKEN and CARTER, *Hon. Auditors*

JOHANNESBURG, 24th JUNE, 1916.

(Signed) JAMES LITTLEJOHN, *Hon. Treasurer*

(Signed) FRED. ROWLAND, *Secretary*, *Fellow Chartered Inst. Secretary*

to remove any members who are in arrear with their subscription from the Roll, it has been decided to retain on the Roll for another year, those who are at present in arrear for two years.

Journal.—The *Journal* has been published eleven times during the past year, but in the coming year it has been decided to publish it every month.

The procedure inaugurated last year of printing papers in advance not having proved entirely satisfactory, a reversion was made to the previous practice.

The following is the list of papers and notes submitted during the year:—

July-August, 1915.—"Recent Investigations on Dust in Mine Air, and the Causation of Miners' Phthisis," by Dr. J. Moir.
"Loss of Cyanide and Value of Protective Alkali," by Dr. J. Moir. (Contribution).
September, 1915.—"A Description of an Improved Method of Removing the Broken Ore from Stopes," by Mr. M. Weinbren. (Note.)
"Use for Worn Tube-Mill Liner Bars," by Mr. J. M. Dixon. (Note.)
"Presidential Address," by Mr. Jas. E. Thomas.
"Cyanide Consumption on the Witwatersrand," by Mr. H. A. White.
"The Prevention of Hydrolysis in Cyanide Solutions," by Mr. H. M. Leslie.
October, 1915.—"Note on Covering of Sand Dumps with Slime Residue, and Building Slime Residue Dams," by Dr. W. A. Caldecott. (Note.)
"Clean-up Room Practice at the Simmer Deep," by Messrs. W. H. Jane and E. Davey.
November, 1915.—"Osborn Bars for Mortar Boxes," by Messrs. H. A. White and J. M. Neill. (Note.)
"Chief Sources of Accidents in the Witwatersrand Mines," by Mr. Chas. E. Hutton.
February, 1916.—"Use of Welded Cam Shafts," by Mr. Jas. E. Thomas. (Note.)
"Dust Sample Tubes," by Mr. F. W. Watson. (Note.)
"Non-inflammable Explosives Bags," by Mr. H. W. Pursglove. (Note.)
"The Conglomerates of the Witwatersrand," by Dr. E. T. Mellor.
March, 1916.—"A Device for Regulating the Underflow of Classification Cones," by Mr. E. H. Johnson. (Note.)
"A Note on Tube-Mill Liners," by Mr. A. J. Herald. (Note.)
"Notes on Rare Minerals in Madagascar," by Mr. T. P. Waites.
"Analysis of Niobium Titanium Minerals, with some New Tests for Niobium, Tantalum and Titanium," by Dr. J. Moir.
"Some New Methods of Testing for Molybdenum," by Dr. Moir.
"A Simple Method of Calculating the Relative or Percentage Humidity in Mines from Dry and Wet Bulb Readings," by Dr. J. Moir. (Contribution.)
April, 1916. "Concrete Shaft Equipment at the Bantjes Consolidated Mines," by Messrs. W. W. Lawrie and G. Hildick Smith.
"Non-inflammable Explosives Bags," by Mr. R. Murdoch. (Contribution.)
May, 1916.—"Miners' Phthisis," by E. M. Weston (Note.)
"The Recovery of Zinc from Zinc-Gold Slime," by Messrs. R. Fiander and R. Murdoch. (Note.)
"On Some Diseases of the Respiratory Organs incidental to Miners, as portrayed by Dr. Agricola in A.D. 1550," by Dr. J. de Fenton.
June, 1916.—"The Encouragement of First Aid Work on the Mines: Some suggestions based on experience at the Crown Mines," by Mr. A. J. Brett.

Mining Exhibition.—Owing to the war it has not been considered advisable to hold the Fourth Exhibition until times are normal.

Awards and Grants.—A grant of £25 has been made to the Seymour Memorial Library for the purchase of books. In view of the necessity of having the latest and best literature available on industries and manufactures at this opportune time, a Committee of your Council has forwarded a list of suggestions for the purchase of the best books possible, for the favourable consideration of the Book Committee.

The Society continued its policy of offering prizes to the best students in Chemistry and Metallurgy at the Evening Classes, as shown by the results of the Annual Examinations at the School of Mines and Technology.

Visits.—In conjunction with the S.A. Institution of Engineers an interesting visit was made to Vereeniging in November, when the Union Steel Corporation Works, the Victoria Falls and Transvaal Power Co's Plant, and the Vereeniging Brick and Tile Works were inspected. In December an instructive visit was made to the works of the New Transvaal Chemical Co., at Delmore. Both were well attended, and the local manufacture of products useful to the Country was followed with keenest interest.

Roll of Honour.—As previously mentioned, it has been most difficult to obtain a full list of the members on active service, but the following Roll of Honour is made up as completely as possible. The co-operation of members is especially requested in this connection, as the Council desires to make it complete. Two members have been honoured with the Military Cross—Major Ralph Stokes and Lieut. H. R. Ruggles-Brise.

Killed in Action.

Lieut. W. J. N. Dunnachie, Royal Engineers.
Lieut. W. G. Fronde, Imperial Light Horse.
Lieut F. N. Gerds, Royal Engineers.
Lieut A. D. Johnson, 2nd South Staffordshire Regiment
Capt. R. W. Robinson, 5th Royal Irish Fusiliers.

Awarded Military Cross.
Major Ralph Stokes, Royal Engineers and Assistant Inspector of Mines, General Headquarters, France.
Lieut. H. R. Ruggles Brise, Royal Engineers (also served in G.S.W.A.

On Active Service (Rank and Regiment given when known).—
W. Anderson.
Lieut. M. S. E. Archibald, Royal Engineers.
Lieut. M. Aymard, Royal Garrison Artillery.
Pte. M. F. Balfe, 3rd S.A. Infantry.
Lieut. H. Standish Ball, Royal Engineers.
Lieut. R. L. Barratt, 1st S.A. Horse.
Lieut. E. G. Baskett, Oxford and Bucks Light Infantry.
J. P. Beardwood, Mechanical Transport, East Africa.
D. B. Bell, Rhodesian Regiment.
E. Blewett (Overseas).
E. J. Bolitho.
J. P. Caddy, Australian Forces.
Capt. A. F. Dick-Cleland, Royal Engineers.
Lieut. P. A. Cockburn, Royal Engineers.
B. I. Colhngs (Overseas).
E. P. Cowles (Overseas).
C. T. Creed.
Sergt.-Major N. W. Dakin, 10th S.A. Infantry.
Lieut. J. G. Dealy, 12th S.A. Infantry.
Sergt. W. Donkin, 2nd Rhodesian Regiment.
Capt. J. J. Drought, East African Mounted Rifles (Intelligence Officer. *Mentioned in Dispatches*).
O. D. Filley.
B. L. Gardiner, East Africa.
Lieut. T. W. Gilbert, Royal Garrison Artillery.
J. M. Gourley (Overseas).
W. H. Graham, Rhodesian Platoon, 3rd Batt. Kings Royal Rifles.
Tpr. J. E. Halford, King Edward's Horse.
J. V. Harris.
Q.M.S. W. B. Heatley, 7th S.A. Infantry.
Pte. B. A. Johnson, 5th Batt., The Buffs.
C. H. M. Kisch.
Lieut. F. H. Lathbury, Royal Engineers.
Tpr. J. I. Lloyd, East African Mounted Rifles.
Pte A. N. Locke, Mechanical Transport.
Capt. J. C. Mance, Train Conducting Officer, France.
H. McAlpine, East Africa.
Gunner A. R. McCormick, Machine Gun Brigade, East Africa.
J. S. Moir, East Africa.
W. Stanley New, 7th S.A. Infantry (wounded).
J. W. Newbery, 2nd King Edward's Horse.
Lieut. W. Nicklin, Royal Engineers.
Major R. C. Nicolans (Overseas).
Capt. P. N. Nissen, Royal Engineers.
Lieut. E. Pam, Royal Engineers.
Lieut. G. S. Paterson, Royal Garrison Artillery.
Capt. P. F. Patterson, 7th York and Lancaster Regiment (Acting Adjutant).
F. D. Phillips (Overseas).
Major W. H. Pickburn, S.A. Heavy Artillery.
C. K. Pitt, Royal Engineers.
Sergt. W. H. Poynton, 27th Batt. Australian Forces.
Lieut. T. Pryor, Queen Victoria's Sappers and Miners, Madras.
Signaller P. Rabone, 2nd Rhodesian Regiment.
Corpl. A. E. Reynolds, S.A. Squadron, Royal Flying Corps, East Africa.
Lieut. A. Malcolm Sim, Royal Engineers.
Lieut. F. L. Simpson, Essex Yeomanry.
Lieut. G. Simpson, Jr., 18th Durham Light Infantry.
Lieut. L. Solomon, 177th Co., Royal Engineers.
Tpr. A. G. B. Spence, 9th S.A. Horse.
Corpl. N. H. Studolph, 2nd Rhodesian Regiment.
Sergt.-Instructor L. A. E. Swinney, Artists Rifles O.T.C.
H. Thomas, 9th S.A. Infantry.
W. E. Thorpe, S.A.M.C., Overseas Contingent.
Capt. T. G. Trevor, 2nd Mounted Engineers, East Africa.
Capt. F. J. Tromp, 1st Monmouth Regiment.
Capt. R. F. L. Weeks, Royal Engineers.
M. Wemhuen, S.A. Motor Cycling Corps, East Africa.
A. Willcox (Overseas).
Capt. G. W. Williams, Royal Engineers.

On Munition Work in Europe —
R. A. Alston.
W. Cullen.
W. E. John.
A. Milliken
K. B. Quman
Alex. Richardson.

The total, so far as is at present known, is 76 now on service, or practically 10% of the membership, a very gratifying figure.

S. A. Red Cross Society.—As mentioned in the last Report, failing the Government calling upon the Society for any assistance in its prosecution of the war, the Council lent the services of your Secretary and the use of its offices to the S.A. Red Cross Society. In October last the Red Cross Society amalgamated with the Central Good Hope Red Cross Committee in regard to war work, until the termination of hostilities, under the title of Red Cross South Africa, and as such was recognized by the Government as the official body between the public and the Government for all Red Cross work. With the East and Central African campaign under the command of General Smuts, active operations commenced, and heavy demands were made for comforts and extra equipment, which occupied a great part of your Secretary's time, both during and after office hours, and it is gratifying to learn that every demand made upon the Red Cross by the military hospitals, whether within the Union or in other parts of the Continent, has been met in full. Many appreciative letters have been received from those to whom comforts have been sent, not the least being one from a former member of the Society, Dr.—now Lieut.-Col. Donald Macaulay, who is in charge of the Hospital Ship "Ebani."

The S.A. Red Cross Society also arranged the Competitions for the Society's Ambulance Shields, which were again secured by teams from the Crown Mines.

Other War Activities.—At the request of Mr. William Cullen, who is filling with distinction an important position under the Minister of Munitions of War, enquiries were made jointly by this Society and the S.A. Association of Analytical Chemists, to ascertain whether suitable chemists were available in South Africa for munition work overseas. With the kind assistance of the press, over one hundred names were registered, but subsequently it was found that sufficient were available for the purpose in Great Britain.

In November last the Council was approached by the General Manager of Railways, in common with the representatives of other bodies, to attend a meeting, convened at the request of the Government, to consider suggestions made by the British Minister of Munitions regarding the question of the manufacture in this country of munitions, machinery, etc., with a view to relieving as far as possible the manufacturing centres in Great Britain.

Your President was appointed on the Central Executive Committee which was formed, with Mr. McArthur Johnston as his alternate on the General Committee. A Johannesburg Local Committee was also formed, on which Messrs. Meyer and John Watson represent the Society.

The work of the Central Executive Committee has taken up a good deal of the time of your President, and a report of its work has been submitted to Parliament and printed in the Press. With a view to assisting the President, the Council appointed an Industries Development Committee, consisting of Mr. Jas. E. Thomas (Chairman), Dr. W. A. Caldecott, Prof. G. H. Stanley, Messrs. K. L. Graham, E. H. Johnson, A. McA. Johnston, J. A. Jones, F. A. G. Maxwell, H. Meyer, S. H. Pearce, J. Watson, H. A. White and J. A. Woodburn.

Many matters have been discussed, chiefly the local manufacture of shoes and dies, litharge, lubricants, zinc, etc. The Council is gratified to report that at least one practical scheme has materialised, and it is hoped that shortly a plant will be in operation at the Robinson Mine for the manufacture of shoes and dies from scrap.

The Council is indebted to Dr. Caldecott and Prof. Stanley, representing this Society, together with Prof. Buchanan and Mr. Bernard Price, representing the S.A. Institute of Electrical Engineers, and Mr. A. C. Hobby, representing the S.A. Institution of Engineers, who very kindly co-operated in bringing the negotiations to a satisfactory conclusion.

It is much regretted, however, that although members of the Society were notified of the work of the Government Committee on Munitions and Industries, not a single suggestion or idea was received by the Council from any member, for the consideration of the Committee, which would lead to the initiation of local industries or manufactures, despite such work being of great Imperial importance.

A movement, initiated by the S.A. Institution of Engineers, has been made to further scientific research with special reference to its application to trade and industry in South Africa. A conference of the Councils of the various Scientific and Technical Societies was held on the 8th inst, when your Council was well represented, at which various resolutions, which have been published in the Press, were passed to further the movement, and it is hoped that with the assistance of the Government considerable good will result. Dr. W. A. Caldecott, Pr. t. G. H. Stanley and Prof J. A. Wilkinson have been appointed by your Council to represent this Society on the Central Committee.

During the year your Council met the representatives of the sister Societies, concerning a proposed joint building, but so far no definite results can be reported. Similar action was taken with regard to a Joint Students Society, which it is proposed to form when the war is over. Your Society's interests will, however, be carefully safeguarded in any negotiations which may be concluded, and your thanks are due to Profs. Stanley and Wilkinson, who represented your Council in both these matters.

Patents Bill.—At the request of Mr. Emile Nathan, M.L.A., the Patents Bill then before Parliament was considered, and certain suggestions were made by representatives of this Council, in conjunction with representatives of the S.A. Institution of Engineers and S.A. Institute of Electrical Engineers, to the Government and Mr. Nathan, which it is hoped will prove of value to those interested in Patents, the Minister of Justice having promised to give the suggestions every consideration in the regulations made under the Act.

International Engineering Congress.—The Society was represented at this Congress in San Francisco by Messrs. Chas. Butters,

G. M. Taylor and W. Neal (Corresponding Members of Council), and an appreciative letter has been received from the Congress regarding their services.

University Bills.—Your Council, in conjunction with practically every other public body, made representations to the Government regarding the University Bills, which were recently dealt with by Parliament, and is now represented by Prof. Wilkinson on the Committee which has been formed for the purpose of securing a University for the Rand.

In addition, the Council approached the Government and members of Parliament for the Rand and other parts of the country, urging that steps should be taken to make more definite provision in the Bills for scientific research. Although nothing could be done in that respect, it is expected that the matter will not be overlooked by the bodies which are being formed to govern the several Universities.

General.—During the year 10 Ordinary and 1 Special meeting of the Council were held, as well as the usual number of meetings of the Editorial, Finance, and the special Committees already referred to. The attendances at the Council meetings, which averaged 12 per meeting, have been as follows:—

J. E. Thomas ...	11	C. Toombs	9
R. A. Barry(1) ...	1	M. Torrente(2) ...	3
H. A. White ...	8	F. Wartenweiler(3)	1
J. A. Wilkinson	8	F. W. Watson ...	8
J. Littlejohn ...	3	J. Watson	8
J. S. Cellier... ...	8	E. M. Weston(3)	2
E. H. Croghan ...	5	A. Whitby	6
K. L. Graham ...	3	J. R. Williams ...	1
J. Gray (2)	8	W. A. Caldecott	5
W. W. Lawrie(3)	3	A. McA. Johnston	2
H. St. Martin(4)	0	J. Moir	1
H. Meyer	9	W. R. Dowling	6
E. Pam(5)	1	G. H. Stanley ...	9
G. Hk. Smith(6)	6		

(1) Resigned September, 1915.
(2) Elected Vice-President in October, 1915.
(3) Elected February, 1916.
(4) Resigned November, 1915.
(5) Resigned February, 1916.
(6) Elected September, 1915.
(7) Died 17th November, 1915.
(8) Elected February, 1916.
(9) Elected November, 1915.

Your thanks are due to the Corresponding Members of Council and to the members of the various Committees of the Council for much valued work in the interests of the Society.

The Society is again indebted to the local Press for reports of its proceedings, and also to Messrs. Aiken and Carter, Hon. Auditors, for the painstaking care with which the audit of our books is carried out.

The President: In moving the formal adoption of the Report, I have prepared a few remarks which I consider applicable to the present time. I do not want you to take this in the nature of a valedictory address; an item which, I am glad to say, has gone out of fashion in this Society at all events.

I think the Society may be congratulated on the manner in which it is weathering the storms caused by the present world crisis. The decrease in the number of members is regrettable, but must be expected in times like these.

As this is the close of my term of office as President I wish to take advantage of the opportunity of thanking the Vice-Presidents, the Members of Council and the Secretary, Mr. Rowland, for the hearty support they have accorded me in conducting the affairs of the Society. I also desire to thank those of the Past-Presidents who have shown their continued interest in the Society by attending the Council Meetings, acting on Sub-Committees, and generally helping with suggestions and advice. The good work of the Finance Committee is deserving of special mention. It is only natural that the activities of the Society should be curtailed at such a time, and the task of making ends meet with a sadly diminished revenue has been no small achievement. I would appeal to those whose subscriptions are still outstanding to make a special effort to pay off their arrears. The liability was voluntarily undertaken, and I am sure the members of the Society have no wish to repudiate their obligations. Such conduct may befit our enemies, but surely not us.

The question of applying the lessons learnt by the great war at present raging, lessons which are being written every day in the blood of our comrades and fellow citizens of the Empire, is occupying the grave attention of thoughtful men in Great Britain and the Dominions. It is now generally realized that our methods of education and lack of properly organized research work have nearly cost us our very existence. The remedies to be applied are being discussed by nearly all the scientific and technical societies of the Empire. As mentioned in

the Annual Report, a combined effort is being made by such societies in South Africa to promote scientific and industrial research in the Union and Rhodesia. This is most certainly a subject that must command the whole-hearted support of everyone interested in the welfare of this, our Country.

The Government has passed the University Bills brought before it, but I trust it will go further and follow the example set by other nations of sending young men and young women, who have shown marked ability, to study methods employed in other countries, so that this Country may benefit by the experience so gained.

I understand that a precedent for this has already been established, by the sending of young men to study conditions of agriculture abroad.

The present war has been called "An Engineers' War," but it must be admitted that the chemist, the metallurgist and the miner are also playing parts equally important, and we must see to it that this Country shall not be left behind, and one might say, defenceless, for lack of training the rising generations in the arts and sciences that mean so much in peace as well as war. In that training I would strongly advocate that the business side should not be neglected, so that the scientific and technical knowledge acquired may be economically applied. The day has passed when the term "Scientific" implied an almost complete lack of business knowledge, since business is now a science.

Considering how much the world owes to scientific and technical men, it seems very strange that such men should have so little say in the government of their countries. The science of governing seems almost to be the special preserve of men trained in law. Though eloquence and the art of quickly detecting flaws in an opponent's deductions are essential, it is equally essential that scientific and technical knowledge at first hand should be brought to bear on the affairs of the Country.

I greatly regret that the Notice re Munitions and Industries, issued by the Council on the 3rd December last, calling for suggestions from members for developing the resources of this Country should have met with no response. Though it is not given to all of us to take an active part in fighting our enemies, it is surely our plain duty to do what we can to assist those who are fighting our battles, and to render their sacrifices not in vain by helping to make this Country as self-supporting as possible.

A little is being done at present in this direction, as you will have observed from the Annual Report, but I fear that only meagre results can be expected until the active sympathy of the Government is obtained, and a department established, whose officials shall devote their whole time and attention to the furtherance of scientific and industrial research, and who shall work in accord with similar establishments in other parts of the Empire.

When industries are established in this Country, as they will be sooner or later, I trust that the old-fashioned and stultifying practice of preserving intense secrecy as to processes, etc., will be utterly discountenanced. Everything is gained by the free exchange of ideas and discussion of methods. Secrecy is apt to make one think that concealment is the end desired.

I do not think I have very much to add. Much has been said at the meetings of various scientific and technical societies held lately about the need for scientific research. I am sure every member of this Society, who has the welfare of the Country at heart, will do all he can to further the movement in that direction.

Prof. J. A. Wilkinson (*Vice-President*): I have very much pleasure in seconding the adoption of the Annual Report, which has been read by the Secretary. During the past year I have served on the Finance Committee of the Society, and I think that the Society can congratulate itself in having to guide it in its finances, a Treasurer who has been doing this work for many years and who has brought us through this crisis because it is a crisis—so satisfactorily. The work of the Society has been detailed in full, both in the Annual Report and in the remarks which have fallen from the chair, and I can add nothing but to commend this report to you for careful study. I am sure it will show you that the men whom you chose last year to carry on the work of the Society during that year have not failed in that duty.

The President: The Report is now open for discussion. If there is no discussion, will you take the Report as agreed to.

The Report and Balance Sheet were adopted unanimously.

SCRUTINEERS' REPORT AND DECLARATION OF ELECTION OF OFFICERS AND COUNCIL.

The Secretary: The following is the Scrutineers' Report on the Ballot for the Election of Officers:

Johannesburg,
28th June, 1906.

The President,
Annual General Meeting of Members.

Dear Sir,

We have to report that we have this day examined 85 ballot papers, handed to us by the Secretary.

Our scrutiny enables us to declare the result of the ballot as follows:—

President.
Prof. J. A. Wilkinson (unopposed).
Vice-Presidents.
Mr. H. A. White.
Mr. J. Gray.
Mr. G. Hildick Smith.
Hon. Treasurer.
Mr. J. Littlejohn (unopposed).
Members of Council.

Mr. A. Whitby.	Mr. C. Toombs.
Mr. K. L. Graham.	Mr. J. Chilton.
Mr. F. W. Watson.	Mr. H. R. Adam.
Mr. J. Watson.	Mr. H. Meyer.
Mr. W. W. Lawrie.	Mr. F. Wartenweiler
Mr. E. M. Weston.	Mr. F. G. Macdonald.

Yours faithfully,
Sgd. S. Newton; A. Thomas; P. T. Morrisby; E. A. Osterloh.

The President: On your behalf, as well as my own, I wish to congratulate your new President, Prof. Wilkinson, the Vice-Presidents, and the Members of Council, on their election to office, and also to wish them a very useful and prosperous year.

Prof. J. A. Wilkinson (*Vice-President*): I have accepted the office, the highest which it is in your power to bestow, to which you have elected me this evening, and it is now my duty to express my sincere thanks for the honour done me. It is, however, not only my duty, but at the same time a pleasure since, during the period in which you have selected me as one of your administrators on your Council my experiences have been of the happiest kind, and I look forward with renewed hope to the year that is before us. Also with a great amount of satisfaction in that you have accorded me such support as I myself would have desired in electing the Council which has just been announced. The work of this Society is writ large in the annals of the mining industry of this town and of South Africa, and it will be my earnest endeavour during my forthcoming year of office to increase its utility, widen its influence, and strengthen the bonds, which bind its members together.

GENERAL BUSINESS.

VOTE OF THANKS TO SCRUTINEERS.

Mr. H. A. White (*Vice-President*): I think it is my duty, as well as a great pleasure, to propose a very hearty vote of thanks to the scrutineers who have conducted their arduous and unpaid duties for us, as they have done in past years, and to thank them very heartily for the work they have done.

Mr. S. Newton (*Member*): On behalf of the scrutineers, I beg to thank you for your kind remarks.

ELECTION OF AUDITORS.

Mr John Watson (*Member of Council*): I have much pleasure in moving a hearty vote of thanks to the Auditors for their work during the past year, and to propose their re-election for the coming year.

Mr. J. R. Williams (*Past-President*): I have much pleasure in seconding the motion.

The resolution was agreed to with acclamation.

THE ENCOURAGEMENT OF FIRST AID WORK ON THE MINES: SOME SUGGESTIONS BASED ON EXPERIENCE AT THE CROWN MINES.

By A. J. Bell.

The object of this paper is to give as shortly as possible some suggestions with a view to arousing increased interest in first aid work on the mines. I do this with a good deal of reluctance, and only in the hope that it may arouse discussion, and thereby bring forward some new and useful ideas, and not from any impression that what has been done at the Crown Mines, is necessarily the best that can be done in this way. A further reason is that the request comes from this Society, and as I take a considerable interest in first aid work, I naturally recognise fully the part it has played in the furtherance of this work, both by presenting shields for competition by ambulance teams on the mines, and also more especially in

the formation of the South African Red Cross Society.

Possibly the simplest way to start a discussion will be to give a short history of what has been done on the Crown Mines, and to try and explain the reasons that have led to these steps being taken.

The first occasion on which I was made to realise the necessity for a more active interest in first aid work was several years ago, when the doctor one day brought to my notice the fact that an employee who had met with an accident, in which, I think, both his legs were broken, was brought to the surface on a stretcher in a skip without any preliminary treatment. On further enquiry into this matter I found that the one idea in the minds of the men concerned was, that, as the man was seriously injured, the important thing was to get him into the hands of a medical man with the shortest possible delay, and, as there was no one at hand with any first aid knowledge, the unfortunate individual was hurried to the surface as he was. I was at once struck with the fact that to any one who had no training or understanding of the nature of a man's injuries, this was a perfectly natural procedure and one for which it was really not possible to blame the men concerned. It therefore appeared that the best way to avoid such occurrences in the future was to ensure that, when an accident occurred, it should be possible to summon some person with a knowledge of first aid work within a short period.

The whole question was then discussed with the Medical Officer, who urged strongly that some serious attempt should be made to put first aid work on a more satisfactory basis. As a result of this he was asked to put forward some scheme which should form the basis of an ambulance organization for the mine.

One of the first points was, that he must have the direct and active co-operation of the management, in which case he was convinced that his own enthusiasm would ensure success. It was not agreed, largely in order to show that the management did really take an active interest in the movement, that a general meeting should be called at which the outline of the scheme agreed upon by the Medical Officer and the management should be announced.

The chief points in this scheme were briefly as follows:

That there should be one Chief Ambulance Officer for the mine, and that under him there should be subordinate Ambulance Officers, either at every shaft or at such points that they could reach any shaft within, say, five minutes. In the case of all these men, they should be men ordinarily in the employ of the Company, who would continue to work in their usual capacity and would be paid a bonus for keeping themselves thoroughly efficient and up-to-date in first aid work, and for attending to any accident that occurred in their particular section of the mine.

In the particular case of the Crown Mines the Hospital Orderly has been appointed Chief Ambulance Officer and the Clerks of the Mine Captain's offices Subordinate Ambulance Officers. It was laid down that these Ambulance Officers should be responsible for keeping the first aid equipment at the shaft top and underground in thoroughly good order. Further, that when an accident occurred the injured person or persons should be left where they were unless, of course, their position was dangerous, until such time as the Ambulance Officer at the shaft head had been sent for. It was his duty then to proceed at once to the scene of the accident with the first aid box, having previously telephoned for the Chief Ambulance Officer if the accident was of a serious nature. At the meeting this scheme was put forward and it was also pointed out that the directors were wholeheartedly in favour of doing everything that could be done to lessen the pain and distress caused by accidents, and had therefore generously allowed a sufficient sum to provide a first class equipment for the use of the ambulance classes and ambulance teams, and further, had promised a sum of £75 per annum, to be awarded as follows:—(1) £30 for a prize for the best surface team on the mine. (2) £30 for the best underground team on the mine, and (3) the remaining £15, for the better of the two winners of the previously mentioned prizes. I may state that it has since been under consideration whether it would not be advisable to allocate the £15 prize as a second prize for underground teams instead of giving it, as at present for the better of the two, surface and underground, winning teams. The reason for this suggested alteration is that, owing to the very much larger number of accidents occurring underground, the work to be carried out by men having first aid certificates is much greater below than on the surface, and hence it is more important to encourage the efficiency of the underground men. The original

idea was, of course, that there should exist a considerable amount of rivalry between surface and underground men, which this prize would encourage and therefore stimulate their keenness and efficiency. This, I think, it most certainly does, but, probably, the point above mentioned is the more important. All who attended the meeting and possessed either out of date first aid certificates, or none at all, were asked then and there to sign on for attendance at ambulance classes, and the names of men, who would volunteer to get together ambulance teams with a view to entering for the competitions mentioned, were called for.

As a result of the meeting some most enthusiastic and capable first aid men were discovered, and largely through their influence, as well as that of the Medical and Ambulance Officers, the movement was given a thoroughly satisfactory start. A committee was formed to carry out arrangements, etc., for the competitions, and to handle first aid matters generally.

The scheme outlined above has been and is still in operation, and has, I think, been thoroughly satisfactory. It was found later, however, that in certain parts of the mine, where old workings were being re-opened and reclamation work carried on, long distances separated the working places, and that as a rule there were only a few workmen employed at each of these working places. Under these conditions, where operations are scattered over large areas, it was found best to pick out a few men with ambulance certificates, who took an active interst in the work, and pay them an extra 1s. per shift. For this extra payment it became part of their duty to take charge of any accident in the section of the mine allotted to them. This not only served to get the work done efficiently, but encouraged other men in the section to take up first aid work with the idea of being competent to fill any vacancy that might occur amongst the men drawing this bonus. As a result of the above organization of the work, the Medical Officer informs me that every case that comes to Hospital as the result of an accident is found to have received thoroughly efficient treatment before arrival. I am confident that very many accidents would under other circumstances have meant much more serious wounds and, therefore, resultant consequences to employees, and also that many additional lives would have been lost but for the treatment injured employees receive at the earliest possible moment before being brought to the surface. It is, further, a very great comfort to the management to feel sure that every employee, who has the misfortune to become involved in an accident, will get proper treatment within a very short time of the occurrence.

More recently it has been felt on the Crown Mines that the first aid work was being very largely confined to those who were by law compelled to obtain certificates, and to certain others who realised the danger of accidents and, therefore, the necessity of being prepared to deal with them. The management now see that first aid teaching has a further use besides the actual knowledge it gives of how to relieve pain and suffering, and believe that this side of the question is of the utmost importance. As stated above, it is largely those, who realise the danger of accidents, who take up first aid work, and it is now thought the converse will be the case, and that those who have been taught to deal with injuries caused by accidents will be more apt to realise that such accidents may conceivably happen to themselves or those directly under their supervision. Particularly is this the case of gassing accidents, for it seems only reasonable to suppose that, after a course in the treatment of gassing in which it is very fully pointed out that it usually occurs in dead ends, the first thing to be done is to turn the air on in such places, and any man going to work in such a place will at once think of the possibility of gassing, and be on the look out for it, and will therefore always see that his air pipe, at any rate, is in order. Similarly the knowledge of the serious injuries caused by other accidents is likely to make those who previously did not give it a moment's thought or reflection and, as a result, were inclined to be careless, more careful. If there is anything in this aspect of the question, then it is most important from the point of view of the prevention of accidents, that every man underground shall, if possible, have the education appertaining to the results of accidents, that is given him in a course of first aid instruction.

The more unthinking a man is, then the greater the necessity to attract him to a first aid class. With this object in view it has been decided on the Crown Mines that, in future, any underground employee who attends the full course of lectures for a first aid certificate will be refunded his fee for the classes, namely, 10s., and that further, any underground employee who obtains a first aid certificate will be paid a bonus of

£2 10s. It is also proposed that a further bonus of £1 10s. will be offered to any underground man obtaining the advanced certificate; these bonuses, however, do not apply to officials who have to obtain a certificate under the regulations.

It will be seen from these remarks that, what we are trying to do, is to attract men to the classes so as to enable them to have the benefit of the education, and in that way possibly effect the prevention of accidents rather than merely trying to increase the number of the men in the mine with first-aid certificates. That this step is appreciated by the men concerned, is proved, I venture to think, by the following gratifying letter addressed to the Management, Crown Mines, Ltd., dated 15th May, 1916, and signed by 35 of the employees who attended the last class:—

Sirs,

"We, the undersigned members of the 1916 Crown Mines First Aid Class desire to cordially tender you our very hearty thanks for the manner in which you have displayed interest in this matter on our behalf.

Offering, as you have done, to reimburse the 10s. entrance fee to each member who attended all lectures, as also a bonus of £2 10s. to each of those underground men who passed the medical examination and obtained a certificate, we believe to be unique in its way, and consequently appreciated the more. We have reason to believe your lively co-operation, combined with the substantial offer, has gone a long way in obtaining results."

We are, Sirs,

Yours gratefully.

There are many other smaller points which tend to encourage the movement, among which may be noticed the following: The managers make a point of attending the classes in order to obtain a certificate, although not required by regulation to hold one. Also the mine overseers of the various sections are allowed to use their discretion in permitting ambulance teams at various intervals to have their practice in mine time instead of in their own time. The daily paid men are granted their pay for the day, if the services of a team are required at any function in town or elsewhere. Encouragement is given to all underground employees, who show any interest in first aid work, at all times to carry first aid pockets with them.

It has been suggested that, as a rule, some certificated layman, such as a manager, an underground manager or other official should be asked to co-operate with the doctors as one of the judges at the competitions. Such a step, I am assured, would be welcomed by the medical men acting in that capacity.

It has also been suggested by Dr. Irvine that it would be very advisable that all first aid equipment, such as stretchers, first aid boxes, etc., should be standardized on all the mines. The idea which seems to me to be quite on the right lines, is that any man trained up in the use of certain equipment on any one mine would be immediately efficient on any other mine to which he transferred his services, provided this equipment was standardized and therefore identical.

In conclusion, I may say that in my opinion, as the result of experience on the Crown Mines, there are four essentials in the successful organization of first aid work.

Firstly. The sympathy of the directors, who must be willing to agree to the necessary expenditure.

Secondly.—The direction and aid of a medical officer enthusiastic in the cause.

Thirdly.—The active co-operation of the management, which will naturally be followed by that of all loyal officials.

Fourthly.—The services of an official as Chief Ambulance Officer, who has a thorough knowledge of the practical side of the work, and is willing to devote a large part of his time outside his real working hours, to the assistance of the doctor, who delivers the first aid lectures, by taking the classes through additional practical work, and, also to the instruction and help of the various teams that get started on the mine.

It is unfortunate that I am unable to give any figures that would tend to show the improvement that has followed these steps. A comparison of the number of days spent in hospital by accident cases is not possible owing to a difference in procedure in the hospital practice during the period, while a comparison of the monies paid out in accident compensation before and after the organization of first aid work is also impossible owing to the varying number of employees, and the fact that the amount of compensation payable for injuries to natives was altered about the time these steps were taken.

I feel it would not be right to close my remarks without stating that the credit for the foundation and organization of the scheme on the Crown Mines is entirely due to Dr. Irvine, whose strenuous advocacy of

first aid work is well known. In this he has always been seconded most heartily by the Chief Ambulance Officer, Mr. Anderson, and your energetic and enthusiastic Secretary, Mr. Rowland in his capacity as Secretary of the Red Cross Society.

The President: In thanking Mr. Brett for his paper, and before calling for a formal vote of thanks to him, there are a few remarks I would like to make. I quite agree with Mr. Brett that it is a pity the law does not call upon mine managers to hold the ambulance certificate, required of other officials. There is also a strange omission in the law, which I do not quite understand, that is, that reduction works managers do not require to have certificates. As it is, I think it is a subject that requires careful attention. I might mention that I have taken my certificate, and am very pleased I did so. It is a subject that every man, not only those who work on the mines, but those who go about in every day life, in motor cars, and that sort of thing, should know something about. If we only had a little more of the proper knowledge applied to this subject, I think quite a lot of good might be done. I would also recommend that the activities of the Red Cross Society be used in the direction of stirring up our managers to take more interest in this work. Our medical men tell us they are too busy to give us their services. The Red Cross Society tells us, on the other hand, they can supply lecturers at any time we want them. I will now call upon Prof. Wilkinson to move a vote of thanks to Mr. Brett for his paper.

Prof. J. A. Wilkinson *(Vice-President):* It is common knowledge, as far as the members of this Society are concerned, that three years ago, on your behalf, I took up the subject of ambulance work, with which the paper just read deals so fully, and the outcome was the formation of the South African Red Cross Society. The record of that Society, I venture to say, speaks for itself, but owing to my intimate connection with it during this period I have a peculiar pleasure in rising to propose that we accord to Mr. Brett our warmest thanks for having given us the opportunity, not only of hearing what can be and has been done here in this direction, but also of renewing our interest in this work by discussion. Personally speaking, it is my intention a little later to take the opportunity which has thus presented itself and now, therefore, I would ask you to accord this vote with acclamation.

Mr. H. A. White *(Vice-President):* It is with very great pleasure that I rise to second this vote of thanks to Mr. Brett for the paper he has read here to-night. I think he has illustrated the fact that, where there is a will there is a way; the directors of the Crown Mines have shown their willingness, and the management and the medical department of the Crown Mines have shown the way in which provision for first aid should be made. It is obvious it was not an easy matter to get that organization into working order as it is at present. The author has referred to the point that it is not easy to demonstrate in figures the benefits derived; but still, after all, that is not an argument that too much stress need be laid upon. I do not know that anyone can undertake to prove that England is going to get anything out of the present war commensurate with the tremendous expenditure of life and treasure. There is a duty which directors have here, beyond seeming dividends for their shareholders. In such ways bread cast upon the waters will always be returned even if after many days.

Mr. T. P. Waites *(Member):* It is a case of the two extremes meeting in me: Mr. Brett, as a mine manager, and I, as a miner. It is very pleasing to learn that so much interest is being taken in an essential subject, such as this is, by managers of mines. My experience, as an underground worker, has been that this interest is most noticeable by its absence. Judging by myself and me, I speak subject to correction, I think, too, that there would be far less use for first aid if there was a little more supervision—skilled supervision—on the mines. I do not like to make any reflections, but I think that our Government and the Minister of Mines and, probably, the management, are equally responsible for, what I might term, a lack of essential supervision. As for instance, in reading Mr. Hutton's paper one sees that the majority of fatal and other accidents in our mines is caused by falls of hanging. I know that risks are taken, and I know that, however good the management may be, there is still, in even the best managed and supervised mines, a certain element of luck; and this element of luck, I believe, is rather counted on too much. From my own experience, and noticing the after results of visits from

mining inspectors and their assistants. I am satisfied that if the number of mining inspectors were increased, the number of their assistants increased, that if unnotified visits were paid to the mines, and if a little more attention were paid by the inspectors of mines as to whether various places were fit to work in, a great deal of this element of luck would be left out.

I am, at present, attending a course of lectures given under the auspices of the Red Cross Society. We are a class of about 30; but I am quite satisfied that, in that class, half the terms which are used by the lecturer are not understood by those who are listening to him; and I venture to suggest that a little simplicity in terminology, both in the books and in the expressions used by the lecturer, would tend to make these lectures more clearly understood, and would probably attract a far larger number of underground workers to attend these lectures.

Dr. L. G. Irvine (*Member*): It is, I hope, a true sign of the times that we should have the General Manager of one of the largest mines on the Rand submitting a paper on the "Encouragement of First Aid Work on the Mines." It shows the advance which the movement (and the mine managers) have made in the past few years. I have always urged that first aid work should be everywhere organised as part of the routine working of every mine, and that it should constitute a regular recognised department under the direction of the medical officer and under the personal care of the managers. Mining has its inevitable casualties just as fighting does, and just as behind the fighting line comes the army ambulance corps, so the industrial army also requires its trained ambulance organisation. This has been done on the Crown Mines, in the way described by Mr. Brett, and the management of the mine are to be congratulated on the thoroughly sympathetic spirit in which they have co-operated in the movement.

When I first took up first aid work some years ago neither the mine managements nor the men realised the immense good it could accomplish, not only that it might not infrequently save a life, and more frequently a limb, that it would always diminish suffering and very often reduce the amount and the period of subsequent incapacitation, but that, quite apart from its humanitarian aspect, it paid in hard cash. I paid the management in lessened expenditure for compensation, and paid the workmen in reducing the time during which no wages were coming in.

First-aid used to be looked upon as the business of a few well-meaning individuals, who were apt to lose their heads when called upon to do any real work. It was no part of the business of a genuine miner. The miner's job was to do mining, not to "be a doctor." And so the sort of thing happened that Mr. Brett has described. A man with both legs broken was hustled up the shaft without treatment—think what that means—or a man who was seriously gassed and required immediate artificial respiration, was brought up to the surface and was perhaps found to be dead when he got there, although he was alive when first brought out of the danger zone, or a man suffering from severe haemorrhage was hoisted up without any attempt being made to stop the bleeding. I have known of all these things happening, and not once only. And all because the only duty of the underground man appeared to be to get the injured person into the doctor's hands as soon as possible. But getting him there takes some time, and the doctor with the best will in the world could not always be at the shaft head just when he was wanted. There is always a gap, an interval, often a long one, between the time the man is hurt in a mine and the time he can come under a medical man's hands, and in that interval lives may be lost and things may go wrong. It is just this interval that first-aid is designed to fill.

Hence the necessity for the appointment on every mine of ambulance officers who will be at hand at once when called upon, so that everyone injured underground shall have his injuries treated by first-aid methods underground and on the spot. Things are different now on the Crown Mines. There is a most capable and experienced chief ambulance officer, there are six subordinate ambulance officers, and in addition there are over a hundred and thirty certificated first-aid men on the property. The chief ambulance officer tells me that he receives from the latter most willing and efficient assistance. I should like to see this system extended to every mine on the Reef, and there is no reason why it should not be done. But, as Mr. Brett points out, the directors must help us as they have done on the Crown Mines by active furthering of the scheme, and good results will never be attained without the active participation of the manager

and mine officials and of the mine medical officers. Every mine manager should make a point of taking a first-aid certificate. Mr. Brett has mentioned a suggestion that certificated and experienced mine officials would prove useful members of the judging bodies at inter-mine competitions. I think the suggestion is a sound one, but perhaps it would be desirable that such men should hold at least an advanced certificate in first-aid. Inter-mine competitions are most valuable, not only in stimulating interest in the movement, but in bringing out under the stimulus of competition new ideas and methods of treatment.

Mr. Brett has mentioned also an important point, the standardisation of equipment. We have given much consideration to the best equipment of a standard first-aid box for mine use and to the best type of underground stretchers. I think we have arrived at satisfactory results in both cases. Efficient first-aid work is impossible without proper equipment. Time is lost in the preparation of make-shifts, which in any case are not so good as proper appliances. A very large number of the miners on the Crown Mines also now carry a first-aid packet of emergency dressings similar to that used in the field. Issue of these was made on the suggestion, I believe, of Mr. Anderson. I would endorse Mr. Brett's statement that the teaching of first-aid is likely to lead to the prevention of accidents, by making men realise better what accidents result in and how they occur. The "First Aid" man becomes naturally the "Safety First" man. The two go hand in hand.

There are only two other things I should like to say. First, regarding the criticism of Mr. Waites that lecturers on first-aid are apt to use terms which are not understood by their hearers. I think that all lecturers endeavour to make their teaching as simple as possible. They certainly should always try to do so. On the other hand, although the description of anatomical and surgical matters given in a first-aid course is very elementary, it cannot be simplified beyond a certain point. No technical material can. If Mr. Waites were to lecture on mining to a class of ordinary individuals who were not miners, he would find he could not avoid some use of technical terms which his hearers had not heard before. It is really astonishing how ignorant very many people are regarding the simplest facts about their own bodies. Nobody should take up the study of first-aid unless he is willing to take some trouble over it and to study it seriously. For the layman it *is* a study, but it is a study which any intelligent man can and ought to master. The Red Cross Society has purposely set a fairly high standard in its teaching, because certificates which can be gained without any trouble are not worth having, and should not be granted, since the smattering of knowledge so gained is promptly forgotten. We want to turn out efficient first-aid men. A badly trained man is not only useless—he is often a danger. Personally, I never pass a man who cannot show me that he can arrest serious haemorrhage promptly and efficiently, and perform artificial respiration correctly. We want our certificates to be worth something, not only to their holders, but to their fellow-workmen. It is much wiser to have on a mine a few well trained first-aid men than a large number of people imperfectly trained.

The second thing I want to say is that I think the mining community owes a great depth of gratitude to this Society for having taken for several years an active interest in the furtherance of first-aid. The shields presented by the Society as prizes in the annual inter-mine competitions have proved a great stimulus to efficient first-aid work. Further, the Chemical, Metallurgical and Mining Society took a leading part in the foundation of the South African Red Cross Society. I think that that step has been amply justified, and its success has been very largely dependent on the fact that the Red Cross Society has had as its first Chairman and its Secretary two of your own officials, Professor Wilkinson and Mr. Fred Rowland. Both have rendered untiring service to the new movement, not only in its ordinary routine work of teaching first-aid on the mines, but in the larger war work which has since devolved upon the Red Cross organisation. I know that these gentlemen have rendered a great public service in each of these directions.

Finally, I hope that the suggestions made by Mr. Brett will not fall on deaf ears, and that with the help of the "Prevention of Accidents Committee" of the Rand Mutual Insurance Society, which is now taking an active part in the first-aid movement, the Witwatersrand may soon have no cause to fear comparison in respect of provision for treatment of the injured with any industrial area in the world.

The President: Arising out of the re-

marks of Mr. Waites and Dr. Irvine, there are one or two things that struck me, particularly Mr. Waites saying that lecturers used expressions and terms not intelligible to the ordinary man, and also the point brought out Dr. Irvine, namely, that there was nothing derogatory or unmanly in one man helping to save the life of another. There are two organizations which have been in existence, one for quite a long time, and the other a comparatively new one, for teaching boys to be manly: One, The Boys' Brigade, which has been in existence for 25 years, and the other, The Boy Scouts. They have both made a great point of teaching boys first aid. Surely, if it is within the capacity of boys whose ages range from 10 to 16 to understand first aid work it is within the capacity of a grown man to understand it. It seems rather strange that the largest mine on the Reef and practically one of the smallest mines, namely, the Crown Mines and the Luipaardsvlei Estate, should have two of the keenest competing ambulance teams on the whole Rand; it shows that a great deal can still be done by the mines which fill the intermediate stages.

The vote of thanks was carried by acclamation.

THE CHIEF SOURCES OF ACCIDENTS IN WITWATERSRAND MINES.

By Chas. E. Hutton, M.Inst.M.M., Inspector of Mines (Member).

(Printed in Journal, November, 1915.)

REPLY TO DISCUSSION.

Mr. Chas. E Hutton *Member*: Mr. Weston remarked that my paper was subjected to some disadvantage as the question of "accidents due to explosives" had already been fully discussed; the subject, therefore, not being altogether fresh, discussion would be meagre. Admitting that "accidents due to explosives" have been in the past dealt with, I am disappointed that discussion on other classes of accidents, particularly those brought about by falls of rock, the most prolific of all causes, has not taken place.

Mr. Pam admits the necessity of firmly securing the capped fuse to the primer, and states that string is specially issued for this purpose. I am not, however, satisfied that all managements give sufficient attention to this all important regulation. Since reading my paper, another fatal accident has been recorded, in which a miner and his native helper were killed through not firmly attaching the capped fuse to the primer. There is, in my mind, no doubt that if miners would always observe this regulation, accidents occurring whilst "charging up" and through "misfires" would be greatly reduced.

Prof. Cellier and Messrs. Pam, Pooler, White and Price all deal with the question of instructing the native workers, so that they may be made aware of certain dangers in connection with their work.

At the outset I would entirely agree with Prof. Cellier and Mr. Pooler that it would be most unfair to expect the Inspectors to go round teaching the 150,000 natives, as suggested by Mr. White. I would even go a little further and say not only would it be unfair but practically impossible.

Mr. Weston would like to see notices in various native languages posted in the compounds, stating how the dangers of underground work must be avoided; such posters might have some good effect, but if they could be made pictorial they would be much more impressive.

The best method, as Mr. Pam observes, is to verbally warn new boys of the dangers when they arrive at the compounds, and again to repeat this warning after the natives have become somewhat acquainted with underground conditions; this, in conjunction with the miner, who is in daily contact with his gang, assisting the management by further impressing upon his boys the various risks to be avoided, should lessen the accident rate.

I do not quite agree with Mr. Price when he states that the miners must be accomplished linguists to acquaint the "boys" working under them of the dangers of the mine. "Kitchen Kafir," aided by gesticulation is usually quite sufficient to make the native comprehend; as a matter of fact, the miners of to-day, generally speaking, are very fair native linguists.

I do not take exception to Mr. Price's remarks on discipline, but they call for some comment. In all probability he did not mean it to be inferred from his words that men are best used when the constant endeavour is to get as much out of them as possible. Admittedly no self-respecting man in charge of a gang will allow his natives to take "rest cures" during their working hours, but on the other hand, no man can expect perpe-

tual high pressure of the human machine. To obtain the best results in the long run "dogged does it," and in the end the steady pull, and the long pull, gives the best results. Nobody should drive any workman, but what they should do is to encourage a fair average amount of work daily.

Mr. Price, who claims to be an enthusiast for discipline, states that the manager's instructions did not appeal to him; whether the order appealed to him or not, it was his duty to carry out the instruction, and so conform with the system, which, no doubt, he did. Discipline is discipline, and the system is system. The system may not perhaps be as good as it might be, but the only means of establishing discipline is by general conformity to the system. That is why the officials of the Mines' Department press for the observance of the mining regulations.

In regard to Mr. Price's opinion that one of the chief sources of accidents is "the almost lack of discipline among the natives by their overseers," I would point out that the Mines Department records do not support this.

Mr. J. R. Williams states that I had shown "very clearly that the want of proper detonators was responsible for something like 147 deaths and a large number of other serious accidents," admitting, however, that I had not said so in as many words, but he inferred it. I did not intend to convey this impression. I tried to make clear that the great majority of the accidents under this head were due to misfires and fumes brought about by either using detonators weakened by dampness or through the miner failing to do his work properly.

I am unable to show by figures which of these two causes is the more important, but am of the opinion that it is the latter.

Mr. Williams' point is that a No. 6 detonator is not strong enough to completely detonate the line of cartridges in a drill hole, and proves his point by an actual experiment.

We have always been given to understand by explosive experts that a No. 6 detonator in good order has sufficient strength, and strength to spare to bring about complete detonation of a charge; Mr. Williams states otherwise.

In view of other expert opinion, it is difficult to concur with Mr. Williams, but I am at one with him that by using a No. 8 detonator we have an assured factor of safety, and perfect detonation will always result if the miner does his work properly.

NOTES ON RARE MINERALS IN MADAGASCAR.

By T. P. Watts (Member).

(Printed in Journal, March, 1916.)

DISCUSSION.

The President: I might mention that I have tried to get discussion from some gentlemen who have been to Madagascar, but they all seem very chary of expressing any views. Whether it is that they are tied down to somebody, or do not like saying too much about the country, I do not know.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Ignition of Gases by Impulse Electrical Discharge.—"We are informed by the *Electrician* that a paper on this subject was read recently by Dr. W. M. Thornton before the Royal Society. The ignition of gases by impulse discharge is considered first as a function of sparking distance. It is shown that the shorter the distance the greater the spark, so that the volumes of the least igniting spark are, in a typical case, the same for all spark lengths. Ignition may occur with intense momentary brush discharge, generally with the true disruptive spark. The products of combustion are found to be ionised and to carry a positive charge. The gases examined were mixtures in air of hydrogen, methane, propane and pentane; of ethylene and acetylene; carbon monoxide and cyanogen; coal gas and a mixture of equal volumes of hydrogen and methane. Hydrogen, propane, pentane and carbon monoxide rise gradually in difficulty as the percentage of oxygen is reduced; methane is ignited by the same spark whatever the percentage of gas may be; acetylene and cyanogen have the stepped atomic type of ignition; ethylene is more inflammable in rich mixture. Hydrogen and methane in equal volumes are ignited as methane in type, hydrogen in magnitude. The limits of inflammability of the paraffins are shown to be reached the upper limit when there is twice the volume of combustible gas to that in the mixture for perfect combustion, the lower limit when the volume of oxygen is twice that for perfect combustion less one atom to the molecule. The ignition of coal gas is through methane. Four types of electrical ignition are given, covering all from the most rapid to the slowest rate of discharge from the poles."—*Indian Engineering*, April 29, 1916, p. 255. (J. A. W.)

METALLURGY.

Advancement in the Metallurgy of Zinc.—"With regard to the future of the zinc industry in this country (United States) the principal factor to be taken into account is the disposition that will be made of the Broken Hill ores from Australia after the war. Some of these are now coming to the United States, a part is going to England and the remainder is being stocked in Australia. If

the smelting of these ores in Belgium and Germany is resumed when peace has been declared, the effect upon the American zinc industry will be decidedly adverse.

Normally the United States supplies about 320,000 tons yearly, out of a total production of over 1,000,000 tons; but the proportion is now much larger. Australian ore production represents about 200,000 tons of spelter.

Another factor affecting the future will be the birth of the zinc smelting industry in Russia. That country has large zinc deposits and a smelter has recently been built. It is also reported that new zinc smelters of considerable capacity will be built in England.

From a metallurgical standpoint the most radical development in zinc ore reduction during the past year is evolution on a commercial basis of the new electrolytic process, which avoids smelting altogether. By means of this a refined spelter is produced directly from the roasted ore. The origin of this was the French process tested out on a considerable scale by the Standard Silver Lead Mining Co., Silverton, B.C., and this has since been improved upon.

As stated editorially in the Mining and Engineering World, Oct. 23, several of the larger smelting companies in the west have already inaugurated the process. Among these the Consolidated Mining and Smelting Co. has, throughout 1915, been operating an experimental plant, and is now erecting one to operate on a commercial scale, while the Anaconda Copper Mining Co., after trying out a plant of 1-ton capacity in 24 hours, is building one of large capacity as described in Mining and Engineering World, Dec. 25. At Murray, Utah, the American Smelting and Refining Co. is pursuing a similar course by installing a small plant, to be followed in due course, by one of considerable size. In California an experimental plant is treating Bully Hill ore and in the east chemical works are using the process in handling zinciferous pyrite. The preliminary treatment consists of a dead roast in a special Wedge rotary zinc roaster, the calcines from which are leached with a weak sulphuric acid, viz., about 10° B. Precipitation then follows. Under proper conditions good electrolytic zinc is secured. Success in carrying out this process depends to a large extent upon the roast, particularly where the ores or concentrates treated carry iron; but the roast can be carried to such a point that the calcine tests 0·5% FeO or less, and not more than a third of this enters into the solution. The process will be applied principally to ores which are low grade, or produced in localities where fuel is costly. The principal question is whether fuel will cost more in smelting or in generating the electric current necessary for electrolysis, with the difference, however, that the consumption of the latter is in direct ratio to the zinc-content of the ore or concentrate, while in smelting the quantity of fuel used depends upon the tonnage of ore treated. There are, of course, additional factors entering into the problem, such as elements in the ore, like iron or lime, which affect reduction by either process, but the principal factors are zinc content and the cost of fuel.

The present high price of spelter affords an opportunity, such as operators have never had before, to develop the electrolytic zinc process on a commercial scale. Normally it would not be practicable, except through costly experimenting, as the margin in low grade ore is not sufficient to yield a profit from small experimental plants under the conditions of ordinary years. Any workable process, however, will pay if it can be carried out commercially for six months or a year; and by the end of that time the experience gained will enable it to be successfully maintained. There is every reason, therefore, to look forward with confidence to the future of electrolytic zinc as an increasingly important element of production, and one that will be of particular benefit to western producers."—M. F. Chase.—*Mining and Engineering World*, Jan. 1, 1915, p. 15. (H. R. A.)

Manganese Bronze: Manufactured from Scrap Material.—"1. At the Washington Navy Yard the accumulation of a large quantity of miscellaneous non-ferrous scrap has led to the careful study of the metallurgical problems connected with the successful use of such materials. It is proposed to deal in this paper with the production of manganese bronze ingots alone and to describe the materials used and the foundry practice, so that the experience gained here may be of value to others who are confronted with a similar problem.

2. The accumulation consisted chiefly of skimmings, turnings and trimmings, and obsolete or defective castings of compositions so doubtful that it was not considered advisable to use the materials directly in the production of castings. The yellow-metal scrap only is worth considering, although small amounts of red-metal scrap can be used to obtain the necessary tin. The following materials were available with composition approximately as shown:—

1. Naval Brass: Copper 62%, zinc 37%, tin 1%.
2. Cartridge-case Metal: Copper 68%, zinc 31·6%, nickel 0·4%.
3. Manganese bronze: Copper 59%, zinc 41%.

4. Commercial brass can be used in small quantities but should be avoided, as the lead content is too high.

3 Of these materials (1) naval brass and (2) cartridge-case metal can be and have been used in the manufacture of cast naval brass, but the demand for cast naval brass has not been great enough to warrant holding scrap for use in that alloy alone, so it has been necessary to work out an economical and practical method for manufacturing manganese-bronze ingots.

4. The results of experiments during about six (6) months have shown that it is practicable to make high-grade ingots in an oil-fired Rockwell furnace of about 2 (two) tons capacity. This has been accomplished in spite of the well-known prejudice against open flame furnaces in the manufacture of non ferrous alloys. Oxidation has been reduced to a very small amount by using wood scraps from pattern shop and salt. The bath is protected by the molten salt, and the wood insures a reducing rather than an oxidising atmosphere in the furnace. The zinc losses are lower than is the case when no covering is used.

5. Before undertaking the manufacture of manganese bronze itself a special hardener is made. This hardener is really the secret of the whole process, and although it can be made in any desired proportions it has been found that the following is more satisfactory: 100 lb. copper, 25 lb. mild steel, 25 lb. (80%) ferro-manganese. The ferro-manganese and mild steel are melted and the copper (usually copper wire or other scrap) added as fast as the mixture will take it. Pots should be well stirred in the furnace. The alloy is best made

using pots containing about 250 lb. each, and pouring these into a ladle for casting. Great care should be taken to skim the pots before pouring into the ladle. The hardener can be cast into ingots in green-sand open moulds. After carefully skimming, both in the pot and in the ladle, it will be found that practically all of the carbon has been eliminated, as it separates from the alloy, floats on the top and is removed by the skimming. The resultant alloy will be quite clean and can be readily broken into pieces of any size. The alloy has a characteristic blue-gray colour.

6. This hardener has proved most satisfactory as a means of introducing the desired percentages of iron and manganese. Aluminium and tin are added at the end of the heat, as needed, to make up the desired percentages.

7. In determining the proper amount of scrap for a charge the approximate analysis of scrap on hand must be known. It is necessary to consider the scrap as so many pounds of copper, tin and zinc, and sufficient accuracy will be found if we work out the mixture to contain 57 % copper and add aluminium and tin as may be necessary. Suppose there is on hand a stock of scrap condenser tube containing approximately, 70 % copper, 29 % zinc and 1 % tin. In order to bring the copper content to 57 %, it will be necessary to use only about 78 lb. of scrap per 100 lb. of manganese-bronze. Zinc is added to reduce the percentage of copper.

8. The composition desired is as follows: Copper 57 %, zinc 40 %, iron 1·00 %, manganese 0·75 %, aluminium 0·75 %, tin 0·50 %.

To obtain manganese 0·75 %, about 5 lb. of hardener should be used for every 100 lb.; this gives 2½ lb. copper, so that only enough scrap (condenser tube) should be added to bring copper content to 57 %—that is, 54·5 lb. of copper, which will be obtained from 78 lb. of scrap. This amount of scrap will carry 0·78 % tin. In addition to these, one (1) lb. of aluminium should be added to every 100 lb. of charge as computed. Aluminium, manganese and tin will be burned out to a slight extent.

We now have 78 lb. condenser tube, 5 lb. hardener, 1 lb. aluminium. To this should be added 16 lb. zinc to bring total to 100 lb. This gives a zinc content of 38 %, assuming that there are no losses. About 5 lb. to 8 lb. more should be added to make up for losses, which actually do occur, and the desired composition will finally be obtained.

9. After analysis of the heat the amount of zinc necessary to bring zinc up 41 % can be calculated, and this amount should be added when remelting for pouring into castings. The best bronze, showing the highest tensile strength, is a bronze containing 41 % zinc. If zinc drops to even 38 %, tensile strength is reduced and elongation is increased and a soft bronze obtained.

10. In melting in the oil furnace the most difficult scrap to melt should be charged first, although all but finals may be charged at once. As soon as melted the hardener should be added. In about half an hour charge the remaining scrap (if charge is not made all at same time) and continue the melt. After the heat is well up add zinc, then tin (if necessary) and finally aluminium; stir well and tap. Small ladles are used for pouring the ingots. Ingots are numbered to show the heat and turned into the store awaiting analysis. The cost of the method is high, on account of the labour in pouring and marking ingots, but, counting in furnace loss, labour, fuel and upkeep of furnace it is less than 2 cents per lb., so that scrap worth 7½ cents per lb. can be converted into manganese bronze to cost not over 10 cents per lb.

11. One of our heats gave 82,000 lb. tensile strength 28 % elongation. Quite frequently 75,000 lb. tensile strength and 20 % elongation are obtained in sand castings. If high pouring temperatures are avoided and the metal is poured when it ceases to give off zinc fumes in large volume, excellent values will be obtained so long as the zinc content is kept at 41 %."—Albert J. B. Brooks, *Page's Engineering Weekly*, Oct. 22, 1915, No. 580, Vol. xxvii., p. 290. (J. E. T.)

Heat Treatment for Gear Steel.—"A new heat treatment for steel, whereby such articles as gear wheels or pinions having a hardened surface especially adapted to resist abrasion together with a tough interior adapted to resist strain or shock, is reported to have been invented by Prof. Henry M. Howe, of Columbia University, New York.

The principle of the process is the formation, on the surface of the article, of a steel alloy in the martensitic condition, while the remainder of the steel, its body, or the interior parts, is to exist in the austenitic state. The process is carried out by the casting or forging of a steel having a composition capable of existing in the states of austenite and martensite. This is then treated according to one of the following methods, so as to produce the martensitic conditions in that portion which is to be made wear-resistive and the austenitic condition in the rest of the steel:—(1) By manipulating the temperature of the different parts so as to produce austenite in the one part and martensite in the other. (2) By shifting the metal in part of the object into the martensite state by mechanical deformation. (3) By varying the composition relatively in the two parts so as to make one part so rich in the obstructive elements, manganese, nickel, and carbon, that it may remain as austenite and the wear-resistive part less rich in such obstructive elements so that it may slip into the martensite state under proper treatmentment. (4) By combining these different methods, using each to reinforce the effect of the other. In the case of small pinions the article is quenched from a temperature high enough—well above 1,050° Cent.—to cause the whole of it to remain austenitic and then to reheat the surface or wearing parts to a temperature at which they become martensitic, quenching the whole lest the change goes too far and either too much of the austenite changes to pearlite. The martensite thus produced is disposed for a slight depth on the pinion, and is so supported by the body of austenite into which it merges that its natural tendencies to crack or splinter under shocks or blows are avoided, the whole article being exceedingly durable and capable of long wear."—Prof. H. A. Rowe. *Page's Engineering Weekly*, Oct. 29, 1915, p. 308. (J. E. T.)

Aluminium Dust as a Precipitant. "In our issue of June, 1913, we gave a description of the method of precipitation of silver from cyanide solutions at the Nipissing plant at Cobalt, wherein aluminium dust is employed instead of zinc dust. The aluminium does not replace the silver in the cyanogen compounds, and the reaction is actually effected through the instrumentality of caustic soda, sodium cyanide being regenerated and sodium

aluminate being formed. The process was adopted at Nipissing because it was found that with zinc precipitation the zinc taken into the stock solution formed with the arsenic present in the ore a combination inimical to the dissolution of the silver, the substitution of aluminium for zinc showing an increase in extraction of over 4 oz. silver per ton. Since that time its use has been continued with entirely satisfactory results. In August 1914, C. H. Clevenger presented to the American Institute of Mining Engineers a paper reviewing Mr. Hamilton's original article and criticising certain points in it, to which the latter replied in an article in the *Engineering and Mining Journal* of March 27, 1915. The most important question raised by Mr. Clevenger was: 'Is the aluminium dust process likely to result in a greater final profit than the use of zinc, at plants where zinc in solution is no detriment to a high silver extraction?' This Mr. Hamilton tried to answer in the affirmative, by showing that for a 10 oz. silver ore the additional cost of the process over zinc should not be more than 1 cent per ton, against which should be set a saving in cyanide of greater or less amount. Mr. Clevenger questioned such a saving on the ground that all or most of the zinc-potassium cyanide formed in the zinc process was subsequently regenerated and rendered available for doing work, and, on the contrary, even suggested a loss of cyanide in the aluminium process due to adsorption by precipitated aluminium hydroxide.

Mr. Hamilton, while admitting that a variable amount of the zinc double cyanide formed in the zinc process is usually regenerated later on, denied that anywhere near all of it was so rendered available, the proportion not regenerated forming a measure of the cyanide that might be saved by substituting the aluminium process. He also expressed the opinion that while the adsorption of soluble salts by colloidal substances was an acknowledged fact, yet that the figures at the Nipissing indicated that in practice this tendency was small enough to be negligible, probable because, on account of a high caustic content in the solution, aluminium hydroxide was not precipitated, though there was some precipitation of aluminium as calcium aluminate by the small amount of lime added in the mill.

A full-sized working test of the two processes under similar conditions was the only way to settle the theoretical questions raised, and the test was made at the mill of the Butters Divisadero Company. An account of this test is given by E. M. Hamilton and P. H. Crawford in an article published in the *Mining and Scientific Press* for September 11.

The ore treated at this mill is not one of those in the cyaniding of which zinc in solution is detrimental to extraction, so that the chief advantage to be derived from the introduction of aluminium precipitation lay in the possible saving of cyanide by its use. The assay value is low, being about $2 in gold and 7 oz. in silver, and as 1 oz. of silver combines with alkaline cyanide to the amount of 0·08 lb. KCy or 0·06 lb. NaCy it will be seen that at this mill the conditions are not as favourable as they might be for displaying the capabilities of the process in the economy of cyanide. It was soon found that some modification in the system used at Nipissing would be necessary in order to make it applicable to this ore. The first condition with which the authors had to contend at Divisadero was the acidity of the ore, the lime consumption being 30 lb. of high-grade lime per ton. As stated in Mr. Hamilton's original article, it is essential that lime shall be absent from solution at the time of precipitation, because it reacts with the sodium aluminate and is precipitated in the press as calcium aluminate, yielding at the clean-up a product that it is almost impossible to melt. The cost of substituting caustic soda for lime as neutralizer would have been prohibitive, while the plausible alternative of adding just the right amount of lime to insure its being all neutralized by the time the ore treatment was finished had the drawback that, though neutralized, it would not necessarily all be removed, because calcium sulphate is rather more soluble in water than the hydroxide, and would be equally precipitated by sodium aluminate should it reach the precipitation system. The second condition that made the Nipissing process inapplicable here was the fact that the slime would not settle satisfactorily without an adequate quantity of lime in solution at the close of ore-treatment, whereas, at the Nipissing, as already stated, the final settlement takes place without trouble in the absence of lime. To overcome these difficulties Mr. Hamilton devised the plan of treating the solution before precipitation with soda ash, thus throwing out the lime as carbonate and substituting caustic soda therefor:

$$Ca(OH)_2 + Na_2CO_3 = CaCO_3 + 2NaOH$$
$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4$$

After several months experimenting on a large scale they succeeded in developing this idea into a smoothly working process, which at the time of writing has been running six months and gives entire satisfaction. It makes it possible to use all the lime necessary for neutralizing and settlement, it yields a lime-free solution for precipitation, and incidentally manufactures the caustic soda necessary for that operation. The idea at first was to add the soda ash to the slime charge at the end of agitation, so that the precipitate of calcium carbonate might be filtered out with the gangue avoiding a separate operation; but owing to the slow settlement of the slime, caused by the absence of lime, this plan was abandoned, and it was decided to separate the lime from the clear solution just before precipitation of the gold and silver.

During the period covered by the following data the cyanide used was sodium, 130%, costing 21c. per pound laid down at the mine, the cost of zinc was 7·2c. per pound, of aluminium dust 26c., and of soda ash 2·4c. The aluminium precipitation figures represent averages over three consecutive months, during which the total quantity of ore crushed and cyanided was 28,100 tons. For comparison in zinc precipitation, three recent consecutive months have been taken, the total tonnage being practically the same.

Zinc (Tons treated, 28,200).

	Total. lb.	Per ton. lb.	Total cost. $	Cost per ton. $
Cyanide	62,100	2·21	13,104	0·464
Zinc	29,816	1·05	2,146	0·076
Total			$15,250	$0·540

Aluminium – (Tons treated, 28,100).

	Total. lb.	Per ton. lb.	Total cost $	Cost per ton. $
Cyanide	38,068	1·35	7,994	0·284
Aluminium	7,700	0·27	2,002	0·071
Soda ash	43,013	1·53	1,031	0·036
Total			$11,028	$0·391

Thus, although the cost of aluminium and soda was 3c. more than the cost of zinc, yet the aluminium showed a saving of $1,922, or 15c. per ton. The bullion precipitated during the two periods was for the zinc process 134,628 oz. of fine gold and silver, and for the aluminium 173,660 oz. During the latter period the grade of ore was somewhat higher, and also the percentage of extraction, which accounts for the greater quantity of bullion recovered."—E. M. Hamilton and P. H. Crawford, *Mining and Scientific Press*, Sept. 11, 1915. —*Mining Magazine*, October, 1915, Vol. XIII., No. 4, p. 230. (J. E. T.)

Metallurgical Smoke.—"We have before us a very interesting contribution from the Bureau of mines, Washington, on the subject of Metallurgical Smoke. This forms Bulletin 81, and is presented by Mr. Chas. H. Fulton. Metallurgical smoke is discussed in detail in relation to the three main constituents of smelter smoke, viz., gas, flue dust, and fume. Of these three constituents the gases make up the great bulk, and include as obnoxious constituents carbon-dioxide, carbon-monoxide, sulphur-dioxide, and sulphur trioxide. The flue dust consists of the small particles of the various ores, fluxes, or fuels used in the process which are carried away bodily by the gases. The amount of flue dust depends largely upon the type and design of furnace, blast pressure, or draught employed, fineness of charge, etc. Generally speaking it is similar in composition to the charge of the furnace, modified, however, according to the relative fineness of the various constituents of the charge and the relative specific gravities. The amount may be expressed, for example, in the case of an iron blast furnace, in grammes per cubic foot of gas.

It is very difficult to differentiate between flue dust and fume in many cases. Fume consists of the volatile constituents of the ore charged which pass off in the form of vapour along with the gases and then sublime upon the cooler parts of the flues. This is, of course, a very important factor in the smelting of mercury, zinc, and lead. It may also have an important bearing in copper smelting plants, particularly where the ores are arsenical.

Dealing with the gases in metallurgical smoke, several typical analyses are given and are discussed in detail. Dealing with the influence of sulphur dioxide, which forms, perhaps, the most harmful constituent, it is pointed out that air containing 0·007% of this gas is unbreathable, causing a spasmodic contraction of the air cells of the lungs. The proportion of this gas in metallurgical smoke varies widely. In gases from pyritic copper smelting it constitutes 4½% to 9% of the volume of the smoke issuing from the furnace throat. The gases from the roasting of sulphide copper ores contain about 2% to 2·75%. This question has received considerable attention in the States, and some recent Court decisions have fixed the amount of sulphur dioxide that may be discharged from stacks in certain districts at not more than 0·75%. By way of comparison it is stated that the quantity of sulphur dioxide in the air of cities is as follows :—

London	... ...	0·00039% by volume
Glasgow	... ...	0·00012% „ „
Berlin	... ...	0·00035% „ „

Salt Lake City, Utah, varied from 0·001 to none. The gas may be detected in the air by its smell if present in one part per 150,000, or chemically in one part per million. One part per 500,000, if permanently present, will injure soft resinous woods, especially spruce. This quantity is equivalent to 0·002% by volume, or 0·0071 gm. of sulphur dioxide per cubic metre of air.

The effect on vegetation is first to develop yellow spots on leaves, which gradually extend until the whole leaf becomes seared and yellow and then falls off. Plants in damp climates are much more subject to damage than those in dry climates.

Sulphur trioxide, which is the anhydride of sulphuric acid, may occur originally in the gas or may develop from oxidation of sulphur dioxide. It is, of course, extremely corrosive, but fortunately does not exist in very large proportions.

Turning to the composition of flue dust, some interesting analyses for copper furnaces are tabulated. Most of the metals present exist as a sulphur compound mostly sulphate and sulphide. Some very complete typical fume analyses are also included, and the whole matter is discussed very thoroughly.

The report then proceeds to discuss the recovery of flue dust and fume and methods proposed for rendering smoke harmless. As an example of the loss on this account in a plant at the Great Falls, the dust was estimated to contain 3,775 lb. of copper, 106 oz. of silver, and 0·71 oz. of gold per day. This amount was very considerably reduced by adequate provision for dust recovery. The flue system is, of course, one of the main items to consider in reducing the loss in this way. One of the special systems mentioned consists in the use of a network of steel wires in chambers through which the gas passes, forming a baffle to hold the dust. The wires are periodically shaken and the dust removed. Another system which has proved advantageous is the use of special butterfly dampers, whilst in certain lead smelting plants bag houses have been provided in which the dust and fume is recovered. This latter system consists in filtering the stream of smoke through woollen or cotton bags. The details depend largely upon the particular metal being smelted. In some cases, for example, it is necessary to cool the gases and remove sulphuric acid which would quickly deteriorate the bags. On the other hand, zinc fume can be filtered under almost any conditions, since the sulphur readily combines to form zinc sulphate, which is not harmful. The bags are periodically shaken and the contents renewed.

An entirely different method consists in the electric precipitation of fume. Direct or alternating current may be used. If the former, the gas is ionised, and the ions collect upon the solid particles, thereby charging them electrically. The charged particles are then driven to the collecting electrodes. This is more efficient than alternating current in the case of large volumes of rapidly moving gases. The action of the alternating current consists for the most part in the agglomeration of the suspended particles into larger aggregates and a consequent more rapid settling under the influence of gravity.

Much effort has been directed towards finding methods for the removal of sulphur dioxide from the furnace gas, but up to the present with little promise of success. One method described consists of passing the smoke stream upwards through a tower to meet a stream of cold water. The sulphurous water would then be concentrated into

sulphuric acid. The Hall process is described, in which sulphur is removed directly from the sulphide ores in the form of elemental sulphur, preventing the formation of sulphur-dioxide. Other processes are described, but none have been particularly satisfactory." T. SWINDEN, Bureau of Mines, Washington, *Bulletin* 81, C. H. Fulton.—*The Engineering Review*, Sept. 15, 1915, Vol 29, No. 3, p. 80. (J. E. T.)

ZINC IN THE CYANIDE MILL.—"The high price of zinc, in both shaving and dust form, has been causing cyanide operators considerable anxiety since the breaking out of the European war. Since it is fairly well known that a great deal more zinc is used than is theoretically required for precipitation of gold and silver contained in solution, some method of economizing it has been made the object of a great deal of study. Some have attempted to accomplish the same result by recovering the zinc from precipitate. This article gives the result of some experiments that I carried out at the McIntyre-Porcupine mill in the Porcupine gold district of Ontario. While it does not point toward a remedy for the difficulties encountered, it does show what is actually taking place in the mill. I refrain from comment or suggestion, but merely put these facts in the form of tables and as briefly as possible, hoping that they may suggest some scheme of improvement to those who may be considering the subject."

Experimental Data on Zinc in Cyanide Solutions.

Period of experimenting, June 1st to 7th, 1915, inc.

	Tons.
Ore treated	2,010
Solution used in Hardinge ball mills	2,010
Solution used in closed circuits, tube mills and classifiers (overflow being 71·36% moisture)	5,013
Solution used in agitator	5,013
Pregnant solution sent to precipitation	1,703
Solution sent to decantation units: 1,098 tons solution, primary thickener underflow, 35% moisture; 4,703 tons barren solution	5,801
Solution discharged with tailings (35·7% moisture)	729
Zinc dust used (dried at 212° F.)	984
Precipitate (dried at 212° F) recovered	735

Samples of solution were taken every 24 hours in proportion to the quantities treated, and at the end of seven days combined and mixed thoroughly. All zinc determinations were made by potassium ferrocyanide titration with uranium nitrate as indicator.

Analysis of Solutions Used in Experiments.

	lb.
Storage	0·213
Hardinge discharge solutions	0·157
Classifier overflow	0·157
No. 2 Agitator	0·117
Pregnant solution	0·158
Barren solution	0·308
Au precipitated from each ton of solution	83·658
Or 832·31 oz. or	57·000
Ag precipitated from pregnant solution	17·393
Theoretical amount of zinc necessary to precipitate total gold 19 lb	
Theoretical amount of zinc necessary to precipitate total silver 6·178	25·178
Commercial zinc dust used contains 95·04	zinc
Zinc content in 984 lb. used	935·19

Consumption of Zinc in Various Departments.

1. According to Solution Analysis:

Barren solution shows an increase of 0·150 lb. per ton of solution. As 4,703 tons was precipitated, the amount of zinc dissolved in press was	705	
Zinc wasted through secondary reactions	75·4%	

2. According to Precipitate Analysis:

Precipitates contain 32·5% total zinc. As 735 lb. was recovered, the amount of zinc left in press ...	239·38	
Zinc dissolved in press (935·19 - 239·38)	695·81	
Zinc wasted through secondary reactions	74·4%	

3. According to Zinc Precipitated from Solutions in Mill Operations:

	lb.	%
Zinc precipitated in Hardinge mills (2020×0·231) - (2040×0·157) ...	150·96	16·14
Classifier overflows, 5,013 tons, composed of 2,040 tons Hardinge mill discharge solution at 0·157 and 2,973 tons of storage at 0·231, equals 5,013 tons at 0·2009 lb. (5,013×0·2009) - (5,013×0·165) ...	179·96	19·24
Zinc precipitated in agitators (5,013×0·165) - (5,013×0·117) ...	90·234	9·65
Zinc precipitated in decantation* system: 4,703 tons barren solution at 0·308 lb. and 1,098 tons at 0·158 lb. 5,801 at 0·2796 (5,801×0·2796)×(5,801×0·231) ...	...281·928	30·15
	703·082	75·18

*Includes amount of zinc (91·12 lb.) discharge as soluble zinc salt.

A. DORFMANN.—*Engineering and Mining Journal*, Oct. 23, 1915, p. 680. (H. A. W.)

MINING.

SCIENTIFIC HUMIDIFICATION OF MINES.—"Writing in *Coal Age* Mr. J. J. Lyle says:—So far as I am aware no mine has yet been humidified on a scientific basis, or, in fact, in any manner that has been successful. Of those installations regarding which I have accurate knowledge, a few fundamental laws necessary for success have been neglected.

The temperature of a mine is fairly constant, varying with the seasons, so that in winter it is a few degrees cooler than in summer, with but slight fluctuations from day to day or even from week to week.

Generally a mine has sufficient humidity during the summer months, because its temperature is much below that of the outside atmosphere. In winter, however, when the outside air carries but little water vapour and the mine temperature is much higher, the relative humidity is necessarily low.

In order that the air used for ventilation may be able to carry sufficient water vapour to humidify the mine, it must be heated, either by its contact with the mine walls or artificially. The temperature to which it is heated must be at least that of the dew point desired.

Take a mine, for instance, with a temperature of, say, 60° Fahr., where a humidity of 80% is desired with the outside air at 20° Fahr. and containing 1 gr. of vapour per cubic foot. Air

at 60° Fahr. and saturated carries 5·75 gr. per cubic foot. With 80% humidity, therefore, each cubic foot would have to contain 80% of 5·75, or 4·6 gr. Now saturated air at 50° Fahr. will just hold 4·6 gr. per cubic foot. It is evident from this that it is necessary that the air should not be less than 54° Fahr. when the additional water vapour is added. If at any lower temperature it could not possibly hold the required amount of vapour to give the 80% humidity.

It must be borne in mind that water cannot be evaporated unless heat is added; or to state it in another way, if water is caused to evaporate in the presence of air without heat being applied, the air will be cooled 8° Fahr. for every grain of water evaporated per cubic foot.

This being true, if the air entering the mine at zero is raised to a temperature of 54°, the only way the necessary moisture can be added would be by blowing steam directly into it. It is well known that the introduction of steam directly into the air current without the production of fog is a most difficult matter, if it may ever be said to be successfully accomplished.

In the case under discussion, with the incoming air containing 1 gr. of vapour per cubic foot it would be necessary to add 3·6 gr. per cubic foot, which if done without adding heat, as explained, would cool the air to approximately 28° Fahr. To counteract this cooling effect we could heat the air to 82° Fahr. and then bring it in contact with atomised water sprays and add the 3·6 gr. per cubic foot, and as a result of the evaporation it would be cooled to 54°. At this temperature of 54° the air would be saturated, but as it travelled through the mine and its temperature was raised to 60° the humidity would be lowered to 80%. Such a system properly installed would give good results without any fog or other objectionable results.

A simpler and more practical method is to first pass the air used for ventilating the mine through a humidifier, where its temperature is raised and the necessary amount of moisture is added simultaneously. Such a machine is now being adopted in textile mills for cooling and humidifying the air.

Here the air passes through a dense and finely divided water spray, where it is saturated. The water used is circulated by means of a centrifugal pump and is heated by the introduction of exhaust steam. Eliminator-plates at the outlets are used to separate the surplus water from the air, so that the air is left perfectly free of any fog or other small particles of water that have not been evaporated. Because of the recirculation of the spray water employed by this machine, the amount of water required is small and is equal to that evaporated. This would never exceed one gallon for each 12,000 cub. ft. of air used.

The exhaust steam required to heat the spray water (which in turn would heat the air to 54° Fahr.) would be 1½ lb. per 1,000 cub ft. of air handled. Of course, with a large ventilating fan this amounts to considerable steam, but in most mines there is sufficient exhaust going to waste to fulfil the requirements.

To summarize: First, it is necessary that the moisture for mine humidification be injected into the air after its temperature has been raised to approximately that of the mine. Secondly, this moisture must be added in such a manner as not to cool the air below the temperature required to carry the desired vapour. Thirdly, the injection of steam into the air at or near the entrance of the mine is sure to produce fog and cannot add the amount of vapour required owing to the temperature of the air being too low to carry it. Fourthly, the humidity of a mine can be maintained uniformly at a desired point by a properly designed humidifier, with heated spray water, located at the air entrance to the mine."—J. J. Lyle, *Coal Age*, *Cassier's Engineering Monthly*, Feb., 1916, p. 121. (J. E. T.)

The Tungsten Miner's Unusual Opportunity. —"Some months ago it was the remarkable spurt in the price of zinc which concerned most intimately the expectations of the western miner. Spelter was the thing, and every man or corporation owning a property with zinc ore in it was figuring how best to take advantage of the war quotations on spelter. Of course, there was some incidental interest in copper and other metals, but zinc was the spectacle. Now it is tungsten which occupies the spot-light of spectacular performances, because of its extensive use in the manufacture of lathe steel. The virtues of this unfortunate metal are just now beginning to appear to the makers of war materials, and the miners of the ore are enjoying the feelings of the Joplin operators who for months have been selling their blende at top prices. The price of tungsten concentrates has soared to a point where it is no longer possible to be sure what it is. Nederland, Colorado, is the principal tungsten district in America, and the boom in that camp is described by visitors as something very unusual."—*The Mining American*, Oct. 30, 1915, p. 4. (J. W.)

MISCELLANEOUS.

Old Car Wheels for Sheaves. —"Old car wheels make excellent sheaves and carriers for haulage ropes, according to Edward Manion, superintendent of the Wasp No. 2 property, Flat-iron, S.D. At this property two old wheels are bolted together with three bolts, with the flanges on the outside. The old axles may be used for shafts. When used as sheaves the shaft is provided with a single collar at the bottom and extends downward into the foundation timber usually a 2×8 and upward into a wooden support. When used as carriers they give equally satisfactory service.

The chilled treads of the wheels form ideal surfaces for the rope to travel on and are much harder and more durable than the carriers and sheaves ordinarily sold for this purpose. As any mine has at all times a more or less liberal supply of old car wheels and axles, it may never want for carriers and sheaves if this plan is followed."—Jesse Simmons. *Engineering and Mining Journal*, Oct. 16, 1915, p. 639. (H. A. W.)

Rust Prevention and the Painting of Iron. —"Experiments conducted by Liebreich and Spitzer, German scientists, to determine the effect of the number of coats of paint on the rate of rusting of iron have recently been published. They polished four iron plates and painted them—the first with one coat, the second with two, the third with three, and the fourth with four. The plates were then exposed to steam for one day. The paint was dissolved off and the following distinct results noted:—

The iron under the single coat was bright all over, that under two coats was partly rusted, that under three coats rusted still more, while the iron under the four coats was covered with rust. This experiment shows that several coats of paint do not protected as well as one coat.

The explanation of this is that subsequent coats of paint or varnish tend to dissolve part of the previous coat. This has the effect of loosening the previous coat and making it porous, the porosity increasing with the number of coats. Air and moisture penetrate the pores and the iron below is rusted.

Where rust is feared, the coat of paint or varnish is usually made too thick, and the application of a second and third coat naturally does no good. The proper procedure is to remove all old paint by using the sand blast, solvents, etc., and then give the clean iron surface a single coat of the paint or varnish."—LIEBREICH and SPITZER.—*Page's Engineering Weekly*, Dec. 10, 1915, No. 587, Vol. XXVII., p. 404. (J. E. T.)

DESIGN AND APPLICATION OF BALL-BEARINGS.—"The author advocates the use of ball-bearings for tube-mills, and insists that these should be of the self-aligning type with a double row of balls to allow for any sagging of the mill when loaded. He gives an instance of a tube-mill on the Rand having been fitted with roller-bearings and claims that these bearings reduced the total power consumption by about 30%, at the beginning, gradually declining to about 10% at the end of twelve months, when the bearings gave out. This he attributes to the deflection of the mill under load, and points out that this falling off in efficiency is not experienced with self-aligning bearings."—H. I. MOYSEY, A.M.I.E.E., Proceedings, Manchester Association of Engineers.—*Page's Engineering Weekly*, Dec. 10, 1915, No. 587, Vol. XXVII., p. 398. (J. E. T.)

Abstracts of Patent Applications.

(C.) 117/15. Samuel Thomas Tregaskis. Improvements in emergency signalling arrangements for rope hoists. 25.5.15.

This application refers to means for indicating when a skip is accidentally arrested in the head gear tip or elsewhere when it should be moving normally, the idea being to indicate by a visual or oral signal when there is any difference or relative movement between the head gear sheaves in the case under consideration.

(C.) 22/16. E. H. Woodman and J. E. Stone. Improvements in conveyor belt idlers, haulage rope supporting rollers and the like. 29.1.16.

This application is for a particular construction of self-oiling idler, pulley or rope roller, it being specially adapted to the self-oiling type of idler or roller on which the applicants have already been granted a patent.

(C.) 452/15. James Welshir (1), Hugh McLean (2). Improvements in rock drill cradles. 5.7.15.

This application relates to a combination with the shell and caps of a cradle used with rock drills, of wearing plates so formed as to interlock with the shell and caps of the guides, and to present a renewable wearing surface longitudinally, thus prolonging the life of the cylinder slides and cradle guides.

(C.) 166/15. Seymour Edgar Hubbard (1), Alfred Edward Barker (2). Improvements in connection with stope and like conveyors. 20.7.15.

This application has reference to a stope conveyor in which two endless chains are used to carry between them short sections of belt secured to bars connecting the two chains. The chains are carried on rollers hanging from the roof and anchored to the footwall, while the conveyor is driven by a handle on one of the rollers or may be pushed along by hand.

(C.) 299/15. Reginald Scarbrow. An improved process for the treatment of crushed auriferous ore, known as slime. 7.12.15.

This application relates to the improvements in the treatment of slime, which consists in using a cone-shaped vat, fitted with overflow rim, in which the slime charge is circulated. At the same time fresh barren solution is added and the excess thus formed overflows to extractor boxes and is subsequently reused. When the values are thus recovered a decanter removes the balance of the solution after settling and the remaining pulp is discharged.

(C.) 68/16. George Constantinesco (1), Walter Haddon (2). Improvements in power transmission. 14.3.16.

This application refers to apparatus used in connection with the transmission of power by means of mechanical wave motion in a solid column of liquid, as described in the application for a patent No. 12/16, and consists in connecting a receiver completely filled with liquid in the pipe circuit, so that the motors operated by the generator may not cause a dissipation of energy closed off from the circuit. Also the introduction of receivers, as above described, which would have the same effect as a metallic spring.

(C.) 78/16. Arthur George Whittome (1), John Herbert Veasey (2). Improvements in water blasts. 21.3.16.

This application refers to a water blast apparatus provided with a controlling key, which can only be removed when the cock controlling the air and water blast is turned on, a means for preventing the water, even if delivered under pressure, from forcing its way into the air pipes, and also a means for inserting various sized orifices for controlling the relative proportion of air to water. The whole apparatus is enclosed in a strong metal shield to prevent damage to the more delicate parts.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

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ERRATUM.

December Journal: Page 121. The first notice under Chemistry:

For "Determination of Zinc as a Double Sulpho

Read "Silicidised Carbon. Silfrax. Silfrax is"

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(COMPILED BY THE SECRETARY MR FRED ROWLAND F.C.I.S.)

These portions in Italics refer to Original Notes and Papers read before the Society.

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